THE MINERS' POCKET-BOOK.

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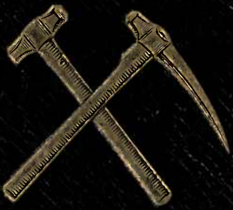

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THE

MINERS' POCKET-BOOK

A REFERENCE BOOK
FOR ENGINEERS AND OTHERS ENGAGED IN
METALLIFEROUS MINING

BY

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FIFTH EDITION, ENTIRELY REWRITTEN

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INTRODUCTION TO FIFTH EDITION.

THE fact that four long editions of this little volume have been exhausted, is in itself evidence that a measure of success has attended the author's effort to cater for the needs of his fellow-workers in the domain of metalliferous mining.

It may truly be said that professional men of no other class are so often confronted by problems and difficulties, and have at the same time so little opportunity of accumulating books of reference or of gaining access to libraries. Hence a portable yet comprehensive means of refreshing the memory with facts, figures and formulæ is very acceptable.

It is not pretended that the whole education of a Mining Engineer can be achieved by reading a single book, or indeed any number of books; but whatever his schooling and subsequent practical experience, no man can afford to neglect keeping himself posted in what others are doing.

The ultimate aim of all mining is financial gain, so that economic or commercial considerations are of primary importance. Lessening of costs means extension of the industry—a consummation which must appeal strongly to us all. This has been kept prominently in view throughout the following pages.

By entirely re-writing the present edition, it is hoped that its general utility has been increased. The deletion of matter possessing but limited interest has made room for a material expansion of the more important subjects connected with underground operations. As before, the published and unpublished results of others work has been fully utilised, and the author takes this opportunity of expressing his indebtedness to the numerous writers and publications quoted.

Comments and criticisms will be gladly accepted for future editions, and may be addressed to the care of the publishers.

C. G. WARNFORD-LOCK.

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THE MINERS' POCKET-BOOK.

POWER.

Units.—The unit of work, i.e. expended power, is the moment or effect of 1 lb. through a distance of 1 ft., and is termed 1 foot-pound (ft.-lb.). Its metric equivalent is the kilogrammetre, or pressure of 1 kg. through a distance of 1 m., which = 7·233 ft.-lb.; so that 1 ft.-lb. = ·138 kg.-m.

Horse-power.—This is the usual measure of work performed, and 1 h.p. is computed to be equivalent to the raising of 33,000 lb. 1 ft. high per minute, or 550 lb. per second. But this is merely theoretical, as a horse can exert that force for only 6 hr. per day; hence 1 work h.p. = the power of 4·5 horses at 3 miles per hr. The metric equivalent of this is 1 cheval or cheval-vapeur = 75 kg.-m. of work per sec., which (× 7·233) = 542·5 ft.-lb., or 1·37 % less than 1 h.p. Then

1 h.p. = 1·013 chev.-vap.; 1 chev.-vap. = ·986 h.p.
1 cub. ft. per h.p. = ·028 cub. m. per chev.-vap.
1 lb. per h.p. = ·447 kg.-m. per chev.-vap.
1 cub. m. per chev.-vap. = 35·084 cub. ft. per h.p.

MAN POWER.

Theoretical.—Various authorities fix the value of man power as follows: (*a*) Equal to raising 70 lb. 1 ft. high in 1 sec. for 10 hr. per diem = 4200 ft.-lb. per min. (Haswell). (*b*) Exerts a force of 30 lb. for 10 hr. a day, with a velocity of 2·5 ft. per sec. = 4500 ft.-lb. per min. (*c*) Ordinary labourer at ordinary work = 3762 ft.-lb. per min. (Smeaton).

Walking.—A man can traverse 12·5 times the distance horizontally that he can vertically.

Walking to and fro on a rocking beam, assuming his weight at 12 stone (168 lb.), he can produce an effect of about 4,000,000 ft.-lb. in 10 hr. per day = 6666 ft.-lb. per min.

He can average, without load, on level ground, 3·7 miles per hour for 8½ hr. = 31·45 miles a day. [The average Chinese coolie can cover 30 miles a day with a load of over 100 lb. continuously for months!] Ascending slight elevations, unloaded, at 5 ft. per sec. for 8 hr. a day = 4290 ft.-lb. per min.

Carrying.—On short distances, returning unloaded, he can carry—(*a*) 135 lb. for 7 miles a day; (*b*) 111 lb. for 11 miles; (*c*) at 1·75 ft. per sec. for 6 hr. a day = 14,700 ft.-lb. per min. (Morin); (*d*) at 2·5 ft. per sec. for 7 hr. a day, always loaded = 13,200 ft.-lb. per min. (Morin). (*e*) carrying up slight elevation, returning unloaded, at ·2 ft. per sec., for 6 hr. a day = 1680 ft.-lb. per min. (Morin). [The Chinese porters carrying tea and salt traverse 10 miles a day with a 200-lb. load, for weeks on end, much of the journey being at 7000 ft. elevation.]

Pulling.—(*a*) Hauling a boat in a canal, a man can transport 55 t. for a distance of 7 miles in a 10-hr. day; (*b*) at a velocity of 2 ft. per sec., for 8 hr. a day = 3120 ft.-lb. per min.

Lifting.—(*a*) Lifting with the hands, at ·5 ft. per sec., for 6 hr. a day = 1320 ft.-lb. per min.; (*b*) pulling and pushing alternately in a vertical direction, at 2·5 ft. per sec., for 10 hr. a day = 1950 ft.-lb. per min.; (*c*) by single pulley, he can lift 36 lb. with a velocity of ·8 ft. per sec., for 10 hr. a day; (*d*) his maximum power at a crane, for constant work, is 15 lb., which, at a velocity of 3·66 ft. per sec. = 3300 ft.-lb. per min.

Pile-driving.—Making one blow every 4 min., at a velocity of 4·5 ft. per sec., for 10 hr., he exerts a force of 16 lb.

Pumping.—(*a*) During 10 hr., with an efficient pump, a man can raise 1500 cub. ft. of water 1 ft. high; (*b*) labourers pumping water 4 ft. high = 3904 ft.-lb. per min.

Jacking.—The power a man can exert on a screw-jack of 11 in. radius continuously is 15 lb., intermittently 25 lb., average 20 lb.; (*b*) 11·5 lb. easily, 17·3 lb. with difficulty, 27·6 lb. extreme limit (Field); (*c*) 13 lb. at a velocity of 5·33 ft. per sec., for 8 hr. a day (Walker); (*d*) at 2·5 ft. per sec., for 8 hr. a day = 2790 ft.-lb. per min. (Morin).

Windlassing.—Two men working at a windlass, at right angles to each other, can raise 70 lb. more easily than one man can 30 lb.

A depth of 100 ft. may be considered the extreme limit for economic windlassing.

The effective power of a man windlassing, within proper depth limits = 2500 ft.-lb. per min.

Shovelling.—(*a*) Shovelling earth 5 ft. high, at 1·33 ft. per sec. = 480 ft.-lb. per min.; (*b*) shovelling 13 ft. horizontally, at 2·25 ft. per sec. = 810 ft.-lb. per min. (Morin); (*c*) shovelling compact gravel from a dump into a sluice-box for 10 hr. a day (Klondike) for average man = 4½–5½ cub. yd., or, say ½ cub. yd. per hr.; (*d*) Kaffirs shovelling blue ground (De Beers) load 30–60 (ordinarily 30–40) trucks, each containing over 20 cub. ft., per 8 hr. shift = 75 cub. ft., or 2·75 cub. yd., or about 5 t. per hr., for which they are paid 1½*d*. per truck; (*e*) unloading iron ore (Lake Erie), 8 men in the hold, loading 1 t. buckets, working 10 hr. a day, handle 5–6 t. per hr. each, and often reach 6–7 and even 8 t.; they receive 6½–9*d*. per t. (Scott); (*f*) The ordinary computation per 10 hr. shift in U.S. is 14, 16, 18, or 20 cub. yd. per man, according as the ground has been previously loosened by plough or by pick, the handiness of the shovel, the surface shovelled on, and the lift for loading (Gillette); (*g*) Shovelling alluvial wash containing big boulders, $2\frac{3}{4}$ cub. yd. per 10 hr. man; wash 2 ft. deep, lift 9 ft., $3\frac{1}{2}$ cub. yd.; wash $4\frac{1}{2}$ ft., lift 6 ft., 5 cub. yd.; wash 3 ft., lift 5 ft., $7\frac{1}{2}$ cub. yd. (Purington).

Digging and Shovelling.—(*a*) Digging and loading into barrow, per 10 hr. day, good average man (England), 8–10 cub. yd. earth, 6 cub. yd. firm gravel or tough clay, 3–5 cub. yd. picking ground (Fairley); (*b*) Chinese labour in Malayan tin mines, 2 men, 1 filling and 1 emptying baskets, with about 60 ft. carry, can excavate about 10 cub. yd. per diem = 5 cub. yd. per man under best conditions. Ordinarily, the figures are 390–520 cub. yd. per man per ann. = (at 300 days) 1·3–1·73 cub. yd. per man per diem. (*c*) Indian coolies in Malayan tin mines, using hoe and basket, carrying same distance, average about 2 cub. yd.

Picking and Shovelling.—(*a*) Picking alone in U.S. (wages $7\frac{1}{2}d.$ per hr.) varies from $1d.$ to $6d.$ per cub. yd., the larger figure being for very tough clay or cemented gravel; average ground is 2–$3d.$ = $2\frac{1}{2}$–$3\frac{1}{2}$ cub. yd. per hr.; (*b*) Picking and shovelling combined (loading into wagons) costs in U.S. 6–$21d.$ per cub. yd. (wages $7\frac{1}{2}d.$ per hr.), according to ground as in (*a*) = $\frac{1}{3}$ – $1\frac{1}{4}$ cub. yd. per hr. (Gillette).

Wheeling.—(*a*) On short distances a man can barrow 150 lb. 10 miles in a day; (*b*) wheeling a loaded barrow up an incline of 1 in 12, at a velocity of ·625 ft. per sec. (say $\frac{1}{2}$ mile per hr.) = 4950 ft.-lb. per min.; (*c*) wheeling a load and returning empty, on the flat, at 1 ft. per sec. (say $\frac{2}{3}$ mile per hr.) = 7920 ft.-lb. per min. (Morin); (*d*) in ordinary contractors' work in the U.S. a man's max. capacity is 15 miles per 10 hr. with 250 lb. (say ·1 cub. yd.) load on level—on grades, 150 lb. load is enough; including time lost in tipping and changing barrows, with a carry of 100 ft., a man can wheel per hr. about 30 loads, or, say 2–$2\frac{1}{2}$ cub. yd., according to the grade. (Gillette.)

Drilling.—(*a*) Rand custom is 4 holes, $5\frac{1}{2}$ ft. deep, per machine per diem.

Labour Economics.—It is surely hardly necessary to insist on the advantages of contract work over day's pay. It is the only proper system, because it is the only fair one either to the employer or to the employed. Even in pre-Union days, when many men honestly tried to do a full day's work for a day's wage, it was not truly equitable to pay the same rate to the efficient and industrious as to the inefficient and lazy—either one was underpaid or the other was overpaid; while the refined arts of modern trades unionism and the cultivated loafing which now prevails among all kinds of workmen, have made any daily wage ridiculously unfair to the employer. A good deal of experience and judgment is necessary in fixing contract rates, to ensure that men may not earn an unduly high pay for limited labour; but it must not be assumed that because labourers under contract earn more money per man per diem than they did under a daily wage, they are necessarily doing more costly work—the converse will always be the case, unless extremely poor care has been exercised in determining the rate. To begin with, it tends to weed out the inferior workman, because it directly encourages the interest and attention and mental qualities of the operator; the poor worker is stimulated to improve himself, or is shamed away from the job on comparing his low earnings with others'; and as the efficiency of the worker is increased, the numbers employed may be decreased, and one naturally lets out the inferior man. This improvement in the work effects a further economy by lessening the need for supervision, and again another in the wear and tear on plant and tools, and the diminished waste of supplies. A single example may be quoted to illustrate the argument. It refers to the Center Star and War Eagle Mines, Rossland, B.C., and is based on about 2500 ft. of driving and sinking, and over 60,000 t. of ore stoped (Davis, E. & M. Jl., 29/6/01).

	Contract.	Daily Wage.
	Cost per ft.	Cost per ft.
Development—wages	25*s*. 2*d*.	34*s*. 10*d*.
,, explosives ..	11 5½	11 7
,, total	36*s*. 7½*d*.	46*s*. 5*d*.
	Cost per t.	Cost per t.
Stoping—wages	18·85*d*.	37·50*d*.
,, explosives	5·00	5·75
,, total	23·85*d*.	43·25*d*.
Advance per month—driving	97·5 ft.	50·8 ft.
,, ,, sinking	58 ft.	27·2 ft.
Average daily wages earned	17*s*. 2*d*.	14*s*. 7*d*.

Another very important consideration embodies such matters as the duration of working hours, overtime, night work, and Sunday work. Where trades-unionism does not make it impossible, it will be found, in almost all cases, that a 10-hr. shift, with an allowance of even 1 hr. rest in the middle, will be much more economical than an 8-hr. shift with only 20 min. interval. It means 2 shifts only instead of 3 per 24 hr., thus lessening supervision costs, and diminishing the time lost (and paid for) in men travelling to and from the face. All overtime is fundamentally and especially poor economy: the work is always inferior, and the wages paid for it are grossly in excess. Night work should always be avoided as much as possible: men cannot and will not work as well at night as in day time; their energy is actually impaired, for no man is able to recuperate in the same degree by day sleep as by night sleep. Tests have been actually made of pitting a gang of superior men on night shift against a gang of inferior men on day, and always with the result that the latter's work was more satisfactory to the employer. Automatic machinery, will, of course, produce as much by night as by day, and there is distinct gain in uninterrupted running up to a certain point, while the labour of attendance is not so exacting but that it may be done with reasonable efficiency. Yet the experience of any mine manager will be that machinery breakdowns in mine and mill, and accidents of all kinds, occur most often at night, and especially during the first three hours after midnight, when the human system is physiologically weakest. Sunday labour should always be restricted to the smallest limits. Not only is every man who puts in 6 days' real work all the better for a rest on Sundays, but it has been proved beyond cavil with railway locomotives that those running 6 days a week only outlasted by years those on 7-day duty, costing far less for repairs, doing better work, and giving a much higher return for the capital invested in them.

Types and Efficiencies.—While the qualities of labour of different racial types vary as greatly as the nationalities themselves, it must never be forgotten that the useful effect of any labour is almost as much dependent upon the manner in which it is controlled as upon its own inherent merits and demerits; and before discussing the traits of some of the principal classes of human labourer, a paragraph may be devoted to insisting on this very material fact. The real efficiency of a shift-boss or other foreman is to be reckoned far less by his own skill and knowledge of the work than by his ability to get the best possible results out of the men he controls. It should not be necessary to insist on this, but in one's own experience—and it must be the experience of most mining engineers who have managed enterprises in foreign countries—there have been so many and such glaring instances of mishandling of the local labour supply, to the exceeding detriment of the undertaking, by miners placed in immediate charge of gangs (excellent workmen though they were themselves), that special emphasis must be laid on it. The average school-of-mines student, being of a class superior in breeding and education, is better equipped for the job than the practical miner, but even he, unfortunately, gets no training in that direction, and often does not think it worth while to learn from his neighbour when he can. Yet the mines which have established the best records for economical and good work can invariably trace much of their success to careful attention to this question. Thorough capability in handling men is much more essential than high technical qualifications to the junior members of the mine staff. During post-graduate courses, students should be careful to cultivate the art most assiduously. The labour troubles of the Rand and Rhodesia have been largely due to ignorance and carelessness and worse in the management of both the Kaffir and the Chinaman; no man, though he be black or yellow, will tamely submit to the senseless brutality of the average Cornish and Australian miner who is placed over him.

White mine labour is familiar to most of us, and does not need much description. For low-grade products, especially when in superficial deposits, the Staffordshire and Cleveland iron miners demand preference, followed by the Welsh quarry men. The Cornishman shines at timbering, but is very conservative and unadaptable to strange conditions. The Basque of the Pyrenees is a born artist at hand-drilling, even in the hardest ground; and the N. Italian is a very good all-round man. The American understands hard work, but commands huge pay, and the criminal records of the Western unions are a stain on civilisation. The mining laws of the Australian colonies are accountable for the average poor quality of the native-born mine labour, and for the insecurity of mining capital.

In India, the best underground labour is furnished by the Moplahs, a fish-eating race from the W. coast. The Tamil, though much employed in excavating and other surface work, both in India and Malaya, is never a hard-ground man. Burma and the Shan States, as well as Malaya (both insular and peninsular), really depend on the Chinaman. The cost of Moplah labour is about $\frac{1}{20}$ that of English labour (Smyth). The Tamil's wages range commonly between 10*d.* and 1*s.* 3*d.* a shift. In some of the Malayan tin gravel mines the Tamil is distinctly superior to the Chinaman, because of his habit of carrying a load always on his head; he is thus able to discharge into a full-sized (2 cub. yd.) truck without the need of any staging or plank, and this is of considerable advantage. Also he tolerates scorching sun-heat better than any other race; but he lacks stamina, and is often a drunkard.

The Korean would be quite useful if he would emigrate. He is, perhaps, slightly less intelligent than a Chinaman, but he is a big strong fellow, and is less conservative and superstitious. Korean coolies earn about 7½*d.* a day; miners, 8*d.*–1*s.*; carpenters and blacksmiths, 1*s.*–1*s.* 3*d.* Koreans become very fair engine drivers, whereas Chinese scarcely ever do.

Japanese may easily be over-rated. Though intelligent, industrious, strong and enduring, they are very quarrelsome and difficult to manage, and have a very high opinion of themselves which one does not always share. They are prone to strikes and lawlessness. In Korea, Japanese fill most of the mechanics' places, and earn somewhat higher wages than either Koreans or Chinese, reaching as high as 3*s.* a day, and occasionally more. In Japan, at the principal gold mines, in 1903, wages ranged as follows (Weigall):

Women (ore-picking), 5*d.*; boys, 6*d.*; mill and cyanide hands and smiths' strikers, 7½*d.*; truckers and assistant timbermen, 10*d.*; miners and blacksmiths, 1*s.*; timbermen, 1*s.* 3*d.*

The Kaffir, viewed as a source of human energy, stands high in the scale of labouring people; but he is deficient in intelligence, and lacks incentive to work. Moreover, relatively to his wants, he is ridiculously overpaid, and hence does not feel the necessity for remaining in constant employ, and does not become as efficient as he might easily do. His muscular capacity is unrivalled, but it is often misdirected; and, because his pay is small, there is a tendency to waste his own labour, and that of his highly paid white overseer, by employing unduly large numbers. The average cost per shift of Kaffir labour—including feeding and compound expenses—is about 2*s.* 7*d.* The Kaffir makes a very fair driller on down holes with a single hammer; he can be crowded much closer than white men in sinking, and he becomes in time as good as his instructor in running a machine drill; but he is hopeless at an "upper," and could never be trusted to do the simplest bit of timbering.

For real hard continuous labour of all kinds the Chinaman stands absolutely supreme. He is the mule among the nations—capable of the roughest work, the hardest tasks under the most trying conditions; tolerant of every kind of weather and ill-usage; eating little and drinking less; stubborn and callous; unlovable and useful in the highest degree. For the drudgery of mining, the hard, uninteresting, monotonous, never-ending toil with hammer and drill, pick and shovel, or hoe and basket, there is only one race worth considering in the heat of the tropics, and that is the patient, industrious Chinaman. Throughout Malaya, which produces nearly three-fourths of all the tin raised in the world, the Chinaman is the backbone of this industry. Of the 300,000 or so there employed in tin-mining, about 150,000 are engaged in open gravel pits, 25,000 in ground-sluicing, and 25,000 below ground.

The ingenuity and general cleverness of the Chinese coolie in overcoming mechanical difficulties are up to a certain point, simply marvellous. As compared with the average European peasant, the Chinese coolie is a first-class engineer. In practical elementary metallurgy, I would be inclined to award the palm to the Mexican over all other peoples, but in the vast field of hydraulic and mechanical engineering the Chinese coolie has no serious rival. Endless examples might be quoted of highly efficient duty performed by Chinese coolies, and many illustrations given of their adaptability and inventiveness; and often, in the initial steps of a new undertaking, it is quite good policy to take advantage of Chinese methods until the future success of the venture is so well assured as to justify capital outlay for large-scale mechanical operations.

So shrewd an observer of men as General Sir Ian Hamilton says of them, "I admit that some of their habits are dirty. Otherwise I can only discover in them qualities so admirable that they fill me with alarm when I think how far we have fallen behind them. To me these Northern Chinese are an astounding set of fellows. I have never in my life even imagined a set of people so passionately, feverishly devoted to work. They put their backs into what they have to do as if their very lives depended on it. Energy is only half the battle: their men and women possess high individual intelligence to guide that energy. When I see a Chinaman mending a road or unloading rice bags with apparently twice the energy of the Western working man and for less than a tenth of his pay, I wonder how it is all going to end."

In tin-mining, on Chinese-owned properties in Malaya, coolies are dissatisfied with less than 80c. (22½*d.*) to $1·25 (2*s.* 11*d.*) per diem; and the day is often only 6 hours, and very rarely more than 7. But settlements are made at oftenest once in six months, and commonly only annually; and the coolie is virtually under a sort of legalised slavery.

On European-managed tin-mines, the Chinese labourer escapes some, at any rate, of the "squeezes" to which he is subjected by his own countrymen, and is, moreover, certain to receive his pay at not more than monthly intervals. Hence he is satisfied with somewhat lower rates, and commonly about 66–80c. (18½–22½*d.*) are ruling figures. As against 80c. to Chinese, often not more than 50c. (14*d.*) will be paid to Tamils doing the same work on the same mine, and doing it practically as well. In exceptional situations, remote from food and opium supplies, and where gambling and other recreations are not encouraged, the Chinaman will not stay for much less than $2 (4*s.* 8*d.*) a day. In the Chinese smelting houses, foremen receive $3 and coolies $2 daily.

Perhaps the most striking feature about Chinese engineering construction, whether on the large or the small scale, is the way in which the Chinaman avoids the use of nails, screws, and bolts. In everything he relies upon wooden joints, often most ingeniously contrived, and strengthens these by binding with the natural rope of the country, viz., the supple and surprisingly strong rotan (rattan) cane. Thus one class of skilled workman—the carpenter—suffices, and there is no need for any imported and costly hardware. Practically every coolie can handle a small joiners' axe with more or less skill.

While alluvial mining and everything to do with it comes naturally to all Chinese, hard-ground mining has to be taught from the beginning. New arrivals have never before seen drill, hammer, or dynamite; yet, such is their intelligence and application, in a fortnight or less they will be doing most creditable work. As with all coloured labourers, they are best on a down hole. The Chinaman always makes a very poor fist at "rising," and can never accomplish an "upper" satisfactorily. He does not admit of crowding quite so closely as the Kaffir. Under supervision, a pair of men can quite well bore a 3 ft. hole within the 6-hr. shift, in moderately hard ground; but, if not watched, they will fire shallow holes. They have no idea what burden to give a hole, or what charge of dynamite it will require. Hence, though the loss is immediately their own, their waste of explosive is considerable. Then again, as a result of their extraordinary jealousy, mutual distrust, and horror of doing anything which by any chance may benefit another, no pair will leave a hole unfired for the next shift to finish; no matter what its depth, or how little ground it may break, they will fire it. With operations on a scale to warrant the extra white wages, it would be infinitely preferable to pay Chinese miners on the basis of footage of holes bored, fixing a minimum limit of depth, and leaving the placing and firing of the holes to be done by skilled Europeans. In that way, one might utilise the Chinaman's good qualities and avoid his bad ones.

As a drill sharpener, the Chinese blacksmith manifests a lack of judgment in tempering the bits, and commits a shocking waste of metal by cutting off blunt ends, unless perpetually checked and fined. In the East, where the white man is so rare and the yellow man so numerous, the former will not undertake any manual work —he will only supervise. Elsewhere it ought to be possible to confine the skilled labour to European mechanics, and in that way the fullest advantage could be taken of each class.

It has been remarked by Hoover, "As we proceed up the scale of skill, the Chinaman falls far behind, until, when we arrive at the skilled mechanic and factory operative, the productive costs with Chinese labour are as great per unit of production as in England and America." To this, I would add that the Chinaman's "skilled labour" is never really skilled, though he is capable of contriving many ingenious things. He lacks the precision necessary for truly skilled work, as we understand the term in a fitting shop; and, though he may make one thing "fit" another accurately, it is always done by the most ingenious and painstaking cutting and contriving, and not by working to exact dimensions.

Yet he is a marvellous worker, and, with all his faults, infinitely preferable, not only to any other coloured labour, whether Asiatic or African or American, but to any white labour, in the world, so far as actual work is concerned. A ready emigrant to any climate, most adaptable to strange conditions, always cheerful under all circumstances, he asks nothing more than the unhindered opportunity to work and earn to the utmost limit of his capacity, to be allowed to spend his money as he likes, to indulge his vicious appetites, propitiate his gods with joss-sticks and crackers, cultivate his vegetables, and generally to keep himself to himself, with the assurance that he will be taken back to China when his contract is finished. Work he loves, not for its own sake, but simply as a means of obtaining the wherewithal to procure his pleasures. And those pleasures are innocent indeed as compared with the drunken habits of the white man.

As a day labourer the Chinaman can loaf more successfully than any other race, but on task work he is supreme. The gambling spirit which dominates every Chinaman makes contracting appeal strongly to him, and no man can or will work harder to earn a possible bonus. Given an inducement, there is no risk he will not run—and come out unscathed—and no undertaking too onerous for him. I have known many men to work in two contract parties at once, that is, putting in say 8 hr. with one (stoping) and 6 hr. with another (shaft-sinking) in every day. Old hands, left alone to do things their own way, will take out a piece of dangerous ground as well as, and in many cases better than, the average white miner. As timbermen, they are equal to the best that Cornwall can produce, so far as actual workmanship is concerned, and they are full of resource in saving labour and material.

Average wages paid to Chinese at the Raub gold mines, Malaya, during the past 4 years, have been as follows: Shift-bosses, 3*s.* 6*d.*; carpenters and smiths, 2*s.* 9*d.*; engineers' fitters, 22½*d.*–7*s.*; timbermen, 2*s.* 4*d.*; shovellers and surface labourers, 16½*d.*; and the average earnings on contract work have been: sinking and rising, 22½*d.* per shift of 6 hr.; driving, 18¼*d.* per 8 hr.: stoping, 16¾*d.* per 8 hr.

The Malay is peculiarly useful in Malaya for light duties, and especially for checking and tallying against the Chinese, their racial hatred for each other being intense; while the Javanese Malay in particular makes a splendid engine-driver, pump man, and battery hand.

ANIMAL POWER.

Horse.

Hauling.—(*a*) At a speed of 10 miles per hr. on a turnpike road, a horse will perform 13 miles per day for 3 years. In ordinary staging, a horse will perform 15 miles per day.

Assuming maximum load that a horse can draw on a gravel road as a standard, he can draw:—

On best broken-stone road	2 to 3 times.
On a well-made stone pavement ..	3 to 5 ,,
On a stone trackway	7 to 8 ,,
On plank road	4 to 12 ,,
On a railway	18 to 20 ,,

(*b*) From results of trials upon strength and endurance of draught horses at Bedford, it was determined that a good horse can draw 1 ton at rate of 2·5 miles per hr., 10–12 hr. per day.

(*c*) Comparative efficiencies of an average horse, according to speeds, duration and medium, are estimated to be :—

Velocity per hour.	Duration of Work.	Useful Effect, drawn 1 mile.		
		On a Canal.	On a Railroad.	On a Turnpike.
miles.	hours.	tons.	tons.	tons.
2·5	11·5	520	115	14
3	8	243	92	12
4	4·5	102	72	9
5	2·9	52	57	7·2
6	2	30	48	6
7	1·5	19	41	5·1
8	1·125	12·8	36	4·5
10	·75	6·6	28·8	3·6

(*d*) Percental capabilities on grades as compared with level road :—

	%		%		%
1 in 100	91	1 in 60	85	1 in 30	70
1 „ 90	90	1 „ 50	82	1 „ 25	64
1 „ 80	89	1 „ 45	80	1 „ 20	55
1 „ 75	88	1 „ 40	77	1 „ 15	40
1 „ 70	87	1 „ 35	74	1 „ 10	10

(*e*) Average daily work of a Flemish horse in North of France, where country is flat and roads are heavy, is :—

Winter, 21·82 ton-miles per day
Summer, 27·82 „ „ } Mean for the year, 25.

(*f*) Ordinary work of a horse may be stated at 22,500 ft.-lb. per min. for 8 hr. a day.

(*g*) Tractive power decreases as speed increases, and this nearly in inverse ratio up to 4 miles per hr.

(*h*) Load for 2 horses will range from 1 to 3 t., according as road varies from soft earth to hard macadam ; mileage for 10 hr. day varies in same way from $7\frac{1}{2}$ to $12\frac{1}{2}$ m. loaded, and equal distance empty : small steep grades and bad spots allowed for.

(*i*) To find useful work of a horse. Rule.—Multiply the number of trips by distance (miles) and by weight (tons) of each trip = tons drawn 1 mile per diem. A horse is capable of exerting a force of 120 lb. travelling at the rate of 2–3 miles an hour, for 10 hr., or for 20 miles per diem. Then—

$$\frac{3 \text{ miles per hour} \times 5280 \text{ ft. in a mile}}{60 \text{ minutes in an hour}} = 264,$$

$$264 \times 120 \text{ lb. force} = 31{,}680 \text{ ft.-lb.}$$

Examples.—(1) Assuming the horse-power at 120 lb., how many cars weighing $2\frac{1}{4}$ cwt. (252 lb.) each can a horse draw on a tramway rising 1 in 120 and taking the friction at $\frac{1}{65}$? *Ans.*

$$(252 \text{ lb.} \times \tfrac{1}{120} =)\ 2{\cdot}1 + (252 \text{ lb.} \times \tfrac{1}{65} =)\ 3{\cdot}877 = 5{\cdot}977,$$

$$\frac{120}{5{\cdot}977} = 20 \text{ cars.}$$

(2) Assuming a car load to weigh 1600 lb., the line rising 1 in 24, and friction at $\frac{1}{65}$, what force is necessary to draw it? *Ans.*

$$(1600 \text{ lb.} \times \tfrac{1}{24} =)\ 66{\cdot}66 + (1600 \text{ lb.} \times \tfrac{1}{65} =)\ 24{\cdot}6 = 91{\cdot}26 \text{ ft.-lb.}$$

(Wood.)

Carrying.—On his back, a horse can carry about 27·5% of his weight, or ordinarily 220–390 lb.

Ploughing.—In "ploughing" for excavation work, a 2-horse team will loosen 250–500 cub. yd. per 10 hr., according to resistance. In America, where team feed and driver's wages total 14*s.* 6*d.* a day, and ploughman's wages 6*s.* 3*d.*, the cost works out at $\frac{1}{2}$–1*d.* per cub. yd. Ordinarily, 1 team can "plough" for 4 "scrapers."

Scraping.—(*a*) With a "scraper" or scoop, in America on short trips, a horse team can go about $2\frac{1}{2}$ miles per hr., taking a load of $\frac{1}{8}$ cub. yd. (in place). On a round trip of 80 yd., the haul will occupy $1\frac{1}{2}$–2 min., $\frac{1}{2}$ min. of which will be spent in loading and tipping. The labour will embrace 1 man loading, 1 leading, and 2 horses.

(*b*) A wheeled scraper will take a larger load, say $\frac{1}{5}$–$\frac{1}{3}$ cub. yd., especially with the aid of a "snatch" team and extra labour, but the cost will not be lessened.

(*c*) In Alaska, a team (2 horses and 1 man) will scrape, on soft schist bedrock, 30–40 cub. yd. ordinary small gravel, 25 yd., per diem. (Purington.)

Mule.

(*a*) The big S. American mule is equal to a horse in strength, and superior in endurance and cost of keep.

(*b*) Underground colliery haulage in Illinois, by mules, on 275 working days per ann. (mules costing 45*l.* ea., depreciation 20%, interest 6%, drivers 10–11*s.* a day), cost 1·2*d.* per t., as against electric haulage ·7*d.* (Peltier.)

Ass.

The powerful Syrian ass carries a huge load—450–550 lb. of grain; but ordinarily the ass is less than half as effective as a horse. On the other hand, he is far more cheaply fed, housed and tended, and is much hardier and more immune against epidemics.

Ox.

The working ox is not equivalent to more than $\frac{2}{3}$ horse, but possesses a marked advantage in team work (oxen pulling so much better together) and in steady straining on the hauling where horses would jerk ineffectively. Oxen should always be shod for road work.

Llama.

In the Andean country (S. Amer.), the llama is the only available beast of burden. He is equal to a load of 100 lb. at 10 miles a day for several consecutive days, and may be pushed to 20 miles in a single day, but will require a week's rest to recover. The llama thrives at 12,000 ft., costs about 1*l.* per head, and is cheaply fed, thriving on the local grass. (Pearse, 1893.)

WIND POWER.

The altitude or head of the atmosphere at uniform density will be the altitude of a column of water 33·95 ft. divided by the sp. gr. of the air (·0012046), or,

$$\frac{33 \cdot 95}{\cdot 0012046} = 28{,}183 \text{ ft.}$$

The velocity due to this head will be—

$V = 8 \cdot 02 \sqrt{28{,}183} = 1346 \cdot 4$ ft. per sec., the velocity with which the air will pass into a vacuum.

When air passes into an air of less density, the velocity of its passage is measured by the difference of their density.

H and h = density of air in inches of mercury; t = temperature at time of passage; and V = velocity of wind in ft. per sec.

$$V = 1346 \cdot 4 \sqrt{\frac{H - h}{h}\left(1 + \cdot 00208\, t\right)}.$$

The force of wind increases as the square of its velocity.

a = area (sq. ft.) exposed at right angles to wind; F = force of wind (lb.); H = h.p.; and v = velocity of plane **a** in direction of wind, + when it moves opposite, and − when it moves with wind:—

$$F = \cdot 002288\, \mathbf{a}\, V^2, \qquad \text{when } v = o.$$

$$F = \cdot 002288\, \mathbf{a}\, (V \pm v)^2 \qquad H = \frac{\mathbf{a}\, v\, (V \pm v)^2}{240{,}384 \cdot 6}.$$

Ex.—A train running E.N.E. 25 m. per hr. exposes a surface of 1000 sq. ft. to a pleasant brisk gale N.E. by E. Required resistance to train in the direction it moves, and h.p. lost:—

E.N.E. – N.E. by E. = 3 points = 33° 45′; V = 14 ft. per sec., a brisk gale; $v = 25 \times 1 \cdot 467 = 36 \cdot 6$ ft. per sec., and $F = \cdot 002288 \sin.^2 33° 45' \times 1000 (14 + \cos. 33° 45' \times 36 \cdot 6)^2 = 305 \cdot 1$ lb.

$$H = \frac{305 \cdot 1 \times 36 \cdot 6}{550} = 20 \text{ h.p.} \quad \text{(Nystrom.)}$$

Horse-power and Sail-area. (Molesworth.)

V = Velocity of wind in ft. per sec.
A = Total area of sails in sq. ft. = 1,100,000 h.p. ÷ V^3.
N = Number of sails.
hp. = $\cdot 00000091 \text{ A } V^3$.

Velocity of tips of sails = 2·6 V, nearly.

Velocity and Force of Wind in lb. per sq. in. (Nystrom.)

Miles per Hour	Ft. per Sec.	Lb. per sq. ft.	Common Appellations of Winds.	Miles per Hour.	Ft. per Sec.	Lb. per sq. ft.	Common Appellations of Winds.
1	1·47	0·005	Hardly perceptible.	18	26·4	1·55	Very brisk.
				20	29·34	1·968	
2	2·93	0·020	Just perceptible.	25	36·67	3·075	
3	4·4	0·044		30	44·01	4·429	High Wind.
4	5·87	0·079	Gentle pleasant wind.	35	51·34	6·027	
5	7·33	0·123		40	58·68	7·873	
6	8·8	0·177		45	66·01	9·963	Very High.
7	10·25	0·241		50	73·35	12·30	
8	11·75	0·315	Pleasant brisk gale.	55	80·7	14·9	
9	13·2	0·400		60	88·02	17·71	Storm.
10	14·67	0·492		65	95·4	20·85	Great Storm.
12	17·6	0·708		70	102·5	24·1	
14	20·5	0·964		75	110	27·7	Hurricane.
15	22·00	1·107		80	117·36	31·49	
16	23·45	1·25		100	140·66	50	Tornado.

Dimensions of Sails. (Molesworth.)

Length of whip	30 ft.
Breadth, base	12 in.
Depth ,, ...	9 ,,
Breadth, tip	6 ,,
Depth ,, ...	4½ ,,

Rule for Angles of Sails. (Molesworth.)

A = Angle of sail with plane of motion at any part.
R = Total radius of sail in ft.
D = Distance of any part of sail from axis.

$$A = 23° - \frac{18 D^2}{R^2}.$$

If radius of windmill sails be divided into six equal parts, the angles at each part, reckoning from axis, will be:—

Distances from axis	$\frac{1}{6}$	$\frac{2}{6}$	$\frac{3}{6}$	$\frac{4}{6}$	$\frac{5}{6}$	tip.
Angle of sail with axis..	$67\frac{1}{2}°$	69°	$71\frac{1}{2}°$	75°	$79\frac{1}{2}°$	85°
Angle of sail with plane of motion ..	$22\frac{1}{2}$	21	$18\frac{1}{2}$	15	$10\frac{1}{2}$	5

In a windmill about 60 ft. diam., the diam. of middle point of arm is 30 ft.; circum. of circle in which that point revolves, 94 ft.; number of rev. made a minute, with a 5 mile an hour wind $\frac{660}{94}$, about 7. The speed of extremities of arms is 1320 ft. a min., or about 15 miles an hour, or 3 times that of the wind. Under ordinary circumstances, speed of outer extremities of arms ranges from 20 to 30 miles an hour, say 30 miles an hour when the wind blows at 10 miles with a pressure of about $\frac{1}{2}$ lb. per sq. ft. The total surface of sails unfurled in a mill 60 ft. diam., is 1250 sq. ft.; say half lost by furling = 625 sq. ft. effective. As the surface is set obliquely to the wind, pressure in direction of motion would be reduced from $\frac{1}{2}$ lb. to about $\frac{1}{7}$ lb., as a mean over the whole of the arms, giving a total pressure in direction of motion of about 90 lb. The mean velocity of arms is half that of the extreme = 15 miles an hour, or 1320 ft. a min. Therefore 90 lb. moving at 1320 ft. a min. = 90 × 1320 = 118,800 lb. moving at 1 ft. a min. = about $3\frac{1}{2}$ h.p. By doubling diameter of a mill, its effective surface (i.e. its power) is quadrupled.

Wind power, from its general distribution, is a more valuable auxiliary than tide or waves. The chief objections to it practically are its uncertainty in amount and the variable speed of the motor itself. But it may be profitable to employ a windmill where the work to be done admits of suspension during a calm or of storage of energy.

The average velocity of the wind is low, in most places between 5 and 10 miles an hour, corresponding, respectively, to pressures of 2–8 oz. per sq. ft. At most inland places it may be taken as about $7\frac{1}{2}$ miles an hour; but in some exposed situations, near the sea, it amounts to as much as $16\frac{3}{4}$ miles an hour. A speed of 10 miles is generally attained during 6–9 months per ann., according to locality, whilst a 16-mile wind may be expected, under favourable conditions, for about 4 mos. There are few days without periods of brisk breezes (15–20 miles an hour), giving wind pressures of 1–2 lb. per sq. ft.

An effective wind motor should be able to work at good advantage up to, say, 5 lb. per sq. ft. pressure at fairly uniform speed; and should be strong enough to stand up against winds of 50–60 miles an hour.

The work yielded by a mill constructed on the best principles should be ·04 ft.-lb. per sq. ft. of sail surface, with a wind velocity of 3·28 ft. per sec., and will increase with the cube of the speed of the wind, subject, of course, to limitations.

WATER POWER.

The natural power contained in a fall of water is equal to the weight of the quantity of water passing over per second, multiplied by the vertical space through which it falls.

1 cub. in. fresh water	= ·03621 lb.; cub. in. × ·00360 = gal. (Eng.).
1 „ ft. „	= 6·24 gal. (Eng.), or 6·32 gal. (U.S), or 62·57 lb., or ·559 cwt., or ·0312 ton (of 2000 lb.), or ·0278 t. (of 2240 lb.).
1 cub. ft. sea water	= 64·11 lb.; weight of sea water = 1·027 of fresh water.
1 „ „ ice	= 57·3 lb.
1 „ yd. fresh water	= 1682·5 lb.
1 gal. (Eng.) „	= 10 lb. or ·16 cub. ft., or 4·543 litres; gal. × ·16045 = cub. ft., or × 277·274 = cub. in., or × ·0044 = tons.
1 lb. „	= ·01607 cub. ft., or ·1 gal.
1 cwt. „	= 1·8 cub. ft., or 11·2 gal.
1 ton (2240 lb.) „	= 35·97 cub. ft., or 224 gal.
1 „ (2000 lb.) „	= 32·11 cub. ft., or 200 gal.
1 litre „	= 61 cub. in., or ·0353 cub. ft., or ·22 gal.
1 kilo „	= 2·204 lb.
1 cub. metre „	= 1000 kilo., or 1000 litres, or 35·31 cub. ft., or 1·308 cub. yd., or 220 gal., or 1 ton approximately.

Rainfall in in. × 2,323,200 = cub. ft. per sq. mile, or × $14\frac{1}{2}$ = millions of gal. per sq. mile.

When p = pressure (lb. per sq. in.), h = head of water (ft.), v = theoretical velocity (ft. per sec.), and g = sp. gr.; then—

$$p = h \times \cdot 4335. \quad h = p \times 2 \cdot 307. \quad \frac{1}{2\,g} = \cdot 0155.$$

Pressure per sq. ft. = ·4335 × 144 = 62·424 lb.

$$g = 32 \cdot 2. \quad 2\,g = 64 \cdot 4. \quad \sqrt{2\,g} = 8 \cdot 025.$$

$$v = \sqrt{2\,g\,h} = 8 \cdot 025\,\sqrt{h}. \quad h = \frac{v^2}{2\,g} = \cdot 0155\,v^2.$$

Theoretical h.p. developed by given quantities of water at given heights of fall.

Cub. Ft. per Min.	Height of Fall (Ft.).											
	5	10	20	30	40	50	60	70	80	90	100	200
10	0·095	0·189	0·378	0·568	0·757	0·946	1·135	1·324	1·513	1·703	1·892	3·783
20	0·189	0·378	0·757	1·135	1·513	1·892	2·270	2·648	3·027	3·405	3·783	7·567
30	0·284	0·568	1·135	1·703	2·270	2·838	3·405	3·973	4·540	5·108	5·675	11·350
40	0·378	0·757	1·513	2·270	3·027	3·783	4·540	5·297	6·053	6·810	7·567	15·133
50	0·473	0·946	1·892	2·838	3·783	4·729	5·675	6·621	7·567	8·513	9·458	18·917
100	0·946	1·892	3·783	5·675	7·567	9·458	11·350	13·242	15·133	17·025	18·917	37·833
200	1·892	3·783	7·567	11·350	15·133	18·917	22·700	26·483	30·267	34·050	37·834	75·667
300	2·837	5·675	11·350	17·025	22·700	28·375	34·050	39·725	45·400	51·075	56·750	113·500
400	3·783	7·567	15·133	22·700	30·266	37·833	45·400	52·966	60·533	68·099	75·667	151·333
500	4·729	9·458	18·917	28·375	37·833	47·292	56·750	66·208	75·667	85·125	94·583	189·167
600	5·675	11·350	22·700	34·050	45·400	56·750	68·100	79·450	90·800	102·150	113·500	227·000
700	6·621	13·242	26·483	39·725	52·966	66·208	79·450	92·691	105·933	119·175	132·416	264·833
800	7·566	15·133	30·267	45·400	60·533	75·667	90·800	105·933	121·067	136·200	151·333	302·667
900	8·509	17·024	34·049	51·075	68·099	85·125	102·150	119·174	136·200	153·225	170·249	340·500
1000	9·458	18·916	37·833	56·750	75·667	94·583	113·500	132·416	151·333	170·250	189·166	378·333
2000	18·916	37·833	75·667	113·500	151·333	189·167	227·001	264·834	302·667	340·501	378·334	756·667

Head, Pressure, and Power, at 100 *gal. per min. at* 62° F.

Head, Ft.	Pressure. Lb. per sq. in.	Power. h.p.	Head. Ft.	Pressure. Lb. per sq. in.	Power. h.p.	Head. Ft.	Pressure. Lb. per sq. in.	Power. h.p.
1	·43	·03	625	270·63	15·79	1825	790·23	46·13
2	·87	·05	650	281·45	16·42	1850	801·05	46·76
3	1·30	·08	675	292·28	17·05	1875	811·88	47·39
4	1·73	·10	700	303·10	17·68	1900	822·70	48·02
5	2·17	·13	725	313·93	18·31	1925	833·53	48·65
6	2·60	·15	750	324·75	18·95	1950	844·35	49·29
7	3·03	·18	775	335·58	19·58	1975	855·18	49·92
8	3·46	·20	800	346·40	20·20	2000	866·00	50·55
9	3·90	·23	825	357·23	20·85	2025	876·83	51·18
10	4·33	·25	850	368·05	21·48	2050	887·65	51·81
11	4·76	·28	875	378·88	22·11	2075	898·48	52·44
12	5·20	·30	900	389·70	22·74	2100	909·30	53·07
13	5·63	·33	925	400·53	23·38	2125	920·13	53·70
14	6·06	·35	950	411·35	24·01	2150	930·95	54·33
15	6·50	·38	975	422·18	24·64	2175	941·78	54·96
16	6·93	·40	1000	433·00	25·27	2200	952·60	55·60
17	7·36	·43	1025	443·83	25·90	2225	963·43	56·23
18	7·79	·46	1050	454·65	26·53	2250	974·25	56·86
19	8·23	·48	1075	465·48	27·17	2275	985·08	57·49
20	8·66	·50	1100	476·30	27·80	2300	995·90	58·12
30	12·99	·76	1125	487·13	28·43	2325	1006·73	58·75
40	17·32	1·01	1150	497·95	29·06	2350	1017·55	59·39
50	21·65	1·26	1175	508·78	29·69	2375	1028·38	60·02
60	25·98	1·52	1200	519·60	30·33	2400	1039·20	60·65
70	30·31	1·77	1225	530·43	30·96	2425	1050·03	61·28
80	34·64	2·02	1250	541·25	31·59	2450	1060·85	61·91
90	38·97	2·27	1275	552·08	32·23	2475	1071·68	62·55
100	43·30	2·53	1300	562·90	32·86	2500	1082·50	63·18
125	54·13	3·16	1325	573·73	33·49	2525	1093·33	63·81
150	64·95	3·79	1350	584·55	34·12	2550	1104·15	64·44
175	75·78	4·42	1375	595·38	34·75	2575	1114·98	65·07
200	86·60	5·05	1400	606·20	35·38	2600	1125·80	65·70
225	97·43	5·68	1425	617·03	36·01	2625	1136·63	66·34
250	108·25	6·31	1450	627·85	36·64	2650	1147·45	66·97
275	119·08	6·94	1475	638·68	37·28	2675	1158·28	67·60
300	129·90	7·57	1500	649·50	37·91	2700	1169·10	68·23
325	140·73	8·22	1525	660·33	38·54	2725	1179·93	68·85
350	151·55	8·85	1550	671·15	39·17	2750	1190·75	69·49
375	162·38	9·48	1575	681·98	39·80	2775	1201·58	70·12
400	173·20	10·11	1600	692·80	40·44	2800	1212·40	70·75
425	184·03	10·74	1625	703·63	41·07	2825	1223·23	71·39
450	194·85	11·38	1650	714·45	41·70	2850	1234·05	72·02
475	205·68	12·01	1675	725·28	42·33	2875	1244·88	72·65
500	216·50	12·64	1700	736·10	42·96	2900	1255·70	73·28
525	227·33	13·27	1725	746·93	43·59	2925	1266·53	73·92
550	238·15	13·90	1750	757·75	44·22	2950	1277·35	74·55
575	248·98	14·53	1775	768·58	44·85	2975	1288·18	75·18
600	259·80	15·16	1800	779·40	45·49	3000	1299·00	75·82

The Miners' "Inch."

The miners' "inch" is an arbitrary measure of the quantity of water which will flow through a given space in a given time, adopted in the early days of American gold-mining, and established by the law of each miners' camp, without any attempt at a universal scale. Thus there are scarcely two localities where the miners' inch has the same signification, the size and shape of the outlet and the manner of discharging the water varying constantly.

The most common basis of calculation is the volume which will pass through an opening 1 in. square in a plank 2 in. thick, with a pressure or head ranging from 4 to 11 in. above the centre of the orifice. But the shape of the aperture is a most inconstant figure, and this alone tends to make the computations inaccurate, besides which, the varying head is an even greater source of discrepancy. Thus the statute "inch" in B. Columbia is given as 1·68 cub. ft. of water per min., and this is defined as equivalent to that quantity which will pass through an orifice ½ in. wide, 2 in. high, and 2 in. thick, with a constant head of 7 in. above the top of the orifice, and every additional "inch" shall mean so much as will pass through the said orifice "extended horizontally ½ in." But actual measurements (Drummond) have proved (*a*) that the first-named orifice discharges 2·147 cub. ft. instead of 1·68, and (*b*) that widening this orifice alters the coefficient of discharge. Standard "inches" in various districts range all the way from 1·39 to 2·34 cub. ft. per min. The convenience of a definite figure has come to be recognised, and as heads much exceeding 6 in. are incompatible with most ditch and flume deliveries, a low figure is desirable. The Institution of Mining and Metallurgy has adopted 1·5 cub. ft. per min. as its standard, and this conforms very closely with the recognised best practice in the United States, being in fact the actual "inch" as fixed by State law in California and Montana. Thus, 1 cub. ft. per sec. = 40 miners' "inches"; or, cub. ft. per min. × $\frac{2}{3}$ = miners' "inches." Another vague term, an outcome of what is considered the necessary flow for a first-class hydraulic undertaking, is the "head" or "sluice-head." This may really be anything: thus—(*a*) "the flow necessary for a box 12 in. wide set in a grade of 8 in. per 12 ft., or 20 miners' 'inches' (= 30 cub. ft.) per min."; (*b*) "equivalent to 60 miners' 'inches' of 1·5 cub. ft."; (*c*) "200 inches"; (*d*) "600 gal. per min.," which is equivalent to about 95 cub. ft. per min.; (*e*) "with a sluice-box set at about ½ in. per ft., 45 cub. ft. or 30 'inches' per min.;" (*f*) "as it became possible to set sluice-boxes on a steeper grade (9 in. per box of 12 ft.), the quantity increased to 100 cub. ft. or 67 'inches' per min." Thus the expression is so inaccurate as to be worse than useless.

Cost of Water Power.

Theoretically, falling water can furnish the cheapest power, while it is also self-renewing, and requires no current expenditure beyond the wear of the transmitting machinery. In many cases, however, costly works are needed before the power can be used. Where the supply is variable, extensive reservoirs may be needed to regulate it and to prevent an excessive supply at certain seasons from becoming dangerous. There is no other source of power which needs such careful preliminary plans and estimates.

At some installations in Switzerland and in Norway, power is obtained at a cost of only 17*s*. 9*d*. per h.p.-year.

At many Californian mines which are supplied with water power by "ditch companies," it is computed that pine-wood fuel at 21*s*. per cord affords steam power at about the same cost as water under a fall of 175 ft. at 10*d*. per miners' "inch," the "inch" being reckoned at 15,000 gal. per 24 hours (= 1·65 cub. ft. per min.). The usual selling price of water there is 7½ to 10*d*. per "inch"; and the cost of pine-wood is 16*s*. 8*d*. to 29*s*. per cord.

Measuring Streams.

The computation of the breadth, depth, and velocity (in ft. per min. as travelled by a float) of a stream is a very simple matter. The sectional area reduced to sq. ft. and × by the speed = cub. ft. per min., and this × $\frac{2}{3}$ (or ·66) = miners' "inches." It is necessary to select a portion of the stream having the most uniform cross section. Measure 120 ft. along the stream, and, at the extremities of this length, and at right angles across the stream, fix two straight cords; then get a few floats of wood (so weighty that when placed in the water they will not project above it so as to be materially affected by the wind). Drop the floats lightly into the current at a little distance above the upper cord, and note the time by a stop-watch that they take to pass over the distance between the two cords. This should be repeated several times with floats both in the middle and near the sides of the stream; the mean is then taken of the surface velocity of all the experiments. Having by these means found the several spaces run over in a given time, the mean proportion of all these trials is taken for the surface velocity of the water. Four-fifths of the surface velocity is a good approximation to take for the mean velocity of the stream, or the velocity it would have, supposing all the particles of the stream to move in every part of its channel with one uniform motion. Regard the 120 ft. as 100 ft. only, so as to have a safe margin of speed. If the channel of the stream has a moderately even outline, measure its depth at regular intervals from shore to shore. Add all these depths together, and divide the sum by the number of soundings. An average depth is thus gained. Calculate then the area of the section by multiplying the average depth in ft. by the width in ft.

To abridge calculation, the accompanying table (p. 21) shows mean velocities corresponding to surface velocities from 120 to 800 ft. per min. for ordinarily free-flowing streams.

Multiply the area by the velocity, and the product will be the flow (in cub. ft. per min.). The test for velocity should be made at the same point where the measurements for depth are made, and a place on the stream should be selected for both where the banks are as nearly parallel as may be, and where the current and flow are the most tranquil. *Ex.*—A stream is 24 ft. broad, and 10 soundings at every 2 ft. on a line from bank to bank give 2, 6, 8, 9, 7, 11, 11, 10, 9, and 2 in. as the depths. The average velocity as determined by float is 4 ft. per sec. What is the flow? *Ans.*—The sum of the 10 soundings is 75 in., which gives an average depth of 7·5 in., equal to ·625 ft. The section area then is 24 × ·625 = 15 sq. ft. The velocity being 4 ft. per sec., the flow = 15 × 4 = 60 ft. per sec. If the stream runs over a bottom so irregular that an average depth cannot be gained, or an average velocity measured, there is no recourse but to construct an artificial channel, having no grade, into which it may be turned while measuring. Considerable allowance for retarded speed must be made when a stream has a very rocky or bouldery bed.

Surface and Mean Velocities of Water (in ft. per min.).

Surface Velocity.	Mean Velocity.	Surface Velocity.	Mean Velocity.	Surface Velocity.	Mean Velocity.
120	98·00	182·5	154·80	245	212·50
122·5	100·25	185	157·10	247·5	214·85
125	102·50	187·5	159·40	250	217·15
127·5	104·75	190	161·70	252·5	219·50
130	107·00	192·5	164·00	255	221·80
132·5	109·25	195	166·30	257·5	224·15
135	111·55	197·5	168·60	260	226·45
137·5	113·80	200	170·90	262·5	228·80
140	116·05	202·5	173·20	265	231·10
142·5	118·30	205	175·50	267·5	233·45
145	120·60	207·5	177·80	270	235·75
147·5	122·85	210	180·10	272·5	238·10
150	125·15	212·5	182·40	275	240·45
152·5	127 40	215	184·75	277·5	242·75
155	129·65	217·5	187·05	280	245·10
157·5	131·95	220	189·35	282·5	247·45
160	134·20	222·5	191·65	285	249·75
162·5	136·50	225	193·95	287·5	252·10
165	138·80	227·5	196·30	290	254·45
167·5	141·05	230	198·60	292·5	256·75
170	143·35	232·5	200·90	295	259·10
172·5	145·65	235	203·25	297·5	261·45
175	147·95	237·5	205·55	300	263·75
177·5	150·20	240	207·85	305	268·40
180	152·50	242·5	210·20	310	273·10

Surface and Mean Velocities of Water—continued.

Surface Velocity.	Mean Velocity.	Surface Velocity.	Mean Velocity.	Surface Velocity.	Mean Velocity.
315	277·8	375	334·2	435	390·8
320	282·5	380	338·9	440	395·6
325	287·2	385	343·6	445	400·3
330	291·9	390	348·3	450	405·1
335	296·6	395	353·0	500	452·5
340	301·2	400	257·8	550	500·0
345	305·9	405	362·5	600	547·7
350	310·6	410	367·2	650	595·5
355	315·3	415	371·9	700	643·3
360	320·1	420	376·7	750	691·2
365	324·8	425	381·4	800	739·2
370	329·5	430	386·1		

Dams, Reservoirs, and Penstocks.

The utilisation of water for power purposes always involves the provision of some retaining structure through which the issue is controlled. The strength of such structures must be adapted to withstanding the pressure of the water, and it must be borne in mind that *water at rest exerts pressure* (*i.e. its weight*) *equally in all directions*—the sides have to support exactly the same load as the bottom, no matter what the area may be. Depth, therefore, is the sole factor of pressure, and is to be computed at 62·5 lb. upon every sq. ft. of surface (sides and bottom) for every 1 ft. in depth.

In all water-power installations dependent upon a running stream provision must be made for dealing with two classes of foreign matter which every stream carries—floating and non-floating bodies. Floating bodies may embrace logs, branches, leaves, ice, and fine silt. All but the last-named may be arrested by a hinged grid of iron bars, covered with wire netting if necessary, sloping against the stream. Screens may be conveniently made in sections not exceeding 7 ft. × 3 ft., mounted on angle-iron frames, and interchangeable, some spares being available while cleaning is going on. No. 6 galvanised wire, ½ in. mesh, is good. It must not be forgotten that twigs and leaves *sink* when sodden, and a screen to be effective must reach bottom. Silt may be best removed by settling pits (with conical bottoms if possible) furnished with a plugged drain and a powerful hose if pressure is available for hydraulically sluicing them out. Non-floating impurities are the gravel which is brought down by heavy floods and by the operations of alluvial miners on the stream above. These are easily and effectively got rid of by a sand-drain in the deepest point of the dam wall, controlled by a rack and pinion gate. It should be in the most direct possible line of the greatest flow of the stream, while the supply for the flume or ditch is taken from slack water. Abundant by-pass for flood-waters is of course always essential.

Channels.

Artificial channels, either ditches, flumes, or pipes, have to be constructed for conveying the water to the point where it is to generate power, after due calculation of the relative proportions and dimensions of the area, the inclination, the volume, and the velocity necessary.

Sectional Area.—To find the sectional area of a channel—

(*a*) when sides are perpendicular: width of bottom (in.) × height of one side (in.) = area (sq. in.); this ÷ 144 = area (sq. ft.).

(*b*) when sides are sloping: width at top (in.) + width at bottom (in.) × depth (in.) ÷ 2 = area (sq. in.); this ÷ 144 = area (sq. ft.).

(*c*) when sides slope to a point: width (in.) × ½ depth (in.) = area (sq. in.); this ÷ 144 = area (sq. ft.).

Wet Perimeter.—The "wet perimeter" of a channel is the transverse length of so much of the bottom and sides as is covered by water. The least wet perimeter necessary to accommodate a given volume is secured when the width of bottom is between 1¾ and 2¼ times the depth of sides, and, needless to say, such perimeter involves the least expenditure in excavation of ditch or construction of flume relatively to volume of water carried. But other considerations interpose. Thus evaporation—which may easily account for a 10% loss on a great length—is much more marked in shallow and in slow currents; while in frosty regions, narrowness will help the formation of an ice-crust which may allow the flow to continue 4–6 weeks longer.

Grade.—To find what grade or fall must be given to a channel of uniform section to obtain a given discharge in a given time. Required discharge (cub. ft. per sec.) ÷ sectional area (sq. ft.) = necessary velocity (ft. per sec.); multiply this by itself; multiply this product by wet perimeter (ft.), and multiply this product by ·0001114; divide this product by sectional area (sq. ft.), and call result *a*. Then multiply velocity by wet perimeter and this product by ·00002426; divide this by sectional area, and call result *b*. Then *a* + *b* = grade per ft. (in decimals of a ft.) required. *Ex.*—Required grade per ft. for flume 20 in. wide and sides 11 in. high to deliver 28 cub. ft. per sec. *Ans.*—[Sectional area = 220 sq. in. (say 1·528 sq. ft.); wet perimeter = 42 in. = 3·5 ft.] Discharge (28) ÷ sectional area (1·528) = velocity (18·32); this × itself = 335·622 × wet perimeter (3·5) = 1174·677 × ·0001114 = ·1308 ÷ sectional area (1·528) = ·0856 = *a*. Velocity (18·32) × wet perimeter (3·5) × ·00002426 = ·001555, this ÷ sectional area (1·528) = ·001 = *b*. Then *a* (·0856) + *b* (·001) = ·0866 ft. per ft. = 86·6 ft. per 1000 ft. = 1·039 in. per ft. = 12·47 in. per 12-ft. box.

Discharge.—To compute discharge (cub. ft. per sec.). Multiply sectional area (sq. ft.) × grade (ft. per ft.) × 9000 ÷ by wet perimeter (ft.); extract square root of quotient and subtract ·1089 ; result = mean velocity (ft. per sec.). This × sectional area (sq. ft.) = discharge (cub. ft. per sec.). *Ex.*—Required discharge from flume 30 in. wide and sides 12 in. high having grade of 1 in 100 = ·01 ft. per ft. *Ans.*—Sectional area (2·5 sq. ft.) × grade (·01) = ·025 × 9000 = 225 ÷ wet perimeter (4·5 ft.) = 50, whose sq. root is 7·0711, this − ·1089 = 6·9622 = mean velocity (ft. per sec.) × sectional area (2·5) = 17·4 cub. ft. per sec. discharge. For the greater frictional loss in a ditch, deduct about 10%.

Dimensions.—To ascertain dimensions needed to afford a given discharge (cub. ft. per sec.) on given grade, the accompanying tables (pp. 25–27) may be used. They are computed on the assumption of smooth and straight channels.

Ditches.

Leakage occurs most extensively in gravelly soils; 1–5 in. of surface per day are extreme losses, with an average, perhaps, of about 2 in., which it will be always safe to count on, except in old ditches. A high velocity decreases loss in this way, but is destructive to the banks; it should rarely exceed 200 ft. per min.

Ditches should always have a uniform grade, otherwise there will be an accumulation at some points and a thinning-out at others, with deposits of sand and silt, and increased danger of breakage. It is also highly advantageous to have a complete system of waste-weirs to carry off surplus waters occasioned by floods, and to lessen the damage of breaks. These should be put in just below where-ever a new stream falls into the ditch, and just above those places where, by reason of a shelly or crumbly soil, the ditch is weak.

At high altitudes, in the spring, difficulty occurs in starting the water, through accumulations of snow in the ditch; it is best to flush out in short sections (a mile or so). Cut a hole in the bank a mile from the head, and when the water has soaked that far it will carry off the unmelted snow through this break with great rapidity. As soon as clear, that hole is mended and another is made a mile farther on.

Damage and leakage from earth-slides, falling trees, roots, springs, animals (trampling and burrowing), and storms, must be provided against and allowed for in calculating capacity.

Cost of Ditching.—(*a*) When plough and scraper can be used, ditching can be done at 10*d.* per cub. yd. If the soil is so rocky as to call for the pick and shovel, it will cost 15–20*d.* A safe figure for a ditch 3 ft. wide at bottom, 4½ ft. wide at top, and 18 in. deep is 15*s.* 3*d.* per rod. (Californian rates.)

(*b*) With labour at 6*d.* per hr., 4*s.* per cub. yd. for rock, and 2*s.* 6*d.* for gravel, earth, and clay. (Kirkpatrick.)

(*c*) Peruvian labour on contract, 6*d.* per cub. yd. (Heath.)

Dimensions of Channels. Base to Perpendicular of the Side Slopes being as 3 : 4. (Hendy.)

Fall per Mile.	Fall per Rod.	T 2·2 ft. B 1·0 ft. D 0·8 ft. Section 1·28 sq. ft.	T 3·3 ft. B 1·5 ft. D 1·2 ft. Section 2·88 sq. ft.	T 4·4 ft. B 2·0 ft. D 1·6 ft. Section 5·12 sq. ft.	T 5·5 ft. B 2·5 ft. D 2·0 ft. Section 8·0 sq. ft.	T 6·6 ft. B 3·0 ft. D 2·4 ft. Section 11·52 sq. ft.	T 7·7 ft. B 3·5 ft. D 2·8 ft. Section 15·68 sq. ft.	T 8·8 ft. B 4·0 ft. D 3·2 ft. Section 20·48 sq. ft.
ft.	in.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.
1	·0375	0·45	1·33	2·67	5·57	9·05	13·46	20·26
2	·0750	0·63	1·88	3·87	7·88	12·80	19·04	28·64
3	·1125	0·77	2·30	4·74	9·65	15·67	23·32	35·08
4	·1500	0·89	2·65	5·47	11·14	18·52	26·93	40·51
5	·1875	1·00	2·97	6·12	12·46	20·24	30·11	45·30
6	·2250	1·09	3·25	6·70	13·65	22·17	32·98	49·62
7	·2625	1·18	3·42	7·24	14·74	23·94	35·63	53·58
8	·3000	1·26	3·75	7·73	15·75	25·60	38·08	57·28
9	·3375	1·34	3·98	8·21	16·71	27·15	40·39	60·76
10	·3750	1·41	4·19	8·65	17·61	28·62	42·57	64·05
11	·4125	1·48	4·40	9·07	18·47	30·02	44·65	67·18
12	·4500	1·54	4·60	9·48	19·30	31·35	46·64	70·65
13	·4875	1·61	4·78	9·86	20·08	32·63	48·54	73·03
14	·5250	1·67	4·96	10·24	20·84	33·87	50·38	75·79
15	·5625	1·73	5·14	10·60	21·57	35·05	52·14	78·44
16	·6000	1·78	5·31	10·94	22·27	36·20	53·86	81·02
17	·6375	1·84	5·47	11·28	22·96	37·31	55·51	83·51
18	·6750	1·89	5·63	11·60	23·63	38·39	57·11	85·93
19	·7125	1·94	5·78	11·92	24·28	39·44	58·58	88·29
20	·7500	1·99	5·93	12·23	24·91	40·47	60·21	90·58
21	·7875	2·04	6·08	12·54	25·53	41·47	61·70	92·82
22	·8250	2·09	6·22	12·83	26·12	42·45	63·15	95·00
23	·8625	2·14	6·36	13·12	26·71	43·40	64·57	97·15
24	·9000	2·18	6·50	13·40	27·29	44·34	65·95	99·23
25	·9375	2·23	6·63	13·68	27·98	45·24	67·32	101·28

T signifies top width ; B, bottom width ; D, depth.

Dimensions of Channels. Base to Sides as 3 : 4—*continued.* (Hendy.)

Fall per Mile.	Fall per Rod.	T 9·9 ft. B 4·5 ft. D 3·6 ft. Section 25·92 sq. ft.	T 11 ft. B 5 ft. D 4 ft. Section 32 sq. ft.	T 13·2 ft. B 6·0 ft. D 4·8 ft. Section 46·09 sq. ft.	T 16·4 ft. B 7·0 ft. D 5·6 ft. Section 62·72 sq. ft.	T 17·6 ft. B 8·0 ft. D 6·4 ft. Section 81·92 sq. ft.	T 19·8 ft. B 9·0 ft. D 7·2 ft. Section 103·68 sq. ft.	T 22 ft. B 10 ft. D 8 ft. Section 128 sq. ft.
	in.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.
1	·0375	28·04	37·1	58·4	96·5	138·3	189·2	261·2
2	·0750	39·67	52·4	82·7	136·4	195·7	267·6	369·4
3	·1125	48·59	64·2	101·4	167·1	239·6	327·7	451·3
4	·1500	59·10	74·1	117·1	192·9	276·7	378·4	522·3
5	·1875	62·71	82·9	130·9	215·7	309·3	423·1	584·0
6	·2250	68·70	90·8	143·4	236·3	338·8	463·5	639·8
7	·2625	74·19	98·1	154·8	255·3	366·0	500·5	691·0
8	·3000	79·53	104·8	165·5	272·9	391·2	535·1	738·7
9	·3375	84·14	111·1	175·6	289·4	415·0	567·6	783·5
10	·3750	88·68	117·1	185·1	305·0	437·4	598·2	825·9
11	·4125	93·02	122·9	194·1	319·9	458·7	613·2	866·2
12	·4500	97·15	128·4	202·8	334·2	479·1	655·4	925·6
13	·4875	101·13	133·6	211·1	347·8	498·7	682·1	941·7
14	·5250	104·94	138·7	219·0	360·9	517·5	707·8	977·2
15	·5625	108·63	143·5	226·6	373·6	535·7	732·8	1011·5
16	·6000	112·18	148·2	234·1	385·9	553·3	756·7	1044·7
17	·6375	115·64	152·4	241·3	397·8	570·3	780·1	1076·9
18	·6750	118·99	157·2	248·3	409·3	586·9	802·7	1108·1
19	·7125	122·26	161·5	255·1	420·5	601·5	824·8	1138·4
20	·7500	125·43	165·7	261·7	431·4	618·5	846·1	1168·0
21	·7875	128·53	169·8	268·2	442·0	633·9	867·0	1196·8
22	·8250	131·55	173·8	274·5	452·5	648·8	887·4	1225·0
23	·8625	134·51	177·7	280·7	462·9	663·4	907·4	1252·6
24	·9000	137·40	181·5	286·7	472·6	677·7	926·9	1279·5
25	·9375	140·24	185·3	292·6	482·3	691·6	946·0	1306·0

T signifies top width; B, bottom width; D, depth.

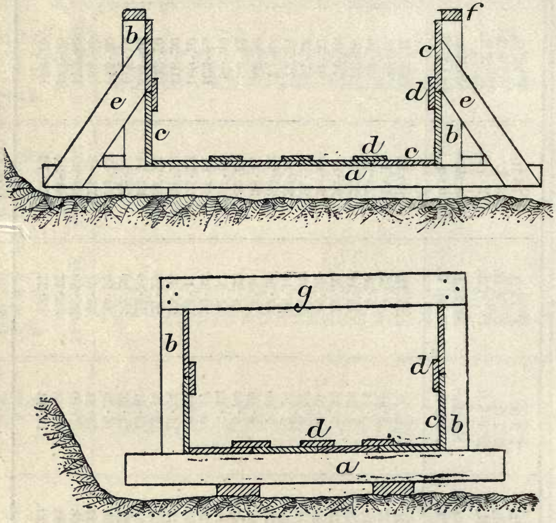
Dimensions of Channels. Base to Perpendicular of the Side Slopes being as 2 : 1. (Hendy.)

Fall per Mile.	Fall per Rod.	T 6 ft. B 2 ft. D 1 ft. Section 4 sq. ft.	T 9 ft. B 3 ft. D 1·5 ft. Section 9 sq. ft.	T 12 ft. B 4 ft. D 2 ft. Section 16 sq. ft.	T 16 ft. B 6 ft. D 2·5 ft. Section 27·5 sq. ft.	T 22 ft. B 10 ft. D 3 ft. Section 48 sq. ft.	T 28 ft. B 12 ft. D 4 ft. Section 30 sq. ft.	T 40 ft. B 20 ft. D 5 ft. Section 150 sq. ft.
ft.	in.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.	cub. ft.
·5 ..	·01875	1·27	3·85	8·63	18·11	38·79	78·2	188·1
·6667	·0250	1·46	4·44	9·96	20·91	44·79	90·3	217·2
·8333	·03125	1·63	4·96	11·14	23·38	50·08	101·0	242·8
1· ..	·0375	1·79	5·44	12·20	25·61	54·86	110·6	266·0
1·25 ..	·046875	2·00	6·08	13·64	28·68	61·32	123·7	297·4
1·5 ..	·05625	2·19	6·67	14·96	21·34	67·26	135·7	326·1
1·75 ..	·065625	2·37	7·19	16·14	33·88	72·57	146·4	351·8
2· ..	·0750	2·53	7·69	17·26	36·22	77·58	156·5	376·1
2·25 ..	·084375	2·68	8·16	18·30	38·42	82·29	165·9	399·0
2·5 ..	·09375	2·83	8·60	19·29	40·50	86·72	174·9	420·6
3· ..	·1125	3·10	9·42	21·14	44·36	95·00	191·6	460·7
3·5 ..	·13125	3·35	10·17	22·83	47·91	102·60	207·0	497·6
4· ..	·1500	3·58	10·87	24·41	51·22	109·70	221·3	531·9
4·5 ..	·16875	3·79	11·54	25·88	54·33	116·30	234·7	564·2
5· ..	·1875	4·00	12·16	27·29	57·27	122·70	247·4	594·8
6· ..	·2250	4·38	13·31	29·89	62·74	134·40	271·0	651·5
7· ..	·2625	4·73	14·39	32·29	67·78	145·10	292·7	703·6
8· ..	·3000	5·06	15·38	34·52	72·43	155·20	312·9	752·2
9· ..	·3375	5·37	16·31	36·61	76·83	164·60	331·9	797·9
10· ..	·3750	5·66	17·19	38·59	80·99	173·50	349·9	841·1
11· ..	·4125	5·93	18·03	40·47	84·94	181·90	366·9	882·1
12· ..	·4500	6·20	18·74	42·27	88·72	190·10	383·2	921·3

T signifies top width; B, bottom width; D, depth.

Flumes.

A good example of flume and ditchwork is presented by the Basic Co.'s canal, Idaho, built in 1898 for gigantic gravel-sluicing operations. Its total length is 43,868 ft., and it terminates at a vertical elevation of about 350 ft. above the stream which feeds it. There are 15,765 ft. of ditch, and 28,102 ft. of flume, 1977 ft. of the latter being on trestle-work, and 163½ ft. in a tunnel. The ditch is of uniform dimensions throughout—60 in. wide at bottom, 135 in. at top, 36 in. deep; angle of sides, 1 in 1; grade, 5·28 ft. per mile.



FIGS. 1, 2.—A MODERN FLUME.

The flume is of two sizes—one section is 54 in. wide, and the other is 48 in. wide; both are 27 in. deep. The grade of the large section is 5·28 and of the small 10·56 ft. per mile. The tunnel is 5 ft. wide at bottom, 4 ft. at top, and 6½ ft. high, inside measurements. The trestles are single deck, the highest bent not exceeding 25 ft.; span between bents is 12 ft. For the flume, a shelf 7 ft. wide was cut in the solid, necessitating much blasting. The average excavation was 17½ cub. ft. per foot run. The average excavation for the ditch was 135 cub. ft.; total excavation, excluding tunnel, 95,648 cub. yd. Owing to steepness of hillside, all excavation was done by pick, shovel and drill, except 16½ days' work for one team with plough and scraper.

The method of construction is shown in Figs. 1 and 2. As the grade was prepared, two continuous stringers or ground sills, 8 × 2 in., were laid, and on these the sills proper *a* and posts *b*, previously framed, were set. All timber was cut in the vicinity, and, together with nails and other supplies, was floated as near as possible to the scene of operations. The principal lumber dimensions are: Fig. 1: sills *a*, 8 ft. × 4 in. sq.; posts *b*, 4 in. sq.; lining boards *c*, 14 × 1 in.; lagging boards *d*, 6 × 1 in.; angle struts *e*, 6 × 1 in.; longitudinal tie *f*, 4 × 2 in. Fig. 2: sills *a*, 6 ft. × 6 × 2 in.; posts *b*, 3 ft. 4 in. × 4 in. sq.; lining and lagging boards *c d*, 14 and 6 × 1 in. respectively; top cross-ties *g*, 6 × 1 in. The sills *a* are duplicated, one each side of the post.

The details of cost are as follows: Excavation, covering labour, coal, blasting-powder, steel, superintendence, depreciation and loss of tools, and the grubbing out of timber along the line, averaged a little less than 10*d.* per cub. yd. As the route was quite heavily timbered, and so steep that the surveying in places was a matter of considerable danger, this figure is unusually low.

The construction work, which covered all carpentering labour, its superintendence, and the wear and loss of tools furnished by the company, amounted to 1*s.* 10¼*d.* per ft. run.

The consumption of lumber per ft. run, including waste, running board on the top pressure-box, and appurtenances, also a telephone pole line on 2 × 4 in. scantling for some of the distance, amounted to 27½ ft., board measure, and the cost of the lumber delivered at the point where it was placed in position averaged 1*s.* 2½*d.* per lin. ft. of flume constructed. The consumption of nails was a shade under 1 lb. per lin. ft. of flume, and the cost a little over 2¼*d.* The cost of flume excavation, per lin. ft., was 6¼*d.*; and of the ditch excavation, including everything necessary to permit the water to pass, 3*s.* 11¼*d.* The total cost of the flume constructed and in operation per lin. ft., excluding bridge and trestle-work carrying it over ravines, was 3*s.* 9*d.* per ft. run.

The scale of wages was as follows: Boys employed in passing lumber, 8*s.* 4*d.* per day; all other labour, 12*s.* 6*d.*, including all the grading crew and two-thirds of the construction crew; blacksmiths and carpenters, 14*s.* 7*d.* A few carpenters of extra ability and experience in constructing and raising trestles were paid 16*s.* 8*d.* The lumber delivered afloat in the canal cost 44*s.* 4*d.* per 1000 ft. board measure.

The capacity of the canal, as proved by experience, corresponds to the delivery at the pressure-box of 40 cub. ft. per second.

(*b*) With lumber at 50*s.*–62*s.* 6*d.* per 1000 ft. board measure, delivered at flume head so that it can be floated down, a flume

$2\frac{1}{2}$ ft. × $2\frac{1}{2}$ ft. can be built at 16*s*. per box (12 ft.), and one 6 ft. wide by $3\frac{1}{2}$ ft. high at 35*s*. 6*d*. per box, at Californian rates. (Van Wagenen.)

Piping.

Velocity.—Low velocities in pipes are always advisable, to diminish the loss of head by friction. The extra cost of increased dimensions is often warranted. In large pipes, the strains created by high velocity are very great, and it is not usual to exceed 4–5 ft. per sec. In one instance where the actual needs were 42·9 in. diam., the size was increased to 46·8 in., augmenting the cost by 9%, but the sectional area by 18%, and bringing the velocity down to about 6 ft. per sec.

To find velocity in a practically straight pipe, the head, length, and diam. being known—multiply diam. (ft.) by head (ft.) = *a*. Then total length (ft.) + diam. (ft.) × 54 = *b*. Divide *a* by *b*, extract square root of quotient and × 48 = desired velocity (ft. per sec.). *Ex.*—Find velocity in a pipe 12,600 ft. long, 6 in. (·5 ft.) diam., and 200 ft. head. *Ans.*—diam. (·5) × head (200) = 100 = *a*. Length (12,600) + diam. (·5) × 54 = 27 = 12,627 = *b*. Then *a* (100) ÷ *b* (12,627) = ·0079. Extract square root = ·0887 × 48 = 4·26 ft. per sec.

Friction.—Retardation or loss of head by friction is shown in accompanying tables (pp. 31–32) per 100 ft., the bold figures at top of columns being inside diam. of pipes (in.); column *a*, velocity (ft. per sec.); *b*, discharge (cub. ft. per min.); *c*, loss by friction (cub. ft. per min.).

Frictional resistance in a pipe full of flowing water does not arise from friction against the pipe itself, but against thin layers of water which adhere to it. The thickness of these layers, which are quite still or move very slowly, depends on the pressure upon the sides of the pipe. The greater this is, the thicker will be the layer, and therefore the smaller will be the diameter of the free area for flowing. In pipes with a constant or slightly varying fall, the line of hydraulic pressure is about parallel with the pipe. In such a case, the pressure is almost the same in every part, and the resistance may be taken as proportional to the length. It is different in the case of a siphon, when the hydraulic pressure varies very much.

Discharge.—To find discharge (cub. ft. per sec.) through reasonably straight pipe, the head, length and diam. being known—multiply velocity (ft. per sec.) by sectional area of pipe (sq. ft.) = discharge (cub. ft. per sec.).

Head.—To find necessary head—length and diam. being known—to produce given discharge (cub. ft. per sec.). Discharge × itself = *a*. Length of pipe + diam. × 54 = *b*. *a* × *b* = *c*. Diam. (ft.) ÷ ·235 = *d*. Multiply *d* by itself continuously 4 times = *e*. Then *c* ÷ *e* = head (ft.) *Ex.*—Required the head to produce discharge of 12 cub. ft. per sec. in pipe 350 ft. long and 8 in. (·666 ft.) diam. *Ans.*—Discharge (12) × 12 = 144 = *a*. Length (350) + diam. (·666) × 54 = 36 = 386 = *b*. Then 144 (*a*) × 386 (*b*)

Loss of Head by Friction.

a	3 *b*	3 *c*	4 *b*	4 *c*	5 *b*	5 *c*	6 *b*	6 *c*	7 *b*	7 *c*	8 *b*	8 *c*
1	2·95	·196	5·22	·147	8·17	·118	11·77	·098	16·03	·084	20·88	·074
2	5·89	·659	10·44	·494	16·34	·395	23·54	·329	32·05	·282	41·76	·247
3	8·83	1·35	15·67	1·02	24·51	·815	35·32	·679	48·08	·581	62·64	·509
4	11·80	2·28	20·89	1·71	32·69	1·37	47·09	1·14	64·11	·977	83·52	·856
5	14·70	3·43	26·12	2·57	40·87	2·05	58·87	1·71	80·15	1·47	104·40	1·28
6	17·70	4·78	31·34	3·59	49·05	2·87	70·64	2·39	96·18	2·05	125·28	1·79
7	20·60	6·35	36·57	4·77	57·22	3 81	82·41	3·18	112·21	2·73	146·16	2·39
8	23·56	8·14	41·79	6·11	65·40	4·89	94·19	4·07	128·24	3·49	167·04	3·06
9	26·51	10·12	47·02	7·59	73·57	6·07	105·97	5·06	144·27	4·34	187·92	3·79
10	29·45	12·32	52·24	9·24	81·75	7·39	117·74	6·16	160·30	5·28	208·80	4·62
11	32·40	14·71	57·47	11·03	89·92	8·82	129·52	7·36	176·34	6·31	229·68	5·52
12	35·34	17·31	62·70	12·98	98·10	10·38	141·30	8·65	192·37	7·41	250·56	6·49
13	38·33	20·10	67·92	15·08	106·27	12·06	153·07	10·05	208·40	8·61	271·44	7·54
14	41·23	23·12	73·15	17·34	114·45	13·87	164·85	11·56	224·43	9·91	292·32	8·67
15	44·20	26·32	78·38	19·74	122·62	15·79	176·63	13·16	240·46	11·28	313·20	9·87
16	47·12	29·72	83·60	22·29	130·80	17·83	188·40	14·86	256·48	12·74	334·08	11·15
17	50·05	33·33	88·83	25·00	138·97	20·00	200·18	16·67	272·51	14·29	354·96	12·50
18	53·00	37·14	94·05	27·86	147·15	22·29	211·96	18·57	288·54	15·92	375·84	13·93
19	55·95	41·12	99·28	30·84	155·32	24·67	223·73	20·56	304·57	17·62	396·72	15·42
20	58·89	45·32	104·50	33·99	163·50	27·19	235·51	22·66	320·60	19·42	417·60	17·00

a	9 *b*	9 *c*	10 *b*	10 *c*	11 *b*	11 *c*	12 *b*	12 *c*	13 *b*	13 *c*	14 *b*	14 *c*
1	26·47	·065	32·70	·059	39·55	·054	47·10	·049	55·30	·045	64·08	·042
2	52·94	·220	65·40	·198	79·10	·180	94·20	·164	110·60	·152	128·16	·141
3	79·41	·450	98·15	·407	118·65	·370	141·30	·339	165·90	·313	192·24	·291
4	105·90	·760	130·85	·685	158·20	·623	188·40	·570	221·20	·527	256·32	·489
5	132·37	1·14	163·50	1·03	197·76	·932	235·40	·855	276·50	·789	320·40	·735
6	158·84	1·59	196·20	1·43	237·30	1·30	282·50	1·20	331·80	1·10	384·48	1·03
7	185·31	2·12	228·90	1·90	276·85	1·73	329·60	1·59	387·10	1·46	448·57	1·36
8	211·80	2·71	261·60	2·45	316·40	2·23	376·70	2·04	442·40	1·88	512·66	1·75
9	238·29	3·37	294·29	3·03	355·95	2·76	423·80	2·53	497·70	2·33	576·75	2·17
10	264·77	4·11	327·00	3·70	395·50	3·36	470·90	3·08	553·00	2·85	640·84	2·64
11	291·26	4·90	359·70	4·41	435·05	4·01	518·00	3·68	608·30	3·39	704·93	3·15
12	317·74	5·77	392·39	5·19	474·62	4·72	565·10	4·32	663·60	3·99	769·02	3·71
13	344·22	6·70	425·09	6·03	514·17	5·48	612·20	5·03	718·90	4·64	833·10	4·30
14	370·70	7·71	457·79	6·93	553·72	6·30	659·30	5·78	774·20	5·33	897·18	4·95
15	397·18	8·77	490·49	7·90	593·27	7·18	706·35	6·58	829·50	6·08	961·27	5·64
16	423·65	9·91	523·18	8·92	632·82	8·11	753·45	7·43	884·75	6·86	1025·36	6·37
17	450·13	11·11	555·88	10·00	672·37	9·09	800·50	8·33	940·00	7·69	1089·45	7·15
18	476·61	12·38	588·58	11 14	711·92	10·13	847·60	9·29	995·30	8·57	1153·59	7·96
19	503·08	13·71	621·28	12·34	751·52	11·22	894·70	10·28	1050·60	9·49	1217·63	8·81
20	529·56	15·11	653·98	13·60	791·07	12·36	941·75	11·33	1105·90	10·46	1281·72	9·71

= 55,584 (*c*). Diam. (·666) ÷ ·235 = 2·834 (*d*). Then 2·834 × itself continuously 4 times = 182·801 (*e*). Then 55,584 (*c*) ÷ 182·801 (*e*) = 304 ft. (Van Wagenen.)

Loss of Head by Friction—continued.

a	15		16		17		18		19		20	
	b	c	b	c	b	c	b	c	b	c	b	c
1	73·58	·039	83·68	·037	94·56	·035	106·00	·033	118·09	·031	130·87	·029
2	147·16	·132	167·36	·123	189·12	·116	212·00	·110	236·18	·104	261·74	·099
3	220·74	·272	251·04	·255	283·68	·239	318·00	·225	354·27	·214	392·61	·204
4	294·32	·457	334·72	·428	378·24	·403	424·00	·380	472·36	·361	523·48	·343
5	367·90	·683	418·40	·640	472·80	·601	530·00	·570	590·45	·537	654·35	·515
6	441·48	·957	502·08	·895	567·36	·841	636·00	·795	708·54	·753	785·22	·715
7	515·07	1·27	585·76	1·19	661·92	1·12	742·00	1·06	826·63	1·00	916·09	·950
8	588·66	1·63	669·45	1·53	756·48	1·44	848·00	1·36	944·72	1·29	1046·96	1·23
9	662·25	2·02	753·14	1·89	851·04	1·78	954·00	1·68	1062·81	1·59	1177·83	1·51
10	735·84	2·46	836·83	2·31	945·60	2·18	1060·00	2·06	1180·90	1·95	1308·70	1·85
11	809·43	2·94	920·52	2·76	1040·16	2·59	1166·00	2·45	1298·99	2·32	1439·57	2·21
12	883 02	3·46	1004·21	3·24	1134·72	3·05	1272·00	2·89	1417·08	2·73	1570·44	2·59
13	956·60	4·02	1087·90	3·77	1229·28	3·55	1378·00	3·35	1535·17	3·17	1701·31	3·02
14	1030·18	4·62	1171·59	4·33	1323·84	4·08	1484·00	3·86	1653·26	3·65	1832·18	3·47
15	1103·77	5·26	1255·28	4·93	1480·40	4·65	1590·00	4·38	1771·35	4·16	1963·05	3·95
16	1177·36	5·94	1338·96	5·58	1512·96	5·25	1696·00	4·96	1889·44	4·69	2093·92	4·46
17	1250·95	6·67	1422·64	6·25	1607·52	5·88	1802·00	5·55	2007·53	5·26	2224·79	5·00
18	1324·54	7·43	1506·32	6·97	1702·08	6·55	1908·00	6·19	2125·62	5·86	2355·66	5·57
19	1398·13	8·22	1590·00	7·71	1796·64	7·26	2014·00	6·86	2243·71	6·49	2486·53	6·17
20	1471·72	9·06	1673·68	8·50	1891·20	8·00	2120·00	7·56	2361·80	7·16	2617·40	6·80

a	22		24		26		28		30	
	b	c	b	c	b	c	b	c	b	c
1	158·36	·027	188·44	·025	221·13	·023	256·56	·021	294·44	·019
2	316·72	·090	376·88	·082	442·26	·076	513·12	·071	588·88	·066
3	475·08	·185	565·32	·169	663·39	·157	769·68	·145	883·32	·136
4	633·44	·312	753·76	·285	884·52	·263	1026·24	·245	1177·76	·228
5	791·80	·466	942·20	·428	1105·65	·394	1282·80	·368	1472·20	·342
6	950·16	·650	1130·64	·600	1326·78	·550	1539·36	·515	1766·64	·478
7	1108·52	·865	1319·08	·795	1547·91	·730	1795·92	·680	2061·08	·635
8	1266·88	1·12	1507·52	1·02	1769·04	·940	2052·48	·875	2355·52	·815
9	1425·24	1·38	1695·96	1·27	1990·17	1·17	2309·04	1·08	2649·96	1·01
10	1583·60	1·68	1884·40	1·54	2211·30	1·42	2565·60	1·32	2944·40	1·23
11	1741·96	2·01	2072·84	1·84	2432·43	1·69	2822·16	1·57	3238·84	1·47
12	1900·32	2·36	2261·28	2·16	2653·56	2·00	3078 72	1·86	3533·28	1·73
13	2058·68	2·74	2449·72	2·52	2874·69	2·32	3335·28	2·15	3827·72	2·01
14	2217·04	3·15	2638·16	2·89	3095·82	2·67	3591·84	2·48	4122·16	2·31
15	2375·40	3·59	2826·60	3·29	3316·95	3·04	3848·40	2·82	4416·60	2·63
16	2533·76	4·06	3015·04	3·72	3538·08	3·43	4104·96	3·19	4711·04	2·97
17	2692·12	4·55	3203·48	4·17	3759·21	3·85	4361·52	3·58	5005·48	3·33
18	2850·48	5·07	3391·92	4·65	3980·34	4·29	4618·08	3·98	5299·92	3·72
19	3008·84	5·61	3580·36	5·14	4201·47	4·75	4874·64	4·41	5594·36	4·11
20	3167·20	6·18	3768·80	5·67	4422·60	5·23	5131·20	4·86	5888·80	4·53

Diameter.—To find diam. required for given discharge (cub. ft. per sec.), head and length being known—head (ft.) × 5280 ÷ length (ft.) = *a*. Discharge (cub. ft. per sec.) × itself × 5280 = *b*. Divide *b* by *a*. Extract fifth root, and × ·235 = diam. (ft.). *Ex.*—Required the diam. of a pipe 6000 ft. long, head 400 ft., to discharge 6 cub. ft. per sec. *Ans.*—Head (400) × 5280 = 2,112,000 ÷ length (6000) = 352 (*a*). Discharge (6) × itself = 36 × 5280 = 190,080 (*b*). Then *b* (190,080) ÷ *a* (352) = 540. Extract fifth root = 3·52 × ·235 = ·8272 ft. diam. (Van Wagenen).

Hydraulic Grade-Line.—This term is applied to an imaginary straight line, extending from a point on the side of the dam (denominated the velocity-head) to the outlet of the pipe. If the pipe be constructed exactly on this line, the water flowing through it, no matter what its velocity or volume, will exert no bursting pressure. In other words, the grade is such that the velocity caused is exactly sufficient to carry down all that the pipe will hold, and there is no outward pressure exerted except that due to the water's weight. If, however, there be a change in the diameter of the pipe at any point, this equilibrium ceases to exist. It is never possible in practice to adopt this line in pipe-laying, but generally close approximation to it is highly advantageous.

To find the hydraulic grade line, calculate the velocity due to total head. Find the head corresponding to this velocity. Lay off this head on the side of the dam from the top of the pipe-opening. Its termination will mark the line of the velocity head. From this point, sight to the outlet of the pipe; the line of sight is the hydraulic grade-line.

In constructing a line of piping, three conditions may arise from the inequalities of the ground to be passed over: (*a*) The pipe may lie below the hydraulic grade-line; (*b*) above it; (*c*) both above and below.

When the pipe is below hydraulic grade line, there is a bursting pressure varying with the distance. To find this pressure at any point, ascertain the distance of that point vertically below the hydraulic grade-line. Call this measurement the bursting-head—say 6 ft. The pressure, then, on each sq.in. of pipe at that point is equal to the weight of a column of water whose base measures 1 sq. in. and whose height is 6 ft. Thus, 1 sq. in. multiplied by 6 ft. (72 in.) = 72 cub. in. = ·04166 cub. ft. × 62·5 lb. (= cub. ft. of water) = 2·6 lb., which is the pressure per sq. in. Consequently if the pipe lies considerably below the hydraulic grade-line, it will need to be of thicker iron than the rest. This law applies in crossing deep hollows.

In a pipe above the hydraulic line there is a decided loss of head, and consequently of power, in portions of the pipe, if it be the same diameter throughout. Find now that point in the pipe which is highest above the hydraulic grade-line, and from that point draw two new grade-lines, one to the pressure box and one to the outlet. Along the former, calculate the bursting pressure as above, measuring the different heads from the new line. Along the latter there will be no bursting pressure, for the grade of the outlet end of the pipe will be so much greater than that of the reservoir end that it will carry off the water very much faster, and will, in fact, act like a gutter, and be partially empty. The remedy for this is to put in pipes having a decreased diameter. To calculate the requisite diameter, assume that the pipe ended at that point where it is highest above the hydraulic grade-line. Calculate the dis-

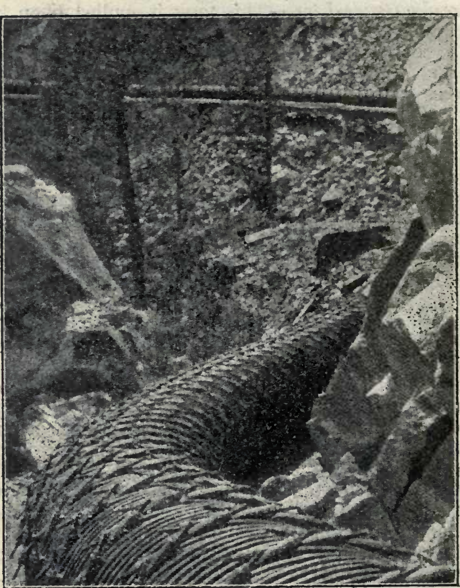


FIG. 3.—WOODEN PIPE-LINE.

charge (cub. ft.) at that point. This will give the amount of water (cub. ft. per sec.) which the outlet section must carry. The head will be the vertical distance from the highest to the lowest point.

Should the pipe be both above and below the hydraulic grade-line, divide the pipe into sections for every passage it makes above the hydraulic grade-line, and make the divisions at the several points where the pipe attains its highest position. Calculate the discharge at the end of each section. The first section will have a head equal to the vertical distance between its discharge and the water level in the pressure box. All succeeding heads will be measured from the level of the discharge just below them to their own discharge. These measurements will furnish a series of heads and grades from which the diameters of pipe necessary may be calculated (Van Wagenen).

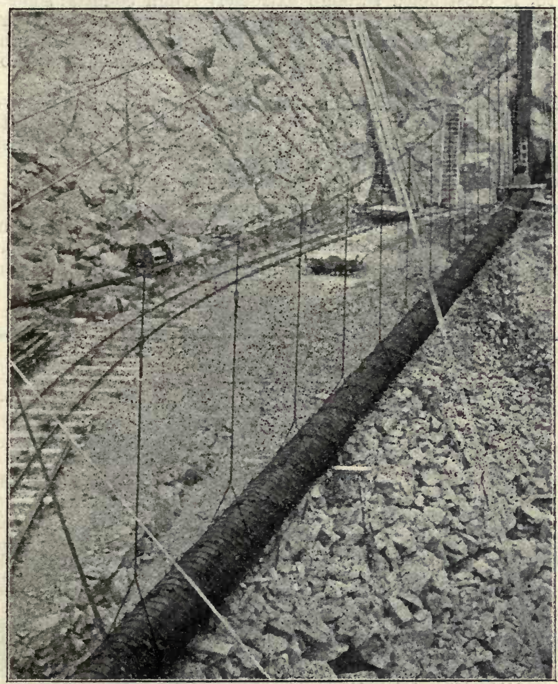


FIG. 4.—WOODEN PIPE-LINE.

Pipes, Wooden.—Under low pressures, up to 150–200 ft. head, wooden stave pipe suitably banded with steel rods, properly spaced, and each provided with turn-buckles for equalising the strain, has of late come into extended use. The pipe is an outgrowth of the old wooden flume. The life of such piping is unusually long, the losses in friction are very small, and in large sizes the cost is much less than for steel.

In a quite recent power plant at Pueblo (U.S.) use is made of a wooden stave pipe, 30 in. diam., the staves being secured by steel bands spaced at 2¼–8 in. centres, according to the head, which, in passing depressions, reaches 215 ft. This pipe-line, 4 miles in length, is constructed through extremely rough country, some of the curves being of less than 100 ft. radius, and one compound curve is stated to have a radius of 35 ft. The pipe passes through a tunnel over 1500 ft. long, and a portion of it is suspended by cable. See Figs. 3, 4.

Pipes, Iron and Steel.—The pipe most generally in use, however, is made of wrought iron or steel. In fact, outside of the United States one could hardly find either suitable timber or the requisite mechanical skill for making wooden pipes.

Strains.—In practice it is rare that the pipe lies more than 20 ft. below the hydraulic grade-line, and consequently the pipe is not called on to withstand more than 10 lb. per sq. in., which is within the strength of ordinary stove-pipe iron. On the other hand, occasionally the static head runs into high figures, examples in opera-

Strength of Sheet-iron Piping. (Van Wagenen.)

Diameter of Pipe, in inches.	Head of Water, in ft.									
	100	150	200	250	300	400	500	600	800	1000
	Resulting Pressure against Sides of Pipe, in lb. per sq. in.									
	43·4	65·1	87	109	130	174	217	260	347	434
	Required Thickness of Pipe, in inches or decimals of an inch.									
2	·009	·013	·018	·022	·027	·036	·045	·055	·075	·095
3	·013	·020	·026	·033	·040	·054	·068	·082	·112	·143
4	·017	·026	·035	·045	·053	·072	·090	·110	·149	·191
5	·022	·033	·044	·056	·067	·090	·113	·137	·186	·237
6	·026	·040	·053	·067	·080	·108	·136	·165	·224	·287
7	·030	·046	·062	·078	·093	·126	·159	·193	·261	·333
8	·034	·053	·071	·089	·107	·144	·181	·220	·298	·382
9	·039	·059	·079	·101	·120	·163	·205	·247	·335	·427
10	·044	·066	·089	·112	·134	·181	·227	·275	·373	·475
12	·053	·080	·106	·134	·161	·217	·273	·330	·448	·575
14	·061	·093	·124	·156	·187	·253	·318	·387	·523	·666
16	·069	·106	·142	·178	·214	·288	·363	·440	·596	·763
18	·078	·120	·159	·201	·242	·326	·409	·495	·670	·850
20	·088	·132	·177	·223	·267	·361	·454	·549	·746	·950
24	·105	·159	·213	·268	·321	·433	·545	·660	·895	1·150
30	·132	·198	·267	·336	·402	·543	·681	·825	1·120	1·420
36	·156	·238	·318	·402	·483	·651	·819	·990	1·340	1·710
42	·184	·279	·372	·469	·562	·759	·955	1·160	1·570	2·000
48	·210	·317	·425	·535	·641	·866	1·090	1·320	1·790	2·290

Thickness and "Numbers" of Sheet Iron.

No. 4	has a thickness of	·250 in.	No. 18	has a thickness of	·055 in.
„ 5	„	·200 „	„ 19	„	·052 „
„ 6	„	·166 „	„ 20	„	·050 „
„ 7	„	·142 „	„ 21	„	·047 ..
„ 8	„	·133 „	„ 22	„	·045 „
„ 9	„	·111 „	„ 23	„	·044 „
„ 10	„	·100 „	„ 24	„	·041 „
„ 11	„	·090 „	„ 25	„	·040 „
„ 12	„	·083 „	„ 26	„	·038 „
„ 13	„	·076 „	„ 27	„	·037 „
„ 14	„	·071 „	„ 28	„	·035 „
„ 15	„	·066 „	„ 29	„	·034 „
„ 16	„	·062 „	„ 30	„	·033 „
„ 17	„	·058 „			

Ordinary Dimensions of Pipe Lines.

Diameter of Pipe.	Pressure.	No. of Iron.	Thickness of Iron.
in.	ft.		in.
22	150	16	0·060
22	150 to 250	14	0·078
22	250 to 310	12	0·098
30	150	14	0·078
30	150 to 275	12	0·098
40	160	..	0·236

tion reaching 940 lb., 1047 lb., and 1165 lb. per sq. in., the falls ranging between 2000 and 3000 ft.

The annexed table shows the thickness of iron piping necessary to withstand given pressures.

The iron used varies generally from No. 16 to No. 11, according to the pressure, the best iron only being employed. The size of the pipe will depend upon the supply of water; with 1500–2000 miners' "inches" of water, a 22-in. pipe will suffice; where the supply is 3000 "inches," a 30-in. pipe must be used, and so on.

No. 14 iron will resist a pressure of 300 ft. head, or 130 lb. to the sq. in.; and an 11-in. pipe of No. 16 iron, a pressure of 500 ft., or 217 lb. to the in. No. 14 iron is ·083 in. thick, and weighs 3·35 lb. to the sq. ft.; No. 16 is ·065 in. thick and 2·63 lb. to the ft. Persons having no practical experience generally make their pipes unnecessarily heavy. (Rep. State Mineralogist, California.)

Riveted pipes cause eddies and consequent loss of head, and are more liable to corrosion. The best system of riveting is Ferguson's spiral. Lap welding is ordinarily too costly. In the Ferguson locking-bar system, the joint is made in the cold by hydraulic pressure, and is superior to all others.

Material economy may be secured by graduating the thickness of the metal employed to the pressure which each section of say 100 ft. of the pipe will have to withstand. With a riveted pipe, it is necessary to bear in mind the shearing effect of the rivets, as well as the tensile strength of the metal. Every pipe should be submitted to hydraulic test at double the pressure which it will be called upon to bear in work, and be rejected for the least sign of weakness. In estimating the working pressure, it is not sufficient to calculate simply the lb. per sq. in. represented by the column of water at rest; a very wide margin must be allowed for pulsation and jar—ordinarily not less than 50% extra, and in extreme cases as much as 150 to 200%. In this respect, the resisting strengths of cast iron, wrought iron and steel plate have been proved to be in the proportion of 1, 6, and 20, which fact should suffice to determine the selection of steel pipe under all circumstances.

The strains which pipes of this nature have to stand are due to internal pressure, bending, expansion, sudden shocks, and possibly also to a crushing strain arising from the creation of a vacuum. The thickness of pipe to withstand the pressure due to any head can, of course, easily be determined; but seeing that the sudden shocks which might arise from quick closing of valves and tardy action of safety appliances, cannot be ascertained with any degree of accuracy, it is well to err on the side of safety.

All pipe lines should be fitted with several spring relief-valves, mainly for the purpose of relieving the almost instantaneous rise of pressure, and consequent shock, arising from the ram action that takes place when the water issuing from the nozzles is suddenly arrested either entirely or partly. This is more likely to occur where small nozzles are used, because, however careful one may be in guarding against such an occurrence, it is almost impossible to prevent foreign substances, such as weeds, twigs, small stones, bits of wood, etc., accidentally getting into the pipe line at the reservoir end. Sometimes nozzles of only ½-in. diam. are used, and of course a very small quantity of foreign matter would entirely or partly close the orifice.

Expansion.—One of the problems in connection with laying down steel-pipe lines for power purposes is expansion. For example, with a difference of temperature of 100° F. a steel pipe 100 ft. long will lengthen ¾ in., which means that a straight pipe a mile long will extend over 3 ft. When working under ordinary conditions, with the pipe full of water, the difference of temperature is never likely to be so much as this, but 50° F. is not by any means unknown. If the pipe cannot be laid in a trench in the ground and covered with turfs (leaving only the joints exposed), it should be painted a dull white, to reflect the sun's rays, and diminish the movement due to expansion and contraction.

Some makers take up expansion in flanged-joint pipes by inserting at intervals short lengths of corrugated pipe, but these are not to be commended.

An ingeniously simple and cheap expansion joint consists of a well-soaked leather washer between two gasket rings on the outside of the pipe, and allowing this to play inside a 6 ft. length of larger pipe.

Lengths.—As to lengths, while 6–7 ft. of large cast-iron pipe is quite enough for convenient handling, 20–30 ft. of steel pipe of the same capacity can be easily manipulated, although it might not be desirable to have it all in such long sections. The fewer the lengths, the fewer the joints, and, as these are the most likely spots for leakage, a lessening of their number is advantageous. But excessive length—anything beyond ordinary railway-wagon length, for example—would be most inconvenient, and liable to result in damage during transport.

Joints.—Joints are of three principal kinds—flanged, screwed and independent. The first is the only feasible form for cast-iron pipes, the flanges being simply faced in some cases, with a plain leaden gasket ring, or an iron one and packing of tarred flannel,

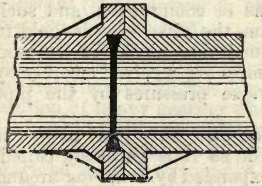


FIG. 5.
FLANGED JOINT FOR PIPES.

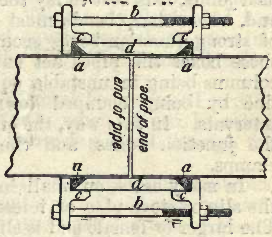


FIG. 6.
BOLTED JOINT FOR PIPES.

etc., between; or one end may be recessed or grooved to receive a corresponding bevel or projection on the other, with round rubber packing. The best of these styles is shown in Fig. 5, which is that adopted by Armstrongs for withstanding 700 lb. per sq. in. The flange is cast on the pipe, and is strengthened by brackets. Wrought-iron pipes may have a flanged joint, and be bolted together; and the flange may be attached by screwing or riveting. With a flanged joint, there is often considerable difficulty in securing tightness; a screwed joint, on the other hand, is very costly, and can be applied only to welded pipe. Incomparably the best joint in the author's experience is the independent bolted joint shown in Fig. 6. It occupies no more space than flanges; it can be made and unmade in a fraction of the time; it is not affected by expansion or contraction of the pipes; it permits a faulty length of pipe to be removed and replaced most rapidly and simply, with a certainty that the new length will fit and make an equally good joint; and it allows of quite as much bend or departure from a straight line as any pipe line should have. It is dependent upon the tarred-rope or rubber packing *a* being compressed into extremely close contact with the pipe-ends by means of the bolts *b*, the shoulders *c*, and the cast-iron ring *d*. It is made by Mephan Ferguson, Footscray, Melbourne, Victoria, and has been widely applied throughout Australia. The author used it in a most trying situation, on a column of 10-in. × 18-gauge and 24-gauge steel pipe 600 ft. high, exposed to daily alternations of temperature amounting to over 50° F., and sometimes a severe "water-hammer," without a single joint even "weeping." Very similar is the "Acme" joint of Lewis & Sons, Wolverhampton, but it necessitates flanging the pipes.

The socket or Kimberley joint is sealed with lead; each pipe length then takes up its own expansion, and the joints can be caulked or re-made easily. These joints also enable the pipe to follow slight inequalities of the ground. To ensure efficiency with lead joints, it is best to lay the pipes straight from point to point, and, where sharp bends must be made, the angles should consist of strong junction-boxes secured on rigid foundations. Between these boxes the pipes act as columns in compression, and such columns being in unstable equilibrium, the pipe must be held in line by being clamped down to foundation blocks at frequent intervals. In this way, the end pressures are received entirely by the junction boxes, and the transverse pressures by the pipe clamps.

In many cases, on small installations, the only joint needed is the slip or stove-pipe or telescope joint, as it is variously called. The larger or female end is slightly expanded by wrapping around it an absorbent material soaked in kerosene, and igniting it; the heat slightly stretches the metal and softens the pitch coating, whereupon the male end can be driven home. With pipes of large diameter, the sections are faced together by using a set of triple blocks on each side, having 2 men operating each set of blocks, and a fifth man to guide the male end of the pipe and at the same time jar it to a tight fit by using a heavy hammer or beetle against a slab of wood on the end of the pipe.

It is often practicable and desirable to divide the line into two or three different sizes. If shipped "made-up," this permits of the pipe being stacked, one section inside of another, thus economising in freight. Or pipe of this character can be cut, punched, formed, and shipped nested at dead weight, and riveted up on the ground into a continuous line. This is sometimes less expensive than to ship the pipe made up.

Pipe, Pressure in.—Pressure (lb.) = diam. (in.) squared × ·341 × height (ft.). *Ex.*—Required pressure in column of water 20 ft. high in pipe 12 in. diam. *Ans.*—12 (in. diam.) × 12 × ·341 × 20 = 982·08 lb.

Pipe, Contents.—The square of the diam. (in.) gives the weight of water (lb.) in 3 ft. length. *Ex.*—Contents of a pipe 15 in. diam. and 9 ft. long $= 15 \times 15 \times 3 = 675$ lb.

Pipe, Discharge.—To find pipe diam. required to effect given discharge at given speed—Discharge (cub. ft.) $\times$ 144 $\div$ velocity (ft. per min.); divide result by ·7854, and take out square root = diam.

Pipe Velocity.—To ascertain velocity required to discharge given volume in given time—Multiply volume (cub. ft.) by 144 and divide product by pipe area (sq. in.).

Water Wheels.

There are various types of water-wheel, according as it is desired to make use of large or small volume and great or little fall.

The height of fall (ft.) $\times$ volume (cub. ft. of water per min.) $\div$ 706 = actual brake h.p. The h.p. required $\times$ 706 $\div$ fall (ft.) = required volume (cub. ft. per min.). Volume available and h.p. required being known, h.p. $\times$ 706 and $\div$ volume (cub. ft. per min.) = height (ft.) necessary to produce the h.p. These are all calculated on 75% efficiency.

With overshot wheels, the factor 706 must be increased to 815; this means only 65% efficiency, which is as much as can be relied on after allowing for loss of power in increasing speed through the medium of heavy gearing wheels.

The old-fashioned water-wheel is, at best, clumsy and cumbrous, but in cases where the fall is less than 20–25 ft., it may be used, provided there is no scarcity of water, and that the cost of transit of so ponderous a machine is not serious; but the danger of accident to the gearing wheels, and the wear of bearings, render it out of place in most instances. It is very largely superseded by turbines, which are so much lighter, and which make so much better use of the water.

A good turbine will give 80–90% efficiency, but not more than 80% should be reckoned on in any case.

The four principal types of wheel and their applicability are:—

Undershot or breast wheels: max. head, 6 ft.; efficiency, 25–50%.

Overshot wheels: head, 5–50 ft.; efficiency, 75%.

Reaction wheels or turbines: head, 5–100 ft.; efficiency, 80–90%.

Tangential or impulse wheels, also known as Pelton wheels and "hurdy-gurdies" (U.S.): head, 50–2000 ft.; efficiency, 85–90%.

Turbines.

For making the most of the power contained in a stream of low fall there is no machine like a well-designed turbine. Generally, where the fall is low it is also inconstant, the supply varying too. Such variations need special provision to get anything like satisfactory efficiency out of half-gate (or less) flows. Yet installations are not unknown where the fall is only 2 ft., and where excellent results are obtained from even quarter-gate volume. For greatly fluctuating falls, the most constant speed without sacrifice of efficiency is probably secured with the Jonval turbine.

The annexed table, wherein efficiency is reckoned at 75%, is applicable to turbines :—

Water needed (cub. ft. per min.) for various powers under stated heads.

	Horse-Power.									
	8	10	15	20	30	40	50	60	70	80
3 ft. fall..	1883	2353	3530							
4 ,, ..	1412	1765	2648	3530						
6 ,, ..	941	1176	1765	2353	3530					
8 ,, ..	706	883	1324	1765	2648	3530				
10 ,, ..	565	706	1059	1412	2118	2824	3530			
12 ,, ..	471	588	883	1176	1765	2353	2940	3530		
15 ,, ..	377	471	706	942	1412	1884	2353	2824	3295	
20 ,, ..	282	353	530	706	1059	1412	1765	2118	2471	2824
25 ,, ..	226	282	424	565	847	1130	1412	1694	1977	2260
30 ,, ..	189	236	353	471	706	942	1176	1412	1648	1883
35 ,, ..	161	202	303	403	606	806	1010	1212	1412	1612
40 ,, ..	141	176	265	353	530	706	883	1059	1235	1412
45 ,, ..	125	157	235	314	471	628	784	941	1098	1255
50 ,, ..	113	141	212	282	423	565	706	847	988	1130
60 ,, ..	94	118	176	235	353	471	588	706	824	942
70 ,, ..	81	101	151	202	303	403	505	606	706	807
80 ,, ..	71	88	132	176	265	353	441	530	618	706
100 ,, ..	56	71	106	141	212	282	353	424	494	565

Turbines, proportions. (Cullen.)

Q The quantity of water in cub. ft. per second.

H The height of the waterfall in ft.

P The H.P. of the water at 75 per cent. .. $= \dfrac{QH}{700}$.

d The inner diameter of the wheel .. .. $= \sqrt{\dfrac{Q}{\sqrt[3]{H}}} + 1$.

N The number of buckets .. .. $= d \times 3 \times 28$.

B The breadth of shrouding .. .. $= \dfrac{d \times 55}{N}$.

s The shortest distance between two buckets $= \dfrac{B}{4 \cdot 5}$.

D The external diameter to point of buckets $= B \times 2 + d$.

A The sectional area in inches between all the buckets .. .. $= \dfrac{Q \times 60}{\sqrt{H} \times 2 \cdot 18}$.

h The height of buckets .. .. $= \dfrac{A}{N\,S}$.

b The breadth of rim for directors .. .. $= S \times 2 \cdot 8$.

r The radius for centre of directing channels $= D \times 3 \cdot 6$.

v The velocity of inner circumference for low falls .. .. $= \sqrt{H} \times 4 \cdot 4$.

V The velocity of inner circumference for high falls .. .. $= \sqrt[3]{H} \times 8 \cdot 1$.

R The revolutions of wheel per minute .. $= \dfrac{V \times 60}{d \times \frac{22}{7}}$.

U The diameter of turbine shaft in inches .. $= \sqrt[3]{\dfrac{P \times 240}{R}}$.

Note.—A $= \dfrac{Q \times 60}{\sqrt{H} \times 2 \cdot 18}$ for high falls; but A $= \dfrac{Q \times 60}{2 \cdot 08}$ for falls under 38 ft. Power is gained by extending the shroud about $\frac{1}{8}$ its breadth past the buckets when the water leaves them.

Ex.—(*a*) Given 100 cub. ft. water per sec. in a waterfall 9 ft. high, required the proportions for a turbine to be driven by 50 cub. ft. per sec., and 25 cub. ft. occasionally, to yield at least 75% efficiency under either conditions. *Ans.*—

$\sqrt{\dfrac{Q}{\sqrt[3]{H}}} + \cdot 1 = 7 \cdot 03$ ft., the interior diameter.

$d \times 3 + 28 = 49$, nearest number of buckets.

$\dfrac{d \times 55}{N} = 7 \cdot 89$ in., breadth of shrouding to point of buckets.

$\dfrac{B}{4 \cdot 5} = 1 \cdot 753$ in., shortest distance between two buckets.

$B \times 2 + d = 8 \cdot 345$ ft., exterior diameter.

$\dfrac{Q \times 60}{\sqrt{H} \times 2 \cdot 08} = 961 \cdot 53$ in., sectional area of bucket opening.

$$\frac{A}{N\ S} = 11 \cdot 175 \text{ in., collected height of buckets.}$$

$$S \times 2 \cdot 8 = 5 \cdot 806 \text{ in., breadth of rim of directors.}$$

$$d \times 3 \cdot 6 = 25 \cdot 308 \text{ in., radius for directors.}$$

$$\sqrt{H} \times 4 \cdot 4 = 13 \cdot 2 \text{ ft., velocity of inner circumference.}$$

$$\frac{v \times 60}{d \times \frac{22}{7}} = 34 \cdot 54 \text{ revolutions per minute.}$$

$$\frac{77 \cdot 14 \times 240}{34 \cdot 54} = 8 \cdot 12 \text{ in., diameter of shaft.}$$

$$\frac{11 \cdot 175}{2} = 5 \cdot 5877 \text{ in. high, first tier of buckets to pass 50 ft.}$$

$$\frac{5 \cdot 5877}{2} = \left\{ \begin{array}{l} 2 \cdot 723 \text{ in. high for second and third tiers, each} \\ \quad \text{to pass 25 ft.} \end{array} \right.$$

(*b*) Required the volume (cub. ft. per min.) and dimensions necessary to produce 34 h.p. from a fall of 99 ft. 2 in. *Ans.*—

$$\frac{34 \times 700}{99 \cdot 16} = 240 \text{ cub. ft. of water per minute, or 4 per second.}$$

$$\sqrt{\frac{Q}{\sqrt[3]{H}}} + \cdot 1 = 1 \cdot 029 \text{ ft., the interior diameter.}$$

$$d \times 3 + 28 = 31, \text{ nearest number of buckets.}$$

$$\frac{d \times 55}{N} = 1 \cdot 826 \text{ in., breadth of shrouding.}$$

$$\frac{B}{4 \cdot 5} = \cdot 406 \text{ in., shortest distance between two buckets.}$$

$$B \times 2 + d = 1 \cdot 333 \text{ ft., exterior diameter to point of buckets.}$$

$$\frac{Q \times 60}{\sqrt{H} \times 2 \cdot 18} = \left\{ \begin{array}{l} 11 \cdot 06 \text{ sq. in., sectional area of openings between} \\ \quad \text{buckets.} \end{array} \right.$$

$$S \times 2 \cdot 8 = \cdot 9288 \text{ in. for rim of directors.}$$

$$\frac{A}{N\ S} = \cdot 888 \text{ in., height of buckets.}$$

$$d \times 3 \cdot 6 = 3 \cdot 694 \text{ in., radius of directors.}$$

$$\sqrt[3]{H} \times 8 \cdot 1 = 37 \cdot 478 \text{ ft., velocity of inner circumference.}$$

$$\frac{v \times 60}{d \times \frac{22}{7}} = 715 \cdot 49 \text{ revolutions per minute.}$$

$$\sqrt[3]{\frac{34 \times 240}{715 \cdot 49}} = 2\tfrac{1}{4} \text{ in. diameter of shaft.}$$

The governing of turbines is a very important matter, and governors have recently come into use whereby the speed does not vary more than 4% should 25% of the load be suddenly taken off, 6% when the lessening is 50%, and 8% when the full load is removed. This result is attained by connecting the ordinary governor with a servo-motor which is provided with a valve acting on the gate which admits water to the turbine.

As an example, at Davos 4 turbines working under a fall of 330 ft. generate 200 h.p. each, the pipe line being 7200 ft. long. If an automatic governor was applied under such a high fall, a sudden change of load would cause the gate of the turbine to be shut very rapidly, and the sudden change of velocity of the water in the pipes would produce a shock or excess of pressure (" water-hammer ") which would be destructive of the pipes. To prevent that, a large air-vessel (4 ft. diam. and 40 ft. high) is connected with the main pipes, and a special contrivance is connected with the automatic governor, to open automatically the waste-water valve, so as to allow the water to run to waste as soon as the governor acts on the gates of the turbines. In this instance, if the full load is taken off suddenly, the normal speed of the turbine does not change more than 4%.

If water-power is used under a lower fall, say 12–50 ft., the head is not sufficient to obtain the necessary pressure for the servo-motor. Then a special pump is connected with each servo-motor attached to the governor. This pump is filled with oil, and serves two purposes at the same time—while governing the turbine it lubricates the foot-step. It is arranged so that the oil from the pump circulates in the foot-step, passing between the plates of the foot-step from the centre to the periphery at a pressure of about 350 lb. per sq. in., so that a thin film of oil practically carries the whole weight. In the instances just mentioned, the turbine-shaft being vertical and the dynamo direct-connected with the shaft of the turbine, the load on the foot-step is about 55 tons, so that provision for the lubrication of the foot-step is particularly important, and justifies the combination of the pump for the servo-motor with the lubricating arrangement of the foot-step, and using oil instead of water. The accuracy of the governor has been found so reliable that in cases where there is a large water-power, and the water is distributed over a number of turbines, each turbine is provided with its own automatic governor.

Pelton or Tangential Wheels.

The simplest of all the machines generating power from falling water is undoubtedly the Pelton or tangential wheel. In first cost, in maintenance, in speed and ease of repair, in range of size, and in applicability to high falls and utilisation of small streams, it is unrivalled.

An excellent form consists of a steel disc with buckets fastened on either side by studs or bolts. Owing to the presence of sand in all water supplies for power purposes, the buckets must necessarily suffer attrition, and in time be worn away and require renewal. A primary essential therefore in a satisfactory Pelton wheel is the utmost facility for access to the buckets, removal of worn out or damaged ones, and substitution of new ones. This is worth insisting on, because the author has had experience of Pelton wheels, made by a well-known firm, in which half the buckets were absolutely inaccessible to any known tool, broken or worn buckets could not be detached, and damaged studs and bolts could not be extracted or replaced.

By using Pelton buckets with the dividing edge and lower lip filed to a knife-edge, and the water surface of the bucket truly filed and polished, the brake h.p. is increased fully 10% over that obtained with buckets not filed and polished; this refers to cases where the greatest nozzle used does not exceed $\frac{7}{8}$ in. diam. Where nozzles $2\frac{1}{2}$–$3\frac{1}{2}$ in. diam. are used, the effect is not so great, not being more than 1 to 2%, according to size of nozzle used. Nozzles bushed with very hard steel bushes are often economical, since cast-iron nozzles may wear very rapidly, increasing the orifice at the rate of $\frac{1}{8}$ in. diam. per month, owing to excessive abrasion. (Short.)

Water under high heads is somewhat difficult to control, on account of the great pressure. Thus, at 1200 ft. head, the pressure is over 520 lb., and with a 27 in. diam. steel pipe $\frac{11}{16}$ in. thick, the bursting stress is nearly 5 t. per sq. in., giving a factor of safety under normal pressure due to the head alone of 5 to 1. Now the inertia of a column of water moving rapidly through a long pipe is so enormous that any attempt to even partially shut off the water suddenly would certainly burst the pipe. The governing of the Pelton wheel must therefore be arranged so as to avoid any shock to the pipe line.

There are three methods of governing under high heads. The first is by means of stand pipes, to which the flow of water is diverted when shut off from the wheel, relief valves being also provided. The second is to deflect the jet or stream away from the buckets, the increase in speed of the centrifugal governor throwing a nozzle-deflecting mechanism into action. In the third, recently introduced, the wheel is divided along the centre line of the buckets into two sections, and the centrifugal force developed in the rotation of the wheel body itself is arranged to cause the two sections to separate slightly. A portion of the water jet is thus allowed to pass between the buckets, instead of impinging directly against them. Being part of the wheel itself, the governing action is instantaneous; it is self-contained, and has only four moving parts.

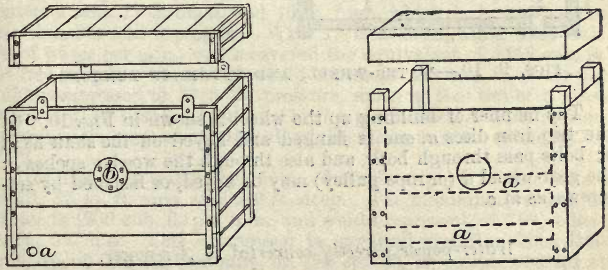
Commonly the working conditions necessitate a nozzle of 2–3 in. diam. In one example, the wheel is a steel disc $1\frac{1}{2}$ in. thick carrying 36 bronze buckets, revolving at a circumferential speed of 11,000 ft. (2·08 miles) per min., with a $2\frac{1}{4}$ in. nozzle. At Kolar, a wheel 60 in. diam., using 37 cub. ft. of water per sec., under an effective head of 382 ft., gives 1250 brake h.p. On the other hand, a 24-in. wheel on the London pumping mains, with a jet of only $\frac{3}{16}$ in. diam., works under a pressure of 900 lb. per sq. in., equivalent to a head of 2100 ft. Sometimes two nozzles are adapted to each wheel, to be used alternatively or together.

The water leaves the wheel at considerable velocity, involving very appreciable wear and tear on the tail race or pit at point of discharge. Hence, this should be lined with easily renewable plating or planking, to prevent ultimate damage to the substructure of the plant.

Small Hydraulic Motors.

There are many situations, especially on mines, where a sufficient stream of water is available for generating up to about 5 h.p. Several simple and easily-constructed motors suitable for utilising such a source of power have been described by G. D. Rice. (En. & Min. Jl., March 5, 1898.)

A wooden water-wheel box is shown in Fig. 7. It is made of $1\frac{1}{4}$-in. matched pine planks, tied by $\frac{1}{2}$-in. iron rods, with a strip of flat iron under the heads; a hole is cut at *a* for efflux of water, and

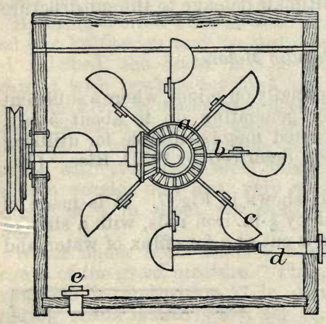


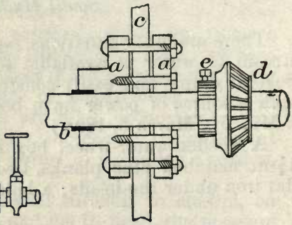
FIGS. 7, 8.—SMALL WATER-WHEEL BOXES.

in opposite sides at *b* (with iron flange bolted on for support), to carry axle of wheel; and a cover is held in place by clamps *c*.

A tinned-iron wheel-box, with internal wooden frame for bearing the wheel, is illustrated in Fig. 8. The edges of the tin box are lapped and soldered; the frame is of 2 × $2\frac{1}{2}$-in. pine, cross-pieces *a* supporting the wheel shaft.

In Fig. 9, the wheel shaft or axle is 2-in. steel, and to it is keyed a bevel-gear wheel for transmitting power at right angles; flanges are bolted on the wheel *a* to carry $1 \times 1\frac{1}{4}$ in. hard-wood spokes *b*, to which the buckets *c* are attached at the maximum distance giving clearance in the box; buckets of the shape shown are approved, and can be bought of almost any size; water is admitted by nozzle *d*, a $\frac{7}{8}$ in. jet, at 65 lb. per sq. in. pressure and 2 ft. radius developing about 4 h.p.; the outlet is at *e*. Obviously the power can be transferred from *a* for parallel driving by a rope pulley, with or without cog-gearing, instead of the bevel-gear; and the jet may be vertical, or lateral, or both.





FIGS. 9, 10.—WATER-WHEEL, AND DETAIL OF BUILDING.

The manner of building up the wheel is shown in Fig. 10. Of the two iron discs *a*, one is flanged and keyed on the shaft as at *b*; bolts pass through both, and also through the wooden spokes *c*: the gear-wheel *d* (or rope pulley) may be keyed, or fastened by set-screws, as at *e*.

Water-power directly converted to Air-power.

Air compressed by the ordinary mechanical methods contains at least the same amount of moisture as the surrounding atmosphere from which it was compressed; and, in parting with the heat necessarily contributed to the air by the mechanical compression, it is inclined to absorb more moisture. There is incidentally a considerable loss of energy in parting with this heat. Air compressed directly by falling water is kept at the same temperature as this water. It is compressed isothermally, and the consequent expansion, when used in motors, produces an almost truly adiabatic expansion line. Tests, however, have shown that air compressed in this manner contains only one-sixth of the moisture originally in the surrounding atmosphere from which it is compressed. This is probably because the moisture in the bubble of air is pressed or squeezed out to its surface, and then absorbed by the surrounding water. Incidentally there is no loss of power in parting with any heat, and there is a practical result which is of more importance—the hydraulically-compressed air can be expanded down to a temperature much below the freezing point, while atmospheric air, with the usual amount of moisture, mechanically compressed, cannot be used at all, owing to the freezing up of the exhaust passages of the motor in which the attempt to use it is being made. Late improvements on this device have been in the method of introducing the air into the mouth of the downwardly flowing water column, so as to ensure the largest proportion of air being taken down with the water, and in methods of decreasing the velocity of the combined air and water at the bottom of the descending column, causing the water to part more readily with the air, the water then passing out at the bottom of the enlarged chamber into an ascending shaft, maintaining upon the air a pressure due to the height of water in the uptake, the air being led off from the top of the enlarged chamber by means of a pipe. (Webber.)

Examples.—(*a*) Head of water, 22 ft.; down-flow pipe 44 in. diam. extends downward through a vertical shaft 10 sq. ft. area and 128 ft. deep. At the bottom of the shaft the compressor pipe enters a tank 17 ft. diam. and 10 ft. high, which is known as the air chamber and separator. With 19·5 ft. head, using 4292 cub. ft. of water per min., was recovered the equivalent of 1148 cub. ft. of free air per min., which would represent 248 cub. ft. of air per min. compressed to 53·3 lb. pressure, showing that out of a gross water h.p. of 158·1, a total of 111·7 h.p. of effective work in compressing air was accomplished, giving therefore an efficiency of 71 %.

(*b*) Head of water, 107·5 ft.; down-flow pipe, 33 in. diam.; shaft, 32 sq. ft. area and 210 ft. deep. The maximum volume of water is 4200 cub. ft. per min., and would represent, at 71 % efficiency, 587 h.p. This compressor is expected to utilise about 5100 cub. ft. of free air per min., or 734 cub. ft. of compressed air at 87 lb. pressure, and give an air motor equivalent of 360 h.p.

(*c*) Head of water, 18½ ft.; diam. of shaft, 24 ft.; diam. of compressor pipe, 13 ft.; depth of shaft, 208 ft. The air is transmitted a distance of 4 miles, with a loss in transmission not exceeding 2 %, through 16-in. pipe, laid with flanged joints and rubber gaskets. The plant gives 1365 h.p. of air at a pressure of 85 lb. per sq. in.

(*d*) Head of water, 45 ft.; there is no shaft, the plant being built against the vertical wall of a cañon; compressor pipe, 3 ft. diam.; upflow pipe, 4 ft. 9½ in. diam.; height, 269 ft.; flow, 53·2 cub. ft. per sec.

(*e*) Operating at Monteponi, Sardinia, on a waterfall of about 93 ft. The apparatus (Fig. 11) consists of 2 cylinders into which alternately water under pressure enters from above, compresses the air in the cylinders, and forces the compressed air into a pipe above. When one cylinder has been filled with water, the water is discharged automatically, while free air enters to take the place of the water, and to prepare for a new action of the apparatus. The distribution of the water is regulated by two floats, which move in the domes placed on the cylinders. These floats are connected by a balance-lever and by two rods which pass through stuffing-boxes on the domes. A Watt parallelogram guides the

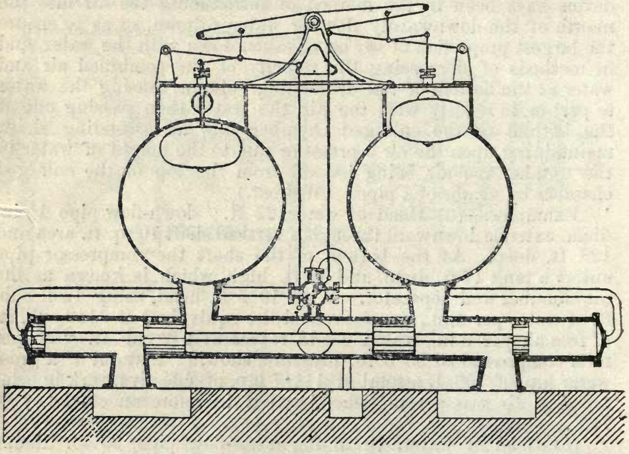


FIG. 11.—HYDRAULIC AIR COMPRESSOR, MONTEPONI.

rods, while two levers carry two vertical rods arranged to operate a four-way valve placed on the compressed-air pipe. This pipe leads the water into a transverse pipe which joins the two cylinders below. Four pistons are placed on one piston rod, which moves in the pipe serving to open the air inlet into one cylinder, while at the same time they permit the flow of water from the other by the connections, and by the lower openings into the free air. To move the rigid system of four pistons to the right or left as required, and to stop the motion of one of the ports, the two extremities are connected by pipes with the two lateral branches of the four-way valve. In this way, one end of the distributing pipe is placed in communication with free air, while the other communicates with air under pressure. When one cylinder is full of water, and the water from the other has run out, the float in the first one will be raised, and its movement will cause the parallelogram to move also, and the latter will give a quarter turn to the four-way valve. The communications with the distributing tube being thus inverted, the system of four pistons will be forced in the opposite direction, and the water under pressure can then enter the empty cylinder, while the water in the other cylinder escapes. Air will enter into this cylinder by the upper air valve, and the compressed air will pass through the pressure valve of the other cylinder and will pass

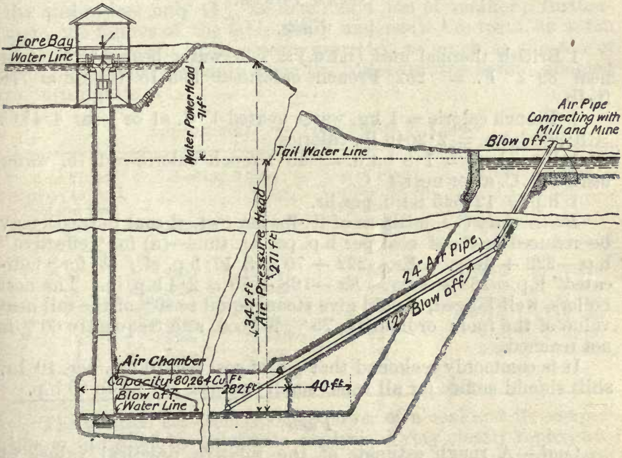


FIG. 12.—HYDRAULIC AIR COMPRESSOR, VICTORIA MINE, MICH.

off through the conduit. This system is very simple. It presents no dead-points, and very little loss by friction. It will compress the air to a point almost equal to the pressure or head of water available. This large surface of water absorbs the heat evolved in compressing the air, so that it is not necessary to cool the valves. It will work with very little attendance or supervision, and is very cheaply operated. Moreover, the first cost of the installation is low, since it replaces the reservoirs for compressed air required in other plants; while no expensive buildings or foundations are needed. (Ferraris, En. & M. Jl , July 13, '01.)

(*f*) A direct conversion of water power to air-pressure, whereby over 82% of the natural power of a waterfall is delivered as compressed air at 117 lb. per sq. in., without any machinery, is described by Woodbridge (En. & M. Jl., Jan. 19 '07). The installation, designed under a guarantee of 4000 h.p. at 70% efficiency, cost less than 92*s.* per h.p., and, allowing 5% interest on first cost, the cost per h.p. per ann. is just under 9*s.* 5*d.* The necessary water is diverted from a river, and affords a net fall of 71 ft. By means of an underground chamber and various pipes (Fig. 12), air is compressed and applied to running drills, pumps, battery, etc., at the Victoria mine, Mich.

STEAM POWER.

Units.

1 British thermal unit (b.t.u.) = 1 lb. water heated 1° F. at or near 39·2° F. = ·252 French calorie = ·555 lb. calorie = 778 ft.-lb.

1 French calorie = 1 kg. water heated 1° C. at or near 4·4° C. = 3·968 b.t.u. = 2·2046 lb. calories.

1 lb. calorie = 1·8 b.t.u. = ·45 French calorie = 1 lb. water heated 1° C. at or near 4° C.

1 h.p. = 12·545 b.t.u. per hr.

Boiler duty (i.e. millions of ft.-lb. per cwt. of coal burned) may be reduced to lb. of coal per h.p. per hr. thus—(*a*) for "effective" h.p.—222 ÷ duty. *Ex.*—222 ÷ 70 = 3·17 h.p. ef.; (*b*) for "indicated" h.p.—168 ÷ duty. *Ex.*—168 ÷ 70 = 2·4 h.p. in. The best boilers, well lagged, should give steam equal to 80% of the full heat value of the fuel; ordinarily, 75%, is good, and frequently 60% is not reached.

It is commonly reckoned that ½ t. of coal per 10 h.p. per. 10 hr. shift should suffice for all usual small engines, say up to 80 h.p.

Fuels.

Coal.—A rough estimate of the relative practical values of several classes of coal may be calculated as follows:—

—	Moisture. %	Ash. %	Combustible. %	b.t.u. per lb. combustible.	Theoretical Heating Value.	Efficiency of Boiler.	Relative Practical Value.	
							b.t.u.	Semi-bit. = 100.
Anthracite.. ..	2	13	85	14,800	12,180	77	9,379	88
Semi-bituminous	2	8	90	15,800	14,220	75	10,665	100
Bituminous (*a*)..	2	8	90	15,000	13,500	70	9,450	89
Bituminous (*b*)..	10	15	75	14,200	10,150	65	6,598	62
Lignite	15	20	65	12,000	7,800	60	4,680	44

The figures in the column headed "combustible" are obtained by subtracting the sum of the percentages of moisture and ash from 100. These multiplied by the average heating value per lb. of combustible give the theoretical heating value of the coal. The figures for boiler efficiency are high as compared with ordinary practice, but they may be realised with properly designed furnaces and with skilful firing. These, multiplied by the theoretical heating values, give the figures for relative practical value of average coals of the several classes, which are converted into percentages in the last column. The table shows that coal is a substance so varied in quality that a ton of one kind may be worth to the steam user only 44 % as much as a ton of another; furthermore, the figures of the table really understate the truth, as a ton of lignite is worth to the steam user 44 % as much as the ton of semi-bituminous coal only when the boiler furnace is well adapted to burn lignite.

Approximate Heating Value of Coals.

% of Fixed Carbon in Coal, Dry and Free from ash.	Heating Value		% of Fixed Carbon in Coal, Dry and Free from Ash.	Heating Value.	
	Calories.	b.t.u.		Calories.	b.t.u.
97	8300	14,940	63	8400	12,120
94	8450	15,210	60	8200	14,760
90	8600	15,480	57	7900	14,220
87	8700	15,660	55	7700	13,860
80	8800	15,840	53	7400	13,320
72	8700	15,660	51	6900	12,420
68	8600	15,480			

The relation between calorific power of a coal and its composition as indicated by proximate analysis is very closely represented by the formula, $P = 82\,C + a\,V$, in which P is the calorific power, C the percentage of fixed carbon, V the percentage of volatile matter and a is a variable coefficient, which is dependent upon the tenor V^1 in volatile matter of the pure fuel, that is, the fuel minus ash and moisture.

$$\left(V^1 = 100\,\frac{V}{C + V}\right)$$

The values of the coefficient a corresponding to different values of V^1 when plotted differ but little from a straight line, the values being as follows :—

V^1	5 %	10 %	15 %	20 %	25 %	30 %	35 %	40 %
a	145	130	117	109	103	98	94	80

In the case of anthracite $a = 100$. The average calorific power of anthracite is 8250 calories. The calorific power rises with the content of volatile matter to the maximum of about 8700 calories when V is between 10 and 30 %, and then falls as the percentage of volatile matter increases further. The formula was deduced from calorific determinations of 600 coals of different kinds, and gives results which in nearly all cases agree within 1 %. (Goutal.)

All coal, therefore, should be examined for carbon, ash, and volatile matter, and be purchased only on its heat value basis as determined by analysis.

Coal invariably deteriorates by storing, especially if exposed to the weather. Good bituminous coal will lose 6 % by lying in the open air for 2 years. Some coals are liable to spontaneous combustion when wetted; others "slack" or fall to powder under the influence of air; and others again intumesce, forming channels and crusts, and cannot be used in gas producers.

Sometimes, with coal producing an abundant ash, the ash carries much unburned coke, which may be jigged out, amounting to as much as 5 %; this can be burned quite well with forced draught.

Great saving may be effected in pulverising coal for mechanical stoking; with equal evaporation it may reach 29 %, and with equal amounts of coal the increased evaporation may be 48 % in favour of pulverised coals.

Coke.—Coke should be tested for moisture, as well as for carbon, ash, and volatile matter; also for sulphur, in some cases. Some coke is too dense, some too friable, and some too ashy (12–19 %).

Wood.—Perfectly dry wood, carrying only 2 % ash is valued at about 7800 b.t.u.; wood with 25 % moisture = 5800 b.t.u. Generally it is computed that 2½ t. dry wood = 1 t. coal; or ·4 t. coal = 1 t. wood. Hard woods are generally superior to soft. The Cape Copper Co., in Namaqualand, use thorn (mimosa), costing 20–25*s.* per long ton, and the consumption averages 6 t. per 24 hr. for 80 h.p. = about 7 lb. per h.p. per hr., or say 2½ t. wood = 1 t. good coal.

Fuels, Miscellaneous: Heating Values.

	b.t.u.
Peat, completely dry, 4 % ash	10,200
„ air-dried (25 % H_2O), 4 % ash	7,400
Tan-bark, completely dry, 15 % ash	6,100
„ 30 % moisture	4,300
Straw, dry, 4 % ash	6,300
„ 10 % moisture	5,450
Liquid hydrocarbons	18,000–20,000
Lignites	8,000–11,000
Wood charcoal	14,500
Hydrogen	62,000
Marsh gas	23,500
Olefiant gas	21,300

Oil.—The heating value of ordinary mineral oil (liquid hydrocarbons) is about 21,000 b.t.u. per lb. (including the latent heat in the steam formed by burning the hydrogen), or about 45 % more than that of pure carbon. In ordinary locomotive boilers, 1 lb. petroleum will evaporate 15 lb. water (as against 1 lb. coal = 7½ lb. water), and in special boilers as high as 18·95 lb. has been reached. For locomotive work, steam is more easily produced and is maintained up the steepest gradients, and great economy is effected by reducing the supply of oil when descending or remaining stationary; the life of the boilers is prolonged, inasmuch as the tubes do not foul; the nuisance of smoke and the danger of sparks are entirely obviated.

Most crude oils, having been obtained from wells, carry small proportions of water, as well as more or less sand or grit. The heavier and more viscous the oil, the greater the tendency there is to hold in suspension these deleterious substances. It can be assumed that no crude oil is perfectly clean. Therefore, in the installation of any oil-burning plant, special provision should be made for straining out all foreign matter. Arrangements should be provided for catching the water as it slowly settles to the bottom of the tanks. Sand increases the wear on the small annular nozzles of the burner, or, when using burners provided with specially small orifices, these may become altogether clogged.

The gauze used for oil strainers should be formed of brass meshwork equal to about half the width of oil orifice in the burner. These strainers are not unusually placed on the oil pipe on each side of the pump, thus ensuring that no grit gets into the pump, and that any particles of old packing or other material from the pump cannot go to the burner through the last filter. A still more desirable plan is to have two strainers between the pump and the burners, so that, when one filter is being opened, the current of oil can be transferred by means of a pass-by valve through the other filter.

There is no practical device that will directly separate the water from the oil. This can only be satisfactorily effected by allowing the water to settle to the bottom of the tanks by gravity. A thread of water blown into the oil burner effectually extinguishes the flame in the furnace, and, if the oil does not soon follow the water, there may be difficulty in relighting without introducing an outside flame. In order to ensure a supply of oil without admixture of water, the oil-suction pipe is caused to swing up or down in the oil tank to a level at which it is known that pure oil can always be obtained; or the movable oil-suction pipe may be carried by a float. The suction tip is thus always maintained at a point within a few inches of the top of the oil. The tip is also surrounded by a steam-heating coil, the object of which is to slightly heat the oil in proximity to the inlet, thus increasing the fluidity of the oil, so that even in cold weather it may be readily pumped. At the bottom of each tank there should always be provided a cock for the purpose of blowing off any water which may have settled.

Certain crude oils at the ordinary temperature of the atmosphere are of great viscosity, which increases as the temperature gets lower. At 30°–40° F., which is not an unusual outdoor temperature, the fluidity of the oil is so slight that it is almost impossible to pump it, or force it to the burner. It is therefore necessary in many regions that there should be means of heating the oil. In all pipes intended for transmission of crude oil, connections should be made to enable steam to be turned into them after shutting off the oil. They can be thus cleaned by the heat and the force of the blowing steam, and any deposited asphalts, paraffins or condensed hydrocarbons can be cleared out before the pipes become choked so as to impair their efficiency.

The heating of the oil should never be carried to such a temperature as will cause decomposition of the hydrocarbons.

With most burners it is desirable that a uniform pressure should be maintained on the oil circuit. A reliable plan is to provide the oil chamber of the pump with what would correspond to an air chamber on a water pump, or to provide a separate tank or chamber in which a constant air pressure is maintained on top of the oil by additional means.

In all oil installations, it is very important that the control of the oil and of the steam or compressed air should be so arranged that in case the delivery of any one is reduced or interrupted, a corresponding reduction or shutting off should be effected in the supply of the other elements. It is especially important that oil should in no case continue to be forced or pumped to the burners when the steam or air required for spraying is shut off, as in such an event the unsprayed oil is liable to flood in upon the hot brickwork, and a furnace explosion is likely to occur.

Boilers.

Duties.—The practice of quoting boiler duties in "h.p." is not sound, because the h.p. must depend on how the steam is utilised, i.e. in what kind of engine. It is much better to have a guarantee of its capacity in lb. of steam raised per hr. at given pressure (say 70 lb. per sq. in.) from stated fuel.

The unit of commercial h.p. developed by a boiler is usually taken as 34½ units of evaporation per hour; that is, 34½ lb. of water evaporated per hour from a feed-water temperature of 212° F. into dry steam of the same temperature. The standard is equivalent to 33,317 b.t.u. per hr. It is also practically equivalent to an evaporation of 30 lb. of water from a feed-water temperature of 100° F. into steam at 70 lb. gauge pressure.

A boiler rated at any stated capacity should develop that capacity when using the best coal ordinarily sold in the market where the boiler is located, while fired by an ordinary fireman, without forcing the fires, while exhibiting good economy. And further, the boiler should develop at least one-third more than the stated capacity when using the same fuel and operated by the same fireman, the full draft being employed and the fires being forced; the available draft at the damper—unless otherwise understood—being not less than $\frac{1}{2}$ in. water column.

The usual practice of boiler makers is to allow 10–12 sq. ft. heating surface and 33 sq. ft. grate surface to 1 h.p. The kind of fuel used would make an important difference as to grate surface.

Many conditions modify calculations in determining the h.p. of a boiler, such as design, quality of material, facilities for cleaning, impurities in the water, calorific power of fuel used, attendance, and draught of chimney, etc. So also the factors of grate and heating surface, temperature of feed water, proper control of air to support combustion, steam and water space, circulation of water within the boiler, all vary with each type of boiler used, and even with different proportions in the same type.

Approximately the h.p., if externally fired, may be calculated thus—$\frac{2}{3}$ circumference of shell (in.) $\times$ its length; then the product of the area of all the tubes or flues $\times$ their length; add these products together $\div$ 144 to find sq. ft.; then $\div$ 14; 144 represents the sq. ft. of heating surface in the shell, tube and flues, and 14 the sq. ft. of heating surface usually allowed per h.p.

For internally fired boilers, the circumference or square of furnace (in.) $\times$ its height; then the circumference of one tube $\times$ its length, and this product $\times$ the entire number of tubes, taking into account also the sq. in. of surface presented by the crown or tube sheet. Add these quantities together and $\div$ 144; the quotient will be the heating surface in sq. ft.; and this $\div 14 =$ h.p.

With a good draught for the furnace, 10 sq. ft. of heating surface, $\frac{1}{2}$ sq. ft. of grate surface, 8 lb. of good coal, and 1 cub. ft. of water evaporated per hr., may be estimated for each nom. h.p. that the boiler should develop.

For cylindrical boilers, each nom. h.p. requires 1 cub. yd. capacity, 1 sq. yd. heating surface, 1 sq. ft. grate-bar surface, 1 cub. ft. water evaporated per hr., and 28 sq. in. of flue area; the area for entrance of air to ash pit should be $\frac{1}{4}$ in. of grate area; depth of ash pit, 18–30 in.

The grate-bars should incline downward towards the front, 1 in. per ft.; not over $\frac{3}{4}$ in. thick and $\frac{3}{8}$ to $\frac{7}{16}$ in. spaces between.

The furnace should have at least 3 cub. ft. of space above each sq. ft. of grate surface.

For horizontal tubular boilers, length of tubes should be 48 times the diam. For soft coal, 4 in. diam. gives the best results, with 1 in. space between the tubes, and $2\frac{1}{2}$-in. space between the tubes and shell of boiler. Tubes should be set in vertical rows to facilitate cleaning.

Firing.—Furnace construction must be adapted to the fuel, and it is absurd to suppose that furnaces designed for burning anthracite or semi-anthracite fuel, will satisfactorily burn fuels containing 30 % or more of volatile matter. The percentage of volatile matter which a fuel gives off when heated is, in fact, a measure of the size of the chamber required for its complete combustion, and it is here that the laboratory examination of fuels yields results of the greatest value.

In ordinary firing with any class of boiler, the air is for the most part admitted to the furnace through the air-spaces between the fire-bars. The layer of coal should be of uniform thickness and not too thick, and the clinker should not be allowed to obstruct the air-spaces. The fire should be fed at short intervals with correspondingly small quantities of coal, instead of allowing it to burn down low before throwing on a large quantity, thus lowering the furnace temperature by the abstraction of the heat required to gasify the volatile constituents of the new supply. The lowering of the temperature leads to a reduction of the chimney draught at the exact time when the highest temperature and the greatest admission are required to effect the combustion of the hydrogen, which, as a consequence, passes away unconsumed, having added nothing to the useful heat of the furnace. A portion of the carbon, also, which at the moment of throwing on fresh coal was floating about in the furnace at a high temperature in search of oxygen to combine with, is cooled down below the temperature necessary to its combustion, and passes wasted away into the atmosphere, either in the form of smoke or combined with the hydrogen as olefiant gas.

The most important causes of low initial furnace temperature are excessive air supply to the furnaces, and too sudden contact of the half-burned gases with the water-cooled tubes or plates. Larger combustion chambers and refractory furnace linings are the proper remedy for the latter evil, and gas testing is the check and remedy for the former.

If in an 8-ft. Lancashire boiler, 12 cwt. coal per hr. be used, it is obvious that ½ cwt. of this should be put in each flue every 5 min. If, as in hand firing, 3 times this amount be put on every 15 min., a very irregular production of steam is the result, and probably a very regular production of smoke, because 1½ cwt. coal are put on a fire which has only the same amount of air going through the bars each minute, or, rather, somewhat less than it had before it received this charge of coal.

It is more economical to work boilers at their full capacity and to have as few boilers at work as possible. Not only are radiation losses thus minimised, but the heat-absorbing powers of the remaining boilers are improved.

Incomplete combustion, radiation, and loss of heat in the chimney are responsible for about 38% of the theoretical heat value in a boiler working alone.

Lagging.—Insufficiency or lack of boiler, steam-pipe and feed-water pipe covering is a fertile source of loss. Experiments have shown that each sq. ft. of uncovered steam or boiler surface wastes on an average 10 cwt. coal per ann. An excellent lagging consists of chopped straw and clay or dung laid on 4–5 in. thick. A wrapping of old gunny bag will hold this on to pipes.

Types.—As to the best type of boiler for the economical production of steam, it may be taken for granted that all surviving well-known types of boilers are capable, each in its own way, and for its own purpose, of effective steam raising if properly handled. Of these, the principal in general use are the Lancashire (double-flued cylindrical) boiler, the Cornish (single-flued); the water-tube, in various forms; the loco-type; and the internally-fired, return-tube boiler. The smaller kinds of vertical boiler, and small boilers of any kind, as a rule, are installed with other objects than fuel economy. Externally-fired cylindrical boilers of all kinds are out of the running, on account of the inevitable deposit of scale on the bottom just where it is exposed to the greatest heat.

The Lancashire boiler, with addition of a good steam dome as a preventive against priming, due to excessive water-feed by careless native stokers, is about the best all round for mine work.

The old-fashioned return-tube boiler, for moderate pressures, and with the exercise of ordinary intelligence in the choice and care of its setting, grates, etc., will do just as good work as any water-tube boiler, and often better than many of them.

No class of stationary boilers is operated uniformly under such high pressures as are loco-type boilers, nor do any boilers evaporate as much water. With the strong draught due to the exhaust steam entering the stack, a tremendously high rate of combustion is possible, and it is customary to allow only 2–2·5 sq. ft. of heating surface per h.p., while 10 sq. ft. is a moderate allowance for stationary boilers.

Water-tube boilers are not much used on mines, as the circumstances are not very suitable to them. With high-speed engines and good water, they may be used in connection with mills and electric stations, but for winding purposes they are quite out of place.

For mining work it pays in the end to have extra boilers, as a stand-by during cleaning and repairs, and for sudden emergencies in extra pumping or baling. No boiler should run more than 3 months without examination.

Care of.—Boilers should be tested to a pressure half as much again as that to which they are to be worked. Though hydraulic tests are more severe, even with warm water, than steam pressure tests, they are not sufficient by themselves to ensure against explosion, especially in the case of bricked-in boilers. They should be supplemented by thorough internal inspection, with plenty of hammer testing for impoverished plates. Bottom shell plates beneath the fire box should be exceptionally carefully examined, owing to their liability to corrosion from moisture and leakages from gauge-cocks, etc. Ashes should not be piled in front of the boiler, as the gases thus produced are stated to be capable of destroying a ⅝ in. plate in less than 18 months. Top blow-offs should be totally avoided, since they are likely to render new boilers dangerous through corrosion of the plates by the slime or silt, which is always left behind, and which accumulates round the bottom of the pipe if ending vertically, or along the sides of it if it is a perforated longitudinal one. They should be replaced by a bottom blow-off. The boiler should be blown out every 24 hr., and the water-gauge every hour. Safety-valves, owing to their being made of one pattern, and fitted to the boiler irrespective of the working pressure, are defective from the commencement, more often than not; they should not be set or checked by the pressure-gauge, since the latter has always an error which fluctuates with the pressure. It ought to be the other way about, the gauge being checked by the valve, and annually tested against mercury. The valve should be lifted every 24 hr., and should dance on the cushion of steam before reseating itself after being released.

Government boiler inspection is insisted on in some countries, and generally performed in the most perfunctory manner. Hence, the English system of an inquiry after an accident works better than any compulsory inspection by Government officials, and has made boiler owners careful.

Mechanical Stokers.—Frequently, cheap fuels, perhaps rather small, from one colliery, contain nearly, or quite, as much heat as other fuels of larger size and much more expensive, from another, or even from the same colliery. It makes little or no difference to a mechanical stoker what the size of the fuel may be—it will burn one as efficiently as the other: in fact, it prefers fine coal. A few examples from English experience may be given: (*a*) Rough slack used in hand firing contained 10,698 b.t.u. per lb. and cost 8*s.* per ton; fine slack, only possible in mechanical stoking, contained 12,070 b.t.u., and cost 6*s.* 9*d.*—economy 37 %. (*b*) Hand firing coal 10*s.* 6*d.* per ton: smudge burned mechanically, 4*s.*—economy, 57 %. (*c*) Economy, 28·8 %. (*d*) Economy, 23 %. In many situations it means permitting the local inferior fuel supply being availed of, as against importing English coal at great cost. On the other hand, mechanical stokers are costly at first, and are as liable as other machinery to suffer from careless handling, and they do not always pay to instal. A coal which cakes and arches, or which clinkers much, is apt to cause much trouble with them.

Forced Draught.—With some very low-class fuels, forced draught, by means of large slow-speed fans handling ample volumes of air at low pressure, is a necessity; and when such fuel has a tendency to clinker, steam blowers may be used intermittently.

Forced draught, as opposed to natural (chimney) draught, permits an increased efficiency through diminished grate area and accelerated combustion, hence better economy.

Liquid Fuel.—To obtain the greatest efficiency from oil fuel, it should be burned in a confined combustion chamber, so as to obtain the highest possible temperature. To do this, the chamber must be of a refractory non-conducting substance, which soon becomes heated to incandescence; and all gases, together with the incoming air, pass through this focus of heat. The heated brick-work is of still greater use in ensuring perfect continuity of the heat supply, especially when burners tend to act in gusts, as they often do under improper action of the pumps, or where there is dirt or water in the oil supply. When the oil is injected with a steam jet, better results are obtained than when air is used as the spraying medium; and the steam-injected oil seems to have a much softer flame, and to be easier on the furnace plates.

Feed Water.—Water is generally described as being soft or hard. It is called hard when it contains considerable quantities of the salts of calcium and magnesium in solution; but when only a small quantity of these salts is present, it is called soft. The chlorides, sulphates, and nitrates of calcium and magnesium are easily dissolved and maintained in solution by water, but the carbonates of these elements can only be maintained in solution by an excess of carbonic acid in the form of bi-carbonates.

The following distinctions are made with regard to hard waters:—

(1) Temporary hardness, that is hardness caused by the bi-carbonates of the alkaline earths, and which disappears in boiling.

(2) Permanent hardness, that is, hardness caused by the chlorides, sulphates, and nitrates of the alkaline earths, which is not lessened by boiling. The sum of the temporary and permanent hardness is the total or aggregate hardness.

In order to express the relative hardness of different waters, the following measurements have been adopted:—

England	..	1 grain of calcium carbonate ($CaCO_3$) per gallon of water.
France	..	10 milligrams of calcium carbonate ($CaCO_3$) per litre of water.
Germany	..	10 milligrams of calcium oxide (CaO) per litre of water.

Calcium, in the form of bi-carbonates and sulphates, is the chief constituent of the dissolved mineral matter in hard water, whilst it occurs also in small quantities as chlorides, nitrates, and nitrites. Next in order comes magnesium in the same combination as calcium. The bi-carbonates of iron and manganese, and the carbonates, chlorides, sulphates, nitrates, and silicates of sodium and potassium, are rarely absent, but seldom occur in large quantities. Water also absorbs oxygen, nitrogen, and carbonic acid, and occasionally sulphuretted hydrogen may be found in it. Organic matter occurs in some lake and river waters. Carbonic acid enables water to dissolve substances, such as the carbonates of calcium, magnesium, iron, and manganese (which pure cold water could only dissolve with great difficulty, in minute quantities, if at all), by converting the carbonates into bi-carbonates, the bi-carbonates being soluble in pure water. It has been found from experience that scale over the heating surfaces of a boiler to the thickness of $\frac{1}{16}$ in. will cause a waste of about an eighth of the efficiency of the boiler, and the waste increases as the square of the thickness. The amount of incrustation varies considerably with the quality of water, and with the regularity with which the operations of blowing through and cleaning out are practised. Occasionally vegetable matter of a glutinous nature, held in suspension by the feed water and precipitated by heat or concentration, covers the heating surface with a thin coating almost impermeable to heat, and which hardens the mineral deposit, so that it is next to impossible to remove it, and hence causes over-heating.

Softening.—The Porter-Clark process is based on the following theory: When ordinary burnt lime or calcium oxide is added to water which contains the bi-carbonates of lime or manganese, the excess of carbonic acid necessary to keep the lime or magnesia in solution in the form of a bi-carbonate combines with the calcium oxide, forming insoluble carbonate of lime, both by the decomposition of the bi-carbonate of lime, and by the combination of the excess of carbonic acid given off in decomposition with calcium oxide added to the water.

In this reaction the carbonates only are precipitated. The sulphates are precipitated by using a small quantity of soda, Na_2CO_3. The sodium carbonate and calcium sulphate mutually exchange their acids, forming insoluble carbonate of lime and soluble sulphate of soda. The latter salt is so very soluble that it is not precipitated even after much concentration upon evaporation, and thus it flows out of the boiler when the mud-plugs are removed for washing-out.

A little alum is mixed with the soda to aid the precipitation of solid matter held in suspension by the water; it also materially assists the precipitation of the salts in the precipitating tanks.

The reaction is represented by the following equations:—

$$CaSO_4 + Na_2CO_3 = CaCO_3 + Na_2SO_4.$$
$$H_2Ca\,(CO_3)_2 + CaO = 2\,CaCO_3 + H_2O.$$
$$H_2Mg\,(CO_3)_2 + CaO = CaCO_3 + MgCO_3 + H_2O.$$

The quantity of lime required to soften 1000 gal. of water is 2·24 lb.; soda 4·5 lb.; and alum ·1 lb. The above quantities give very satisfactory results. It is not found necessary to reduce the degree of hardness lower than 6° and 7°, as the heating surfaces are kept practically clean when water of this degree of hardness is used. The water, after being softened, dislodges old incrustations and deposits from boilers previously using hard water. (W. W. F. Pullen, Proc. Inst. C.E., xcvii. 354.)

On the basis that 1 lb. coal at 8*s.* per ton burnt in the furnace of the boiler will convert 9 lb. water into steam, the cost of evaporating 1000 gal. water would be 4*s.*, whereas the cost of the water, even including softening, should not exceed 6*d.* per 1000 gal. As an average statement, it is safe to say that the use of water of 20° hardness will cause a loss of 15–20 % in fuel; that is to say, the decrease in the efficiency of the boiler due to scale formation, more frequent blowing off, increased repairs, etc., will be about 20 %. This means an increase in the cost of evaporation of about 1*s.* per 1000 gal., whereas the cost of softening (including interest on plant) should not, in an extreme case, exceed 3*d.* per 1000 gal. From the point of view of steam raising, the softening of water is thus well worth consideration.

Various forms of apparatus are in the market. The combined purifier and filter made by the Humboldt Engineering Works is very good.

The cost of installation may range from 20*s.* per h.p. (up to 1000 h.p.) down to 5*s.* per h.p. (in plants of 15,000 h.p.). Chemicals vary from ½*d.* to 2½*d.* per 1000 gal., according to hardness, and commonly are ½–1*d.* Attendance and power should be almost nil. Economies resulting may reach as high as 56 % of the fuel bill.

Heating.—In the conditions of ordinary practice it is computed that a saving of 1 % in fuel is made for each increase of 11° F. in the temperature of the feed-water. The heating medium may be (*a*) live steam, taken direct from the boiler, or (*b*) exhaust steam, which might otherwise go to waste.

At first glance, it would seem impossible to gain anything by taking steam from a boiler to heat the feed-water for that boiler, except in the matter of purification, and the prevention of stresses due to unequal expansion. The idea of a thermal gain, or saving of fuel, would seem preposterous, inasmuch as all the heat taken from the boiler would be returned thereto, neglecting radiation losses, leaving the heat balance just the same. Yet actual economy has been proved by the use of live-steam heaters. The explanation lies in the fact that the hot feed-water is better able to absorb heat rapidly than the colder water, while at the same time the circulation is augmented, which further tends to increase the rapidity of heat transmission. The prevention of heavy scale formations also facilitates the heating and reduces heat losses.

A great consideration is safety. It is no longer a matter of doubt as to what effect cold feed-water has upon a boiler, especially one carrying high steam pressure. Engineers have come to recognise the fact that it is extremely poor practice to introduce comparatively cold feed-water into a boiler whose plates are at a temperature of 100°–200° higher than that of the water. The contraction and expansion set up by changes of temperature hasten the deterioration of the shell, and weaken it, by the well-known principle of fatigue of metals, and contrary to all ideas of safety.

Roughly 1 lb. of exhaust steam at atmospheric pressure possesses sufficient heat to raise the temperature of 6 lb. of water from 50° F. up to 212° F. When this operation is performed in an open heater, the resultant is 7 lb. of boiling hot water, because, in the open heater, the 1 lb. of condensed steam is conserved and utilised as part of the boiler-feed supply. When the operation is undertaken in a closed heater, the resultant is only 6 lb. of hot water for the boilers, because in a closed heater the condensation of the steam used in accomplishing the heating is wasted, as it is contaminated with oil. Therefore a given quantity of exhaust steam will heat more water to a given temperature in an open heater than in a closed heater; or, with a limited supply of exhaust steam, the feed water can be heated to a higher temperature in an open heater than in a closed heater, and consequently an open heater will give the greater fuel economy.

Where the supply of water suitable for boiler-feed purposes is limited, or where the cost of pumping this water is high, it is important that an open heater should be used, since the closed heater under ordinary conditions requires about $\frac{1}{6}$ more fresh water to make up the supply required by the boilers.

Again, an open heater will materially improve the quality of any boiler-feed supply, no matter how bad it may be, giving opportunities for purification that the closed heater cannot give. In an open heater, as already pointed out, $\frac{1}{7}$ of the entire feed supply is made up of condensed steam. This is perfectly free from scale-forming impurities, and is itself an element of purification. Besides this, an open heater gives the opportunity for precipitating within the heater some of the most common and troublesome scale-forming impurities, such as the bicarbonates of lime and magnesia, because, being vented to the atmosphere, it permits the escape of the carbonic acid gases which are liberated when the water is heated to a temperature of 200° F. or over; and provision can easily be made in heaters of the open type for the detaining and removal of the resulting precipitates and other solid impurities (through sedimentation, filtration and flotation). To gain complete protection from the formation of scale in boilers, it may also be necessary—as when the water contains sulphates—to employ chemical treatment in conjunction with special apparatus for the purpose. The use of an open heater also provides for driving off any air and free gases carried in the feed supply.

None of these opportunities for purification is presented in the closed heater. Being operated under a pressure exceeding that of the boiler, it cannot be vented for the escape of the gases, thus making impossible any considerable precipitation of the soluble impurities; nor can any effective provision be made for settling the solid impurities out of the water, because the water is kept in constant agitation.

These advantages are sufficient to determine the use of open heaters in mining districts.

The scant purification which is obtained in a closed heater is at the expense of heating efficiency, for the precipitated impurities build up a coating of scale on one side of the tubes or coils, while on the other side is collected the oil with which the exhaust steam is contaminated. Both materially hinder the transmission of heat, and cases are now on record of closed heaters which, when new and clean, gave temperatures of 200°–210° F., but which, after a few months' service, would not raise the temperature of the water above 160°–170°.

Again, an open heater is much easier to care for; there is free access to the interior, and it can be thoroughly cleaned without disturbing the pipe connections.

An open heater can be made cheaply and easily on any mine, out of an old boiler shell, plated up at the ends, and passing the steam exhaust pipes through it.

The absolute prevention of any grease or mineral oil entering the boiler is of vital importance. Even a coating of quite inappreciable thickness—such as may be left after wiping off carefully all that can be seen—will effectually prevent actual contact between the water and the boiler plates, and a temperature may thereby be induced in the latter of over 800° F., whereas all kinds of iron and mild steel are extraordinarily weakened and rendered brittle at about 630° F.

Engines.

Duty.—Calculations of the duty or h.p. of an engine can only be approximate, as they are modified by such factors as the friction of the moving parts, the loss of steam at valves and joints, amount of condensation, quality of lubricants, amount of load, and so on; but the following rules may be useful:—

(*a*) Piston area (sq. in.) × mean pressure (lb. per sq. in.) on piston × piston speed (ft. per min.) ÷ 33,000 ft.-lb. = h.p. *Ex.*—Find h.p. of engine whose piston is 12 in. diam., stroke 16 in., rev. 140 per min., steam pressure 30 lb. *Ans.*—Area = 12 in. × 12 in. = 144 sq. in. × ·7854 = 113·0976 sq. in. of circular area × 30 lb. pressure = 3392·928 lb. × speed (16 in × 2 × 140) = 4480 in. = 373·3 ft. per min. = 1,266,580·0224 ÷ 33,000 = 38·38 h.p.

(*b*) Piston diam. (in.) squared × stroke (in.) × rev. per min. × 4. Cut off 5 figures from the right and × mean pressure (lb.) =

h.p. *Ex.*—Diam. (12 in.) × 12 = 144 × stroke (16 in.) = 2304 × rev. per min. (140) = 322,560 × 4 = 1,290,240; remove 5 figures = 12·9 × pressure (30 lb.) = 38·7 h.p.

Cost of Steam Power.

Various estimates have been made of costs of steam power under different conditions.

(*a*) Generated in 1000-h.p. units, with coal at 16*s.* per ton (2000 lb.) and allowing 5% interest, 3½% depreciation, 2% repairs and 1% insurance, and taxes = 70*s.* 10*d.* per annual h.p. of 10 hr. daily. (F. W. Dean, Trans. Am. S. Mec. E.)

(*b*) With oil fuel at a price equivalent to 10*s.* 5*d.* per ton (2000 lb.) for coal = 47*s.* 5*d.* (F. W. Dean.)

(*c*) Lowest average cost per h.p. per ann. (8760 hr.) in U.S. = 97*s.*; in England, 98*s.* 8*d.* (J. B. C. Kershaw, Brit. Assoc.)

(*d*) On Rand mines, excluding interest and depreciation, 20*l.* per h.p. per ann., on about 150,000 h.p. employed.

(*e*) At Kolar, India, on 10,500 h.p., using 190,000 t. fuel costing 230,000*l.*, the total cost per. h.p. per ann. exceeds 30*l.* (Smyth.)

(*f*) *Cost per h.p. Year* (300 *working days of* 10 *hr.*).

Coal at per ton (2000 lb.).		Cost per h.p. hour.											
		3 lb.		4 lb.		5 lb.		6 lb.		7 lb.		8 lb.	
s.	*d.*	*s.*	*d.*	*s.*	*d.*	*s.*	*d.*	*s.*	*d.*	*s.*	*d.*	*s.*	*d.*
8	4	37	6	50	0	62	6	75	0	87	6	100	0
10	5	46	10	62	6	78	1	93	9	109	4	125	0
12	6	56	5	75	0	93	9	112	6	131	3	150	0
14	7	65	7	87	6	103	1	131	3	154	2	175	0
16	8	75	0	100	0	125	0	150	0	175	0	200	0
18	9	84	4	112	6	140	7	168	9	197	11	225	0
20	10	93	9	125	0	156	3	187	6	210	9	250	0

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(*g*) High-speed steam engine, with Lancashire boiler :—

	25 h.p. unit.			100 h.p. unit.		
	£	*s.*	*d.*	£	*s.*	*d.*
Coal: 5 lb. and 4 lb. per b.h.p.-hr. respectively, at 12*s.* per ton ..	90	8	0	289	5	0
Water: 4 gal. per b.h.p.-hr. at 9*d.* per 1000 gal.	10	2	0	40	0	0
Labour: 15*s.* and 27*s.* per week, respectively	37	10	0	67	10	0
Interest, depreciation, etc. 10%	35	0	0	100	0	0
	£173	0	0	£496	15	0
Cost per h.p. per ann.	£6	18	5	£4	19	4

(*h*) Cost of fuel per h.p.-hr., with coal at per ton (2000 lb.) as follows:—

Coal, lb. per h.p.-hr.	4s. 2d.		8s. 4d.		12s. 6d.		16s. 8d.		20s. 10d.		25s. 0d.		29s. 2d.	
	s.	d.	s.	d.	s.	d.	s.	d.	s.	d.	s.	d.	s.	d.
4	0	10	1	8	2	6	3	4	4	2	5	0	5	10
5	1	0½	2	1	3	1½	4	2	5	2½	6	3	7	3½
6	1	3	2	6	3	9	5	0	6	3	7	6	8	9
7	1	5½	2	11	4	4½	5	10	7	3½	8	9	10	2½
8	1	8	3	4	5	0	6	8	8	4	10	0	11	8
10	2	1	4	2	6	3	8	4	10	5	12	6	14	7

(*i*) Cost of fuel per h.p.-hr., with wood per cord (128 cub. ft. = 3000 lb.) as follows:—

Wood, lb. per h.p.-hr.	8s. 4d.		12s. 6d.		16s. 8d.		20s. 10d.		25s. 0d.		29s. 2d.		41s. 8d.	
	s.	d.	s.	d.	s.	d.	s.	d.	s.	d.	s.	d.	s.	d.
10	2	9½	4	2	5	6½	6	11½	8	4	9	8½	13	10½
12	3	4	5	0	6	8	8	4	10	0	11	8	16	8
15	4	2	6	3	8	4	10	5	12	6	14	7	20	10
20	5	6½	8	4	11	1½	13	10½	16	8	19	5½	27	9½

(*k*)

Engine.	Fuel.	Cost per Ton.		Consump. per h.p.-hr.	Cost per h.p.-hr.	Cost per ann. (300 days at 10 hr.) per 100 h.p.		
		s.	d.	lb.	d.	£	s.	d.
Simple, non-conden.	Bit. coal	12	6	5	·334	418	2	2
Comp.-Conden. ..	„	12	6	3	·201	251	0	10
Steam turbine	„	12	6	3	·201	251	0	10

ELECTRIC POWER.

The ease with which electric power can be sub-divided and transmitted over great distances without excessive loss permits the establishment of large-scale generating stations—whether from water or steam—and the accompanying economy of large units.

Of the two systems, continuous current and alternating current, the latter, especially on the triphase style, possesses several advantages. It is cheaper to instal, and admits of being generated and transmitted at a high voltage, at a relatively smaller cable cost. It can be transformed to any suitable working voltage in the neighbourhood of the motor, and the absence of a commutator on the motor considerably simplifies the working and attention required. Its chief disadvantage for economical working arises where regulation in the speed of a motor is required, as a regulator has not yet been invented which will do more than consume the surplus current beyond that required for the speed at which the motor is running; accordingly, the same amount of current is required for a motor running at half load as at full load, the surplus being wasted in the regulator as heat. A shunt-circuit continuous-current motor, on the other hand, can be regulated with very little loss of efficiency.

The voltage used varies very much with the conditions: for triphase current it is not, as a rule, less than 1000 volts; it is usually 2000–3000 volts, and sometimes 5000 volts, and, in the majority of cases, it is transformed down to 200–300 volts at the motor. The pressure of continuous current does not, as a rule, exceed 500 volts.

When the power has got to be uniform, as in the case of stamp-batteries and compressors, the advantages of electric motors are pronounced; but, where the load is not uniform, as in the case of hauling from deep shafts, the advantages over steam power are not so great. For triphase motors, no man of greater skill than an ordinary millwright is needed for their operation and handling; for so long as there is oil in the bearings, and electric current is supplied, the work will go on. The oil in the bearings need not be replaced under ordinary circumstances oftener than once in 3 months; and practically the only attention needed is to keep the machinery clean.

In some cases, where with coloured labour there is much shirking on night shift, and stoking is slack, electric power may be of great advantage apart from economy. Thus at Kolar, India, it is found that 20% more work is done, owing to the uniformity of power supplied.

In all electric power installations, a small (5 h.p.) air compressor and necessary hose connections will be found invaluable for cleaning machinery. Pressure should be available at 100 lb. per sq. in.

Costs.

(*a*) Coventry Corporation works report that on 51,114 units sold in 1896, the costs were 6·04*d.* per unit, and the average selling price was 5·85*d.*; while on 1,375,735 units sold in 1905, the cost was ·85*d.* and selling price 2·28*d.*

(*b*) Truckee River Co., Nevada, U.S., sells power to the Comstock mining companies at 16*s.* 8*d.* to 29*s.* 2*d.* per h.p. per month, according to power consumed.

(*c*) The Mysore Govt. (India), utilising Cauvery river falls, which generate 5000 h.p. and deliver 4000 h.p. to Kolar mining companies, expended over 340,000*l.*, and sell power at 18*l.*–29*l.* (average 20*l.* 4*s.*) per h.p. per ann. for the first 5 years, and uniformly at 10*l.* per h.p. per ann. for the second 5 years; and they reckon to recover the whole prime cost in 6 years, and to earn 10% revenue afterwards.

(*d*) Grand-Hornu coal mine (Belgium), triphase plant, using coal, generating 6000 kw., cost per kw. at switch-board, including amortisation on 20 years capital of 34,000*l.* at 4%, is only ·217*d.* on a 16,000 kw. per 24 hr. consumption; with an increase to 24,000 kw. per 24 hr., it is not expected to exceed ·171*d.* These figures = 9*s.* 8½*d.* and 7*s.* 7½*d.* per h.p. per month respectively.

(*e*) It is said that electric power generated by water-fall at Rheinfelden, at Zurich, and at Buffalo costs more than steam power would cost in S. Lancashire with large units.

(*f*) At the Raub gold mine (Malaya), with a water-power plant capable of affording 1500 h.p., but only actually transmitting 300 h.p., of which not more than 230 h.p. is in use at any one time, and transmission over 9 miles of cable, all costs of running and upkeep, based on 230 h.p. used, amount to about 13*s.* 3*d.* per h.p. per month.

(*g*) Lake Superior Power Co., using 18 ft. head in St. Mary river, giving 3,600,000 cub. ft. per min., or 135,000 h.p., of which 110,000 h.p. is available, but only 60,000 h.p. utilised, can sell power at 5*s.* 3*d.* per h.p. per month, and pay interest at 10%, and depreciation at 2½%.

(*h*) An English works, using blast-furnace waste gases, generates at 11*s.* 8*d.* per h.p. per month.

(*i*)

Motor Efficiency.	Kw. per h.p.-hr.	Current at per kw.-hr.						
		d. ·25	*d.* ·375	*d.* ·5	*d.* 1	*d.* 1·5	*d.* 2	*d.* 2·5
		Cost per h.p.-hr.						
%		*d.*	*d.*	*d.*	*d.*	*d.*	*d.*	*d.*
80	·932	·233	·349	·466	·93	1·39	1·86	2·33
85	·878	·219	·329	·439	·87	1·31	1·75	2·19
90	·829	·207	·311	·414	·82	1·24	1·65	2·07

(*k*) Niagara power plant transmits 24,000 h.p. to Buffalo, and sells at about 25*l*. per h.p. per ann.

(*l*) Taking current at 1*d*. per Board of Trade unit, allowing $\frac{3}{4}$ unit to develop 1 h.p., and including interest, depreciation, etc., at $7\frac{1}{2}\%$:—

25 h.p. plant,	209*l*. 10*s*. per ann.	= 8*l*. 7*s*. 7*d*.	per h.p. per ann.
100 „ „	831*l*. 5*s*. „ „	„ 8*l*. 6*s*. 3*d*.	„ „

(*m*) In estimating cost of an electric power supply from various initial sources, the first cost of plant is an important consideration. A good steam plant for using cheap coal will cost for the power house, including buildings, 22*l*. per kw.; a gas plant for using coke-oven gas, about the same; and a gas plant for anthracite or coke, about 24*l*. per kw. Exhaust steam turbines would probably cost 18*l*. per kw. Works costs do not give a true value unless they include depreciation and interest on capital, and it is difficult to keep this item below ·25*d*. per unit. Coal should be economised, even when it is cheap, as the costs for labour, boiler repairs, and depreciation are at least 2*s*. 6*d*. per ton burnt, and in many cases considerably more. Collieries are at a great advantage in having cheap coal, and, with a fairly steady load of not less than 200 kw., the total generating costs may be brought down to ·6*d*. per unit, made up as follows: Coal and boiler charges, ·2*d*.; depreciation and interest, ·25*d*.; labour, stores, and repairs, ·15*d*. With larger plants, these figures may be reduced. The first cost of electric motors has been reduced during the last few years. Motors of fair size can be bought for 2*l*. per b.h.p., including switch gear, as compared with 5–6*l*. for high-class steam engines.

Units.

1 watt = 44·238 ft.-lb. per min.
1 kilowatt = 44,238 ft.-lb. per min.
1 kilowatt-hr. = 1·34 h.p.-hr.
1 h.p.-hr. = ·746 kw.-hr.

GAS POWER.

The gas-engine power plant possesses many qualifications which make it particularly suitable for mining purposes. The first cost is not appreciably greater than that of the steam power plant (when one includes the various auxiliaries needed to attain reasonable efficiency), while in fuel economy it is vastly superior. When fuel must be transported long distances, as is frequently the case, the problem of irregular or even interrupted supply due to strikes, wrecks, floods, and numerous other causes of impeded transportation, is a serious consideration; and a system of producing power which requires only half the usual quantity of fuel must come greatly into favour, if only from that point of view. Besides, an ample supply of water suitable for boiler purposes is seldom to be found in mining localities, while the water consumption in a gas plant is negligible in quantity and may be of very inferior quality. The difference in relative maintenance is probably slightly in favour of the gas plant as against the steam plant. It is perhaps premature to speak of rate of depreciation in gas plants; but, as there is no danger or risk from high pressures, the metal work should be capable of use until quite corroded, and there is nothing to specially cause corrosion. The brick lining of the furnace will, of course, suffer wear and tear, but it is easily renewable at no great cost. The vaporisers may also become incrusted, but one cannot anticipate anywhere heavy depreciation charges. An important feature in the economy of gas plants is that there are no losses due to standing idle, or to radiation, or to excessive cylinder condensation with light loads. At present, the maximum unit is about 500 h.p., but, of course, the units can be (and much more conveniently would be) multiplied, and, even if 500 h.p. were the ultimate limit, the great majority of enterprises would be fully served thereby. A power station equipped with gas engine plant has a great advantage in that it can be put to work at full power after much shorter notice than is required with steam plant. On the score of attention, there is not much to choose between gas and steam plants, but the former should be systematically examined at short intervals, because troubles usually come from hidden sources. There is far greater safety than with high-pressure boilers and steam pipes, and no more care or skill is needed than with the stoking and water-feeding of a steam plant.

In one important respect there is inferiority to steam, namely, that a gas plant has not the same elasticity. It will not tolerate constantly varying loads. If overloaded, it simply refuses to work; if underloaded, that is if run for any length of time on quarter-load, for example, the area of incandescent fire becomes reduced, and the quality of the gas falls off (in the case of producer plants). This is remedied to some extent in modern installations, quite sufficient for ordinary fluctuations of load; but no gas plant is directly applicable to such a varying load as is presented by the mine hoist: here some form of accumulator must be interposed, and none is more convenient than an electric motor.

Several kinds of gas are suitable for gas engine use, but the kind mostly used in large plants is producer gas, containing 130–160 b.t.u. per cub. ft. Producer gas can be made from bituminous or anthracite coal, coke, wood, or charcoal.

The gas generator can easily deliver in the gas 80 % of the heat units in the fuel, and the gas engine can convert 25 % of this into delivered power, a total thermal efficiency for the gas system of 20 %. The best steam generators, on the other hand, are doing quite well when, under every-day working conditions, they deliver in the steam 70 % of the heat units of the fuel; and as steam engines will deliver 10–15 % of this as useful power, the combined efficiency of the steam system is 10·5 %, or just about half that or the gas system.

Producer gas is mainly carbonic oxide (CO), the poisonous quality of which makes it necessary that suitable precautions should be taken as to leakage, and that provision should be made for satisfactory ventilation when installing gas producers.

All gases made in modern power plants are of the "semi-water" class, and whether the steam and air are forced through the fuel by the pressure of the atmosphere, or whether the pressure is set up by a special blower, makes but little difference to the gas evolved. But the suction gas producer has advantages over the pressure type for certain purposes, especially gas-engine operation. For developing motive power in an engine, about 70 cub. ft. of gas are required to give 1 b.h.p., and the gas needs, for complete combustion, rather more than an equal volume of air.

The weight of gas engines as compared with that of steam engines per unit of nominal rated capacity is approximately two to one. The usual type of gas engine operates on the four-cycle principle, or in other words, receives an impulse on the piston every fourth stroke, or every second rev. for a single-acting engine. Most steam engines are double-acting, and receive besides an impulse on each side of the piston at each rev., consequently, with approximately equal mean effective cylinder pressure, the cylinder volume of the gas engine must be four times as great for same piston speed. To secure uniform rotation, this volume must be distributed over a number of cylinders, or a very heavy flywheel must be provided.

Besides the greater weight due to increased cylinder volume and heavy fly-wheel, the frame and working parts of the engine must be designed for the great and suddenly-applied strains of the explosions. The gas-engine plant, however, needs no condenser system, no boiler feed-pump, no feed-water heater, no stack, no steam piping, or other auxiliaries of a complete power generating plant, and this offsets the extra gas-engine weight. The gas-engine plant, therefore, with its producer-gas generator complete, aggregates about the same weight as a steam-engine plant with its boilers, stack, condensers, and other auxiliary apparatus. For the smaller sizes of gas engine (under 300 h.p.) operating with suction producers, the weight of the gas-engine plant is considerably less than for equivalent power in high-grade steam plants. The gas engine is a much more costly machine per unit of capacity, but the numerous auxiliaries of the steam plant tend to produce an equality in first cost of complete plants of the two types.

In common with all other machinery, gas plants have weaknesses. The gas may vary in quality, and provision must be made for this. The engine may be slowed or even stopped by the electric firing plug failing to spark. The spindle carrying the plug cannot be lubricated (on account of the heat), and it may stick.

The plug also sometimes gets dirty. The remedy is to change the plug, which can be done in a few minutes. Sometimes the magneto arrangement goes wrong, but it is simply righted. The valve-springs corrode rapidly, but their cost is trifling, and they should be renewed at each cleaning of the valves (say fortnightly). Pistons need attention every three months.

As to running expenses, it is claimed that on 80 % of full load the cost for 54 hr. per week of a 50-week year will not exceed ·3*d.* per h.p.-hour (with coal at 26*s.* per ton, oil, waste, rent, water taxes, and 10 % for interest charges, etc.); for fuel alone, ·1*d.* per h.p.-hour will suffice. For larger plants the cost falls; and for longer hours it, of course, falls also, so that for a plant of 110–130 h.p. the cost per h.p.-hour for a year of 7200 hours on continuous 80 % load is set down at ·25*d.*, or at ·06*d.* for fuel only.

The results of a series of tests made on a steam plant in competition with two gas plants (one using exhaust gases and the other producer gas), and at 50 h.p., 25 h.p., and friction load (i.e. empty brake = about 8 h.p.) are shown below:—

Efficiency of Steam and Gas Plants. (G. H. Barrus.)

—	Gas Engines,		Steam Engine, Non-condensing.
	Producer Gas.	Waste Gases.	
Brake h.p.	78·7	102·5	100
Coal per h.p.-hour, lb.	1·8	1·12	3·6
B.T.U. supplied to plant per hr...	1,733,800	1,402,644	4,411,080
" " " per h.p.-hr.	20,640	13,396	44,111
" " to engine per hr.	1,400,374	1,028,712	3,273,400
" " " per h.p.-hr.	16,667	9,797	32,734
" converted into work per h.p.-hr.	2,545	2,545	2,545
Efficiency of whole plant, % ..	12·3	19	5·7
" producer, %	80	73·1	74
" engine, %..	15·3	25·9	7·9

On large powers the comparison is less favourable to gas, because large steam engines are run compound-condensing, and may reduce their coal consumption to 1·5 lb. per h.p.-hr. in the best installations. But probably no gas plant of equal size on any scale would consume as much as 60 % of the fuel used by the best steam plants.

Fuels.—Such fuels as charcoal, coke, anthracite, coal, wood, peat, tan-bark, saw-dust, etc., may be used. In general practice, however, the most economical, cleanest and most satisfactory fuel is charcoal or coke or anthracite, depending, of course, on the locality in which the plant is operated. Charcoal is an excellent fuel, and is always preferable to wood, for the reason that a greater number of heat units are contained in 1 lb. charcoal than in 1 lb. wood, besides the avoidance of tarry matters. Latterly something has been done in the way of using ordinary hard, non-caking, bituminous coals, which in some districts are cheaper than anthracites. But it must be remembered that a mere comparison of the prices of the raw fuels alone does not constitute a complete basis from which to compute the relative costs of a h.p.-hour obtained in given cases. Some fuels have drawbacks (especially in plants of average size) in the removal of the tar which is found in gas made from bituminous matter, requiring more extensive cleaning apparatus (part of which may need engine power to drive it), while the water consumption is increased for the same reason.

Coal.—Anthracite should be broken into pieces of $\frac{3}{4}$–$1\frac{1}{4}$ in. cube, and should be perfectly dry. If the coal is smaller than this, or if it contains small, there is a tendency to clog, and not only make bad gas, but require a large quantity of water in the scrubber. If there is any dust in the coal, it is better to sieve it out rather than attempt to use it. The claim is made that 1 h.p.-hour can be secured with 1 lb. anthracite for small units; for the larger units this amount often drops as low as ·8 lb. A coal carrying 38 % C, 41 % volatile matter, and 21 % ash has done well in producers when mixed with coarse coke.

A 3000 h.p. plant, burning Mexican coal at the rate of 1·25 lb. per b.h.p.-hr., is shown in Fig. 13. The power is transmitted electrically to 7 different mines ranging from 500 yd. to 15 miles distant. There are 4 600-h.p. engines driving 300-kw. triphase generators.

In an example where the coal consumption was 250 lb. per 10 hr., coal cost 21*s.* per ton (2240 lb.), and the average h.p. was 22, the cost per h.p.-hour worked out as follows:—

	d.
Coal (2·81*d.*) per h.p.-hour	·127
Depreciation, 10 %	·120
Attendance, 15*s.* per week	·128
Maintenance, 5*s.* per week	·050
Oil	·002
Water, 8*d.* per 1000 gal.	·002
Coke and firebrick, say	·021
Total cost per h.p.-hour	·450

Taking the total coal consumed in 6 months, and the number of hours worked by the engine, the consumption of coal comes out at ·14*d.*, making the total expense amount to ·463*d.* (A. J. Stevens.)

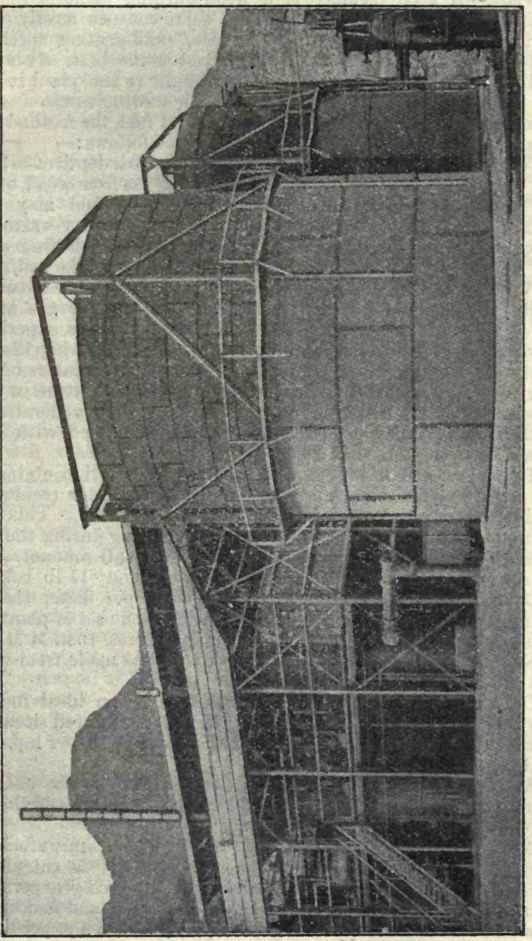


FIG. 13.—3000 H.P. GAS GENERATING AND STORAGE PLANT, MEXICO.

Wood.—A good instance of a wood-burning plant is that at Nacozari (Mex.). The wood fuel used there consists mostly of an inferior scrubby oak, a little "mesquite," and another species of oak locally called "black" oak. This last is the best. It is supplied from neighbouring hills, and is brought to the plant in 3 ft. 6 in. lengths. These are sawn in two with a swing saw.

In starting and operating with such wood fuel, the method that 4 years' experience has shown to be best, is as follows:—

Coke is put on the grates of the generators to a depth of 3–3½ ft. Less than 3 ft. is not advisable when making gas from wood, otherwise volatile products of distillation from the fuel may pass through unfixed, and, by condensing to soot or tar, may waste fuel and cause trouble by dirt. Wood running high in volatile matter, such as pine, would probably demand a still deeper coke bed.

With wood fuel, enough moisture is present in the fuel itself to supply the steam for decomposition. When making gas at the rate of 1100 cub. ft. per min. from 6 ft. 9 in. generators, experience showed best results to be obtained from wood containing 12–14 % moisture. At this rate of gasification, and from generators of this internal diam., good results cannot be obtained if moisture in wood exceeds 20 %. At a higher rate of speed, or with generators of smaller diam., fairly good results would be obtained with wood containing even more than 20 % water.

The driest and best seasoned wood at Nacozari contains not less than 8 % moisture, and it is found that better results are obtained from wood containing slightly more water. This may be due to the fuel value of the wood deteriorating during the long exposure necessary to reduce moisture to a small amount. The fuel consumption per h.p.-hr. was 2·6 lb. wood and ·11 lb. coke.

Burning wood at the rate of 30,000 lb. per diem, the ash accumulates, so that by the fifth day's run on one set of plant it is necessary to shut down to clean. (Douglas, Trans. Inst. M.M.)

No engineer should attempt to work with gas made from wood, if he can get charcoal.

Charcoal.—Most makers regard charcoal as an ideal fuel for their plant, and a supply can generally be got in timbered situations where coal is not available. A consumption of 1¼ lb. per h.p.-hour is seldom exceeded.

At one 50-h.p. plant, the average load is 35 h.p.; the charcoal consumption is 35 lb. per hr., or 175 lb. per 5-hr. run; and the cost of operating is 2*s.* 1*d.* per 5-hr. day.

A small (200*l.*) plant installed in Mexico for unwatering a mine (2 weeks' work) gave following figures: A calculation based on the actual quantity of water raised during a period of 24 hr. showed an average (including all stoppages, and making no allowance for friction of working parts, etc.) of 3·2 h.p. per hr. for the whole period, with a consumption of charcoal in the producer of 308 lb., or 4 lb. charcoal per h.p.-hr. The charcoal, delivered at the mine (and allowing for loss due to pulverisation in transit), cost ·56*d*. per lb., hence the cost per h.p.-hr. came to 2·24*d*. The cost of power per 24-hr. day was 15*s*. 7*d*. for charcoal, 6*s*. 3*d*. for lubricants, etc., and 6*s*. 3*d*. for labour, total, 28*s*. 1*d*. This is much higher than wood costs at Nacozari; but considering the smallness of the unit, and its simplicity and low first cost, it compares very favourably with any other power. (W. H. Rundall, Trans. Inst. M.M.)

Waste Gases.—At collieries where coke is made, there is a tendency to take the gas from the ovens, and, after purification, use it to generate electricity direct by means of gas engines. The saving of power by this method over that of burning the gas under boilers, and generating electricity by the steam so raised, is large.

If the cost for repairs and renewals is not excessive, large gas engines in conjunction with coke ovens and blast furnaces may enter into serious competition even with water turbines as a source of power. It has been calculated that 2,000,000 h.p. is wasted annually in the gases issuing from the blast furnaces of the United Kingdom; the estimate for the United States would be still higher. The waste of gases from coke ovens forms also an item of very great importance.

Gasoline.—On the basis of 1 pint gasoline per h.p.-hr., the fuel cost will be as follows :—

Gasoline per gal... ..	4*d*.	4½*d*.	5*d*.	5½*d*.	6*d*.	6½*d*.	7*d*.	7½*d*.
Power cost per h.p.-yr.	6*l*.	6*l*. 15*s*.	7*l*. 10*s*.	8*l*. 5*s*.	9*l*.	9*l*. 15*s*.	10*l*. 10*s*.	11*l*. 5*s*.

At a small Mexican mine, a 25-h.p. gasoline engine installed to pump and hoist from 200 ft., cost, for gasoline alone (at 3*s*. 4*d*. per U.S. gal. delivered), 4*l*. for every 2 shifts (16 hr.) work. (Rundall.)

In another instance, the cost for pumping 150 ft., with crude Californian oil at 2*d*. per U.S. gal., was under 6*d*. per 1000 cub. ft. water raised.

Two 12-h.p. gasoline locos. at a cement works, Germany, haul 16 cars of 1·75 ton each on level track at a cost of ·189*d*. per ton-mile.

Alcohol.—Ethyl-alcohol forms an ideal fuel—colourless, limpid, of moderate boiling-point (about 50° below that of water), non-freezing, burning without smoke, mixing with water in all proportions (and, therefore, its flame extinguished by water , cleanly, drying off completely when spilled, not attacking rubber gaskets or packings, and non-corrosive for metal tanks and holders. The fact that its flame is bluish, or so-called non-luminous, means that it is almost devoid of free carbon particles, with their intense heat-radiating power—a fact of considerable importance.

The production of alcohol on a large scale is very simple, and the raw materials already exist in considerable variety. All saccharine or starchy growths are available. Saccharine wastes are now largely used in Cuba for alcohol production. At present it is said that the low grades of molasses can be delivered at American coast cities at about 1½*d.* per U.S. gal. About 3 gal. of this crude product are required to produce 1 gal. refined spirit (90 % alcohol), and the cost of production may be estimated at 1½*d.*–2*d.*, making the cost of the alcohol per U.S. gal. about 6*d.*

This alcohol will, in a properly organised engine, equal, volume for volume, gasoline now sold at a much higher price, in producing power.

A number of such engines are in operation as locomotives in Germany:—

(*a*) 9 locos. of 8 h.p., working in 10½-hr. shifts, on a grade of 1:333, costing ·236*d.* per ton-mile.

(*b*) 12-h.p., at a paper-mill, operating 12 hr. daily on a 3-mile track, hauling 5 t. on a 2·5 % grade.

(*c*) 16-h.p., timber hauling, at 2½–7 m. per hr., with draw-bar pull of 1500 lb. at former speed and 500 lb. at latter.

(*d*) 32-h.p., speed 2½–7 m. per hr.

In the distant future, when our stores of coal and mineral oil have been depleted, the internal-combustion engine, using alcohol as fuel, is likely to attain great prominence.

Comparative Costs of Fuels for Gas Engines. (Dugald Clerk.)

Coal gas: calorific value, 600 b.t.u. per h.p.-hr.

Cost per 1000 cub. ft.	Cost of 10,000 b t.u.	30% Efficiency.		25% Efficiency.	
		Fuel Cost per h.p.-hr.	Total Cost per h.p.-hr.	Fuel Cost per h.p.-hr.	Total Cost per h.p.-hr.
s. *d.*	*d.*	*d.*	*d.*	*d.*	*d.*
1 0	·2	·17	·27	·20	·30
1 6	·3	·24	·34	·29	·39

Anthracite: 14,000 b.t.u. per lb. Producer efficiency, 85%.

Cost per ton.	10,000 b.t.u.	Fuel.	Total.	Fuel.	Total.
s.	*d*	*d.*	*d.*	*d.*	*d.*
24	·097	·092	·22	·11	·24
Scotch anthracite: 12,000 b.t.u. per lb.					
15	·071	·068	·16	·08	·18

Coal 11,500 b.t.u. per lb.

Cost per Ton.		Cost of 10,000 b.t.u.	Fuel Cost per h.p.-hr. at 3% and 85%.	Steam Turbine Plant.		
				Fuel Cost per h.p.-hr.	Sale Price per Unit.	Price per h.p.-hr. at 85% Motor Efficiency.
s.	d.	d.	d.	d.	d.	d.
5	9	·029	·028	·082	·55	·49
7	6	·037	·035	·108	·72	·64
8	6	·044	·041	·120	·80	·72

Petroleum (lamp oil): 20,000 b.t.u. per lb.: ·8 lb. per h.p.-hr.

Cost per gal.	10,000 b.t.u.	Fuel.	Total.
d.	d.	d.	d.
6	·33	·53	·63
4	·22	·35	·45

During 1905 a series of tests with suction gas-producers was carried out under the Highland and Agricultural Soc. of Scotland. There were 2 plants of each, one 5–8 h.p., the other 15–20 h.p. Plants were priced at 5–6*l.* per b.h.p. for 10–12 h.p., and about 4*l.* per b.h.p. for 15–20 h.p. The trials were made with pea anthracite costing 9*s.* 3*d.* per ton. Only 2 men were allowed per plant. About 15 min. was required to start from rest, empty, and cold. On the whole of the trials (6 large and 4 small producers) the average coal consumption per b.h.p.-hr. was 1 lb., costing ·05*d.*, which means that a 20-b.h.p. engine costs 1*d.* per hr. for fuel. In the small plants the consumption ranged from ·87 to 1·3 lb.

Fuel Costs under Various Conditions.

——	——	Price.	Consump. per h.p.-hr.	Cost per h.p.-hr.	Cost per ann. (300 days at 10 hr.) per 100 h.p.	
				d.	£	s.
Gasoline ..	Oil engine	7*d.* per gal.	·125 gal.	·88	1093	15
Crude oil ..	„	2*d.* „	·1 „	·2	250	0
Illum. gas	Gas engine	37·5*d.* per 1000 cub. ft.	19 cub. ft.	·712	890	12
Nat. gas ..	„	10*d.* „	13 „	·13	162	10
Anthracite	„	12*s.* 6*d.* per ton (2000 lb.)	1 lb.	·067	83	15
Coke	„	„	1·25 lb.	·083	104	12

Gas Transmission: Main 4 ft. diam., 70 cub. ft. Gas per h.p.-hr.

Initial Pressure, Absolute.		Length of Transmission, Miles.			Gas Delivered, h.p.			Compression of Gas, i.h.p. required.			Efficiency of Transmission.
Lb. per sq. in.	Atmospheres.	Initial Velocity, ft. per sec. 20	30	40	Ft. per sec. 20	Ft. per sec. 30	Ft. per sec. 40	Ft. per sec. 20	Ft. per sec. 30	Ft. per sec. 40	
15	1·02	7·66	3·4	1·92	13,130	19,200	26,260	30	44	60	99·7
15·4	1·05	18·6	8·2	4·65	13,550	19,800	27,100	65	95	130	99·5
16·2	1·1	34·5	15·3	8·6	14,150	20,700	28,300	116	171	233	99·2
17·6	1·2	61	27·1	15·2	15,500	22,600	31,000	273	399	545	98·2
22·0	1·5	133	59	33	19,300	28,350	38,750	780	1,120	1,535	96·2
29·4	2	150	66	37	28,850	37,820	51,600	1,780	2,610	3,565	93·4
44·1	3	178	79	44	38,750	56,700	77,500	4,500	6,596	9,000	89·6
58·8	4	182	83	46	51,650	75,500	103,200	7,780	11,400	15,550	87·0
73·5	5	191	85	48	64,600	94,600	129,100	11,550	16,910	23,100	84·9
88·2	6	194	86	48·5	77,500	113,500	155,000	15,890	23,290	31,875	83·0
102·9	7	196	87	48·9	90,300	132,000	180,700	20,630	34,560	41,370	81·5

Mechanical efficiency of Compressor, 80%.

Gas Engine on Coal Gas at 2s. per 1000 cub. ft.

	25-h.p. plant.			100-h.p. plant.		
	£	s.	d.	£	s.	d.
Gas at 17 and 15 cub. ft. per h.p. respectively	114	15	0	405	0	0
Labour, 1 hr. per day at 6*d*.	7	10	0	7	10	0
Interest, depreciation, etc., 10% ..	16	0	0	50	0	0
	£138	5	0	£462	10	0
Cost per h.p. per ann.	£5	10	5	£4	12	6

Gas Engine on Semi-Water Gas.

	25-h.p. plant.			100-h.p. plant.		
	£	s.	d.	£	s.	d.
Coal: 1 lb. per h.p.-hr. at 20*s*. per ton	30	0	0	120	0	0
Water: ¾ gal. per h.p.-hr. at 9*d*. per 1000 gal.	2	0	0	8	0	0
Labour	12	10	0	22	10	0
Interest, depreciation, etc., at 10% ..	25	0	0	65	0	0
	£69	10	0	£215	10	0
Cost per h.p. per ann.	£2	11	7	£2	2	1

Gas Engine on Illuminating Gas.

Cub. ft. gas per h.p.-hr.	Gas costing per 1000 cub. ft.			
	s. d. 2 6	s. d. 3 4	s. d. 3 9	s. d. 4 2
	Cost per h.p.-hr.			
	d.	d.	d.	d.
15	·45	·60	·65	·75
17	·51	·68	·77	·85
20	·60	·80	·90	1·00

Gas Engine on Natural Gas.

Cub. ft. gas per h.p.-hr.	Gas costing per 1000 cub. ft.		
	d. 10	d. 12½	d. 15
	Cost per h.p.-hr.		
	d.	d.	d.
9	·09	·11	·13
10	·10	·12	·15
11	·11	·14	·17
12	·12	·15	·18

Gas Engine on Producer Gas.

Lb. coal per h.p.-hr.	Coal costing per ton (2000 lb.)						
	s. d. 4 2	s. d. 8 4	s. d. 12 6	s. d. 16 8	s. d. 20 10	s. d. 25 0	s. d. 29 2
	Cost per h.p.-hr.						
	d.	d.	d.	d.	d.	d.	d.
·80	·020	·04	·06	·08	·10	·12	·14
1·00	·025	·05	·07	·10	·12	·15	·17
1·25	·031	·06	·09	·12	·15	·18	·21
1·50	·037	·07	·11	·15	·17	·22	·26

Gas Engine using 20 cub. ft. Illuminating Gas per h.p.-hr.

Gas costing per 1000 cub. ft.	3s. 1d.	3s. 4d.	3s. 7d.	3s. 9d.	4s.	4s. 2d.
Cost per h.p.-yr. (300 days of 10 hr.)	9l.	9l. 12s. 6d.	10l. 12s. 6d.	11l. 5s.	11l. 17s. 6d.	12l. 10s.

Gas Engine using 15 *cub. ft. Natural Gas per h.p.-hr.*

Gas costing per 1000 cub. ft...	8*d.*	9*d.*	10*d.*	11*d.*	12*d.*	12½*d.*
Cost per h.p.-year (300 days of 10 hr.)	30*s.*	33*s.* 9*d.*	37*s.* 6*d.*	41*s.* 3*d.*	45*s.*	46*s.* 10*d.*

Gas Engine using Producer Gas at 1·25 *lb. Coal per h.p.-hr.*

Coal costing per t. (2000 lb.)	8*s.* 4*d.*	10*s.* 5*d.*	12*s.* 6*d.*	14*s.* 7*d.*	16*s.* 8*d.*	18*s.* 9*d.*	20*s.* 10*d.*
Cost per h.p.-yr. (300 days of 10 hr.)	13*s.* 11*d.*	17*s.* 4*d.*	20*s.* 10*d.*	24*s.* 5*d.*	27*s.* 10*d.*	31*s.* 3*d.*	34*s.* 9*d.*

TRANSMITTING POWER.

BELT DRIVING.

Belting.—The resistance of belts to slipping is independent of their breadth, consequently there is no advantage in increasing their dimension beyond that which is necessary to enable the belt to resist the strain. The ratio of friction to pressure for belts over wooden drums is, for leather belts, when worn ·47, when new ·5, and when over turned cast-iron pulleys, ·24 and ·47. A leather belt will safely and continuously resist a strain of 350 lb. per sq. in. of section, and a section of ·2 sq. in. will transmit the equivalent of 1 h.p. at a velocity of 1000 ft. per minute over a wooden drum, and ·4 sq. in. over a turned cast-iron pulley. In high-speed belting the tension or the breadth of the belt should be increased in order to prevent belt from slipping. Long belts are more effective than short ones. A single belt 1 in. wide travelling at a velocity of 1000 ft. per minute, transmits 1 h.p. A double belt 1 in. wide, travelling 700 ft. per minute, transmits 1 h.p. When a double belt is long and runs over large pulleys it may be calculated to do 1 h.p. of work at a speed of 500 ft. per minute. The upper side of the pulley should always carry the slack belt. To throw a belt on to its pulleys, when it has been laid off, it should always be laid on to the pulley that is not in motion first, and then be thrown over the edge of the moving pulley on to its face. A belt will transmit about 30 per cent. more power, with a given tension, when the grain (smooth side of the leather) is in contact with the pulley than when the flesh side is turned inward. The leather is also less liable to crack, as the structure on the flesh side is less dense, and the fibres less extensible. The adhesion of belts is greater on polished than on rough pulleys, and is about 50 per cent. greater on a leather-covered pulley than on a polished iron pulley. Larger pulleys and drums may be covered with narrow strips of leather or with longer strips wound spirally. Pulley covers are manufactured in strips of the desired width, and reduced to uniform thickness by machinery. Belts should be kept soft and pliable by applying tallow occasionally, and neat's-foot or liver oil with a little rosin when they become hard and dried. Rubber belts ought always to be kept free from grease and animal oils. If they slip, moisten the inside of the belt with boiled linseed oil. Fine chalk sprinkled on over the oil will help the belt.

To find Length of Belts.—Rule: Add the diameter of the two pulleys together, multiply by $3\frac{1}{7}$, divide the product by two, add to the quotient twice the distance between the centre of the shafts, and the product will be the required length.

To Calculate Power of Belting.—1 in. single belt moving at a velocity of 1000 ft. per minute = 1 h.p.; 1 in. double belt moving 700 ft. per minute = 1 h.p. Then h.p. of any belt equals its velocity in ft. per minute, multiplied by its width and divided by 1000 for single and by 700 for double belts.

Horse-power and Breadth of Leather Belts. (Nystrom.)

B = breadth of belt in in.
HP = horse-power transmitted by the belt.
V = velocity in ft. per second of the belt.
d = diameter in in. } of the smallest pulley.
n = revolutions per minute } of the smallest pulley.
F = motive force in lb. transmitted by the belt.
Z = half angle of contact of belt on the small pulley.
S = safe working strength in lb. per in. of width of belt, which for oak-tanned leather $\frac{1}{4}$ in. thick, cemented and riveted joints, can be taken at 100 lb. and less in proportion for weaker belts.

$$\text{HP} = \frac{d\,n\,\text{F}}{126050} = \frac{60\,\text{B}\,\text{V}}{1000} \text{ for single thickness} \quad (1)$$

$$\text{HP} = \frac{\text{B}\,d\,n\,\text{Z}\,\text{S}}{15000000} \quad (2)$$

$$\text{HP} = \frac{\text{B}\,\text{V}\,\text{Z}\,\text{S}}{130000} \quad (3)$$

$$\text{B} = \frac{15000000\ \text{HP}}{d\,n\,\text{Z}\,\text{S}} \quad (4)$$

$$\text{B} = \frac{\text{T} + t}{2\,\text{S}} \quad (5)$$

$$\text{F} = \frac{126050\ \text{HP}}{d\,n} = \frac{550\ \text{HP}}{\text{V}} \quad (6)$$

Ex.—A leather belt is to transmit h.p. = 75 horse-power over a pulley $d = 36$ in. diameter, making $n = 80$ revolutions per minute; angle of contact $\text{Z} = 85°$, and the safe working strength $\text{S} = 100$ lb. per in. of width. Required the width of the belt.

$$\text{Width } b = \frac{15000000 \times 75}{36 \times 80 \times 85 \times 100} = 46 \text{ in. nearly.}$$

Breadths of Belts in Inches for different Motive Forces and Angles of Contact. (Nystrom.)

Motive Force.	Whole Angle of Contact 2 Z.									
	60°	70°	80°	90°	100°	110°	120°	130°	140°	150°
F lb.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.
10	0·424	0·372	0·331	0·300	0·275	0·254	0·238	0·223	0·211	0·200
20	0·847	0·743	0·662	0·599	0·549	0·509	0·475	0·446	0·421	0·400
30	1·271	1·115	0·993	0·898	0·824	0·763	0·713	0·670	0·632	0·600
40	1·695	1·487	1·324	1·198	1·099	1·018	0·950	0·893	0·842	0·800
50	2·119	1·859	1·655	1·497	1·374	1·272	1·188	1·116	1·053	1·000
60	2·543	2·230	1·987	1·796	1·648	1·526	1·425	1·339	1·263	1·200
70	2·966	2·602	2·318	2·095	1·923	1·780	1·663	1·562	1·474	1·400
80	3·390	2·974	2·648	2·396	2·198	2·036	1·900	1·786	1·684	1·600
90	3·813	3·345	2·980	2·695	2·472	2·290	2·138	2·009	1·895	1·800
100	4·237	3·717	3·311	2·994	2·747	2·544	2·375	2·233	2·105	2·000
120	5·084	4·460	3·974	3·592	3·296	3·053	2·850	2·678	2·526	2·400
140	5·932	5·204	4·636	4·190	3·846	3·560	3·326	3·124	2·948	2·800
160	6·780	5·948	5·296	4·792	4·396	4·072	3·800	3·572	3·368	3·200
180	7·626	6·690	5·960	5·390	4·944	4·580	4·276	4·018	3·790	3·600
200	8·474	7·434	6·622	5·988	5·494	5·088	4·750	4·464	4·210	4·000
220	9·321	8·177	7·284	6·586	6·043	5·596	5·225	4·910	4·631	4·400
240	10·17	8·920	7·948	7·184	6·592	6·104	5·700	5·356	5·052	4·800
260	11·02	9·663	8·610	7·783	7·141	6·613	6·175	5·800	5·473	5·200
280	11·86	10·41	9·273	8·380	7·692	7·120	6·652	6·248	5·896	5·600
300	12·71	11·15	9·933	8·982	8·241	7·632	7·125	6·696	6·315	6·000
320	13·56	11·90	10·59	9·584	8·692	8·144	7·600	7·144	6·736	6·400
340	14·41	12·64	11·21	10·18	9·241	8·652	8·075	7·590	7·157	6·800
360	15·25	13·38	11·92	10·78	9·988	9·160	8·552	8·036	7·580	7·200
380	16·10	14·12	12·58	11·38	10·53	9·669	9·027	8·482	8·001	7·600
400	16·95	14·87	13·24	11·98	10·99	10·180	9·500	8·928	8·420	8·000

Breadths of Belts for different Motive Forces and Angles of Contact—continued.

Motive Force.	Whole Angle of Contact 2 Z.									
	60°	70°	80°	90°	100°	110°	120°	130°	140°	150°
F lb.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.
420	17·80	15·61	13·90	12·58	11·54	10·69	9·97	9·34	8·84	8·40
440	18·64	16·35	14·57	13·17	12·09	11·19	10·45	9·82	9·26	8·80
460	19·49	17·09	15·23	13·77	12·64	11·70	10·93	10·27	9·68	9·20
480	20·34	16·84	15·90	14·37	13·18	12·21	11·40	10·71	10·10	9·60
500	21·19	18·59	16·55	14·97	13·74	12·72	11·88	11·16	10·53	10·00
600	25·42	22·30	19·87	17·96	16·48	15·26	14·25	13·39	12·63	12·00
700	29·66	26·02	23·18	20·95	19·23	17·80	16·63	15·62	14·74	14·00
800	33·90	29·74	26·48	23·96	21·98	20·36	19·00	17·86	16·84	16·00
900	38·13	33·45	29·80	26·95	24·72	22·90	21·38	20·09	18·95	18·00
1000	42·37	37·17	33·11	29·94	27·47	25·44	23·75	22·32	21·05	20·00
1100	46·61	40·89	36·42	32·93	29·22	27·98	26·12	24·55	23·15	22·00
1200	50·84	44·60	39·74	35·92	32·96	30·52	28·50	26·78	25·26	24·00
1300	55·08	48·32	40·05	38·91	35·71	33·06	30·87	29·01	27·36	26·00
1400	59·32	52·04	46·36	41·90	38·56	35·60	33·26	31·24	29·48	28·00
1500	63·56	55·76	49·67	44·89	41·31	38·14	36·63	33·47	31·58	30·00
1600	67·80	59·48	52·96	47·92	43·96	40·72	38·00	35·72	33·68	32·00
1700	72·04	63·20	56·27	50·91	46·71	43·26	40·38	37·95	35·78	34·00
1800	76·26	66·90	59·60	53·90	49·44	45·80	42·76	40·18	37·90	36·00
1900	80·50	70·62	62·91	56·89	52·19	48·34	45·14	42·41	40·01	38·00
2000	84·74	74·34	66·22	59·88	54·94	50·88	47·50	44·64	42·10	40·00
2100	88·98	78·06	69·53	62·87	57·69	53·42	49·88	46·87	44·21	42·00
2200	93·21	81·77	72·84	65·86	60·43	55·96	52·25	49·10	46·31	44·00
2300	97·45	85·49	76·15	68·85	63·18	58·50	54·63	51·33	48·42	46·00
2400	101·70	89·20	79·48	71·84	65·92	61·04	57·00	53·56	50·52	48·00
2500	105·50	92·95	82·75	74·85	68·70	63·50	59·40	55·80	52·65	50·00

Breadths of Belts for different Motive Forces and Angles of Contact—continued.

Motive Force.	Whole Angle of Contact 2 Z.									
	160°	170°	180°	190°	200°	210°	220°	230°	240°	250°
F lb.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.
10	0·190	0·182	0·175	0·163	0·162	0·157	0·152	0·148	0·144	0·140
20	0·381	0·365	0·350	0·337	0·325	0·314	0·304	0·295	0·287	0·280
30	0·571	0·547	0·525	0·505	0·487	0·472	0·457	0·442	0·431	0·420
40	0·762	0·730	0·700	0·673	0·649	0·629	0·609	0·591	0·574	0·560
50	0·952	0·912	0·875	0·841	0·811	0·786	0·761	0·738	0·718	0·700
60	1·143	1·094	1·051	1·010	0·974	0·943	0·913	0·884	0·862	0·840
70	1·333	1·276	1·226	1·178	1·136	1·100	1·065	1·032	1·005	0·980
80	1·524	1·459	1·401	1·346	1·298	1·258	1·218	1·182	1·149	1·120
90	1·714	1·642	1·576	1·515	1·461	1·415	1·370	1·326	1·292	1·260
100	1·905	1·824	1·751	1·683	1·623	1·572	1·522	1·477	1·436	1·400
120	2·286	2·188	2·102	2·020	1·948	1·886	1·826	1·768	1·723	1·680
140	2·667	2·553	2·452	2·357	2·273	2·200	2·130	2·063	2·010	1·960
160	3·048	2·918	2·802	2·692	2·596	2·516	2·436	2·264	2·298	2·240
180	3·429	3·283	3·152	3·029	2·921	2·830	2·740	2·659	2·585	2·520
200	3·810	3·648	3·502	3·366	3·246	3·144	3·044	2·954	2·872	2·800
220	4·191	4·013	3·852	3·703	3·571	3·458	3·348	3·249	3·159	3·080
240	4·572	4·376	4·204	4·040	3·896	3·772	3·652	3·536	3·446	3·260
260	4·953	4·741	4·554	4·377	4·221	4·086	3·956	3·831	3·733	3·540
280	5·334	5·105	4·904	4·714	4·546	4·400	4·260	4·126	4·020	3·820
300	5·715	5·472	5·253	5·049	4·869	4·716	4·566	4·421	4·308	4·200
320	6·096	5·836	5·604	5·384	5·192	5·032	4·872	4·528	4·596	4·480
340	6·477	6·201	5·954	5·720	5·516	5·346	5·176	4·823	4·883	4·760
360	6·858	6·564	6·306	6·060	5·944	5·658	5·478	5·304	5·169	5·040
380	7·239	6·929	6·656	6·397	6·269	5·972	5·782	5·599	5·456	5·320
400	7·620	7·296	7·004	6·732	6·732	6·288	6·088	5·908	5·744	5·600

Breadths of Belts for different Motive Forces and Angles of Contact—continued.

Motive Force.	Whole Angle of Contact 2 Z.									
	160°	170°	180°	190°	200°	210°	220°	230°	240°	250°
F lb.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.	B in.
420	8·001	7·661	7·354	7·068	6·816	6·602	6·492	6·203	6·031	5·88
440	8·382	8·026	7·704	7·406	7·142	6·916	6·696	6·498	6·318	6·16
460	8·763	8·391	8·054	7·743	7·466	7·230	7·000	6·793	6·605	6·44
480	9·144	8·752	8·408	8·080	7·792	7·544	7·304	7·072	6·892	6·52
500	9·525	9·120	8·755	8·415	8·115	7·860	7·610	7·385	7·180	7·00
600	11·43	10·94	10·51	10·10	9·738	9·432	9·132	8·842	8·616	8·40
700	13·33	12·76	12·26	11·78	11·36	11·00	10·65	10·32	10·04	9·80
800	15·24	14·59	14·01	13·46	12·98	12·58	12·18	11·82	11·49	11·20
900	17·14	16·42	15·76	15·15	14·61	14·15	13·70	13·26	12·92	12·60
1000	19·05	18·24	17·51	16·83	16·23	15·72	15·22	14·77	14·36	14·00
1100	20·95	20·06	19·26	18·51	17·85	17·29	16·74	16·25	15·80	15·40
1200	22·86	21·88	21·02	20·20	19·48	18·86	18·26	17·68	17·23	16·80
1300	24·76	23·70	22·77	21·88	21·10	20·43	19·78	19·16	18·67	18·20
1400	26·66	25·52	24·52	23·56	22·72	22·00	21·30	20·64	20·08	19·60
1500	28·56	27·34	26·27	25·24	24·34	23·57	22·82	22·12	21·52	21·00
1600	30·48	29·18	28·02	26·92	25·96	25·16	24·36	22·64	22·98	22·40
1700	32·38	30·90	29·77	28·60	27·58	26·73	25·88	24·12	24·42	23·80
1800	34·29	32·83	31·52	30·29	29·21	28·30	27·40	26·59	25·85	25·20
1900	36·19	34·65	33·27	31·97	30·83	29·87	28·92	28·06	27·28	26·60
2000	38·10	36·48	35·02	33·66	32·46	31·44	30·44	29·54	28·72	28·00
2100	40·00	38·30	36·77	35·34	34·08	33·01	31·96	31·02	30·15	29·40
2200	41·90	40·12	38·52	37·02	35·60	34·58	33·48	32·50	31·60	30·80
2300	43·80	41·94	40·27	38·70	37·22	36·15	35·00	33·98	33·04	32·20
2400	45·72	43·76	42·04	40·40	38·96	37·72	36·52	35·36	34·46	32·60
2500	47·62	44·78	43·79	41·08	40·58	39·29	38·04	36·84	35·90	34·00

To Compute Width of a Leather Belt.—Assuming a well-defined case (where limit of adhesion was ascertained), a belt of ordinary construction (laced), and 9 in. wide, transmitted the power of 15 horses over a pulley 4 ft. diam., at a velocity of 1800 ft. per minute, with an arc of adhesion of 210°, or of ·6, or 7·54 ft. of circumference, and with an area of 95 sq. ft. of belt per h.p.

Hence, $\frac{4400 \text{ to } 5000 \text{ h.p.}}{d\ v} = w$; w representing width of belt in inches, d diameter of pulley in feet, and v velocity of belt in feet per minute.

NOTE.—Thickness of belt should be added to diameter of pulley. Applying these elements to the formulas of 13 different authors, the result varies from 7·85 to 13·5 in., mean of which is 10·675 in. For double belting width $= \cdot 6\ w$. (Haswell.)

Joining Belts.—For all high speeds, it will be found that diagonal riveting will be much more reliable and satisfactory than lacing.

Pulleys.—The built-up wooden pulley has many advantages, chief among them being their combined lightness and strength, and the quickness and cheapness with which they can be built. An ordinary carpenter can turn the belt face of a 10-ft. diam. by 24-in. face belt pulley in its working position, in two hours. If the shaft runs a little eccentrically, the belt face runs perfectly truly, which would not be the case if the pulley had been turned up in the lathe. Again, it is often found that large broad belts stretch more on one side than the other, with the result that the belt tends to run off the pulley; the remedy for this is to turn the crown a little nearer to one side of the pulley than the other. With a wooden rim pulley, this can be done whilst the belt is running, no stoppage being necessary; in fact, the operation can be performed in a few minutes, whilst with the iron pulley it is sometimes a very serious and difficult matter. If the belt face of the pulley is made properly, and of hard and well-seasoned wood, the face will last for 6–8 years, and it is only a question of an hour, perhaps, to trim up the face again, and the pulley will be as new.

The construction of wooden-faced pulleys is shown in Figs. 14, 15. Sizes of 36 in. diam. and upwards have wrought-iron spokes; smaller sizes, sheet-iron discs. The bosses are of cast-iron. In Fig. 14, *a*, cast-iron boss; *b*, sheet-iron discs, ⅛ in. thick; *c*, 1 in. × 1 in. angle-iron rim; *d*, ⅜ in. rivets, countersunk in angle-iron rim; *e*, well-seasoned hardwood segments, 1¼ in. thick in centre, held to angle-iron rim by ¼ in. diam. bolts with nuts and washers *f*. In Fig. 15, *a*, cast-iron boss; *b*, sheet-iron plates; *c*, flat bar-iron spokes, swelled at *d*, by punching bolt-hole hot; *e*, angle-iron rim; *f*, hardwood segments; *g*, strengthening stay-bars used in pulleys over 6·5 ft. diam. (H. L. Short, Trans. Inst. M. M.)

Insufficient mass in a pulley will cause excessive vertical flapping of belts, with consequent rapid destruction. It may be remedied by bolting cast-iron segments on to the underside of the rim.

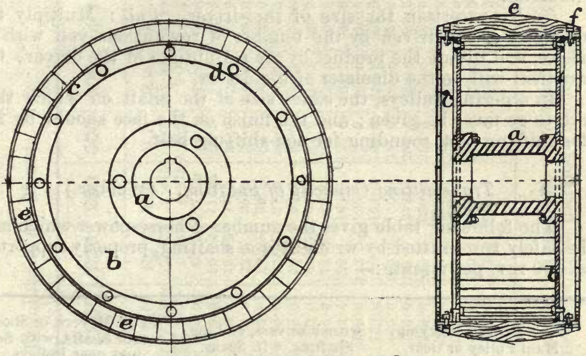


FIG. 14.—WOODEN PULLEY UNDER 36 IN. DIAM.

To Calculate Speed of Drums and Pulleys.

(*a*) The diameter of the driven being given, to find its number of revolutions. *Rule*: Multiply the diameter of the driver by the number of its revolutions, and divide the product by the diameter of the driven; the quotient will be the number of rev. of the driven.

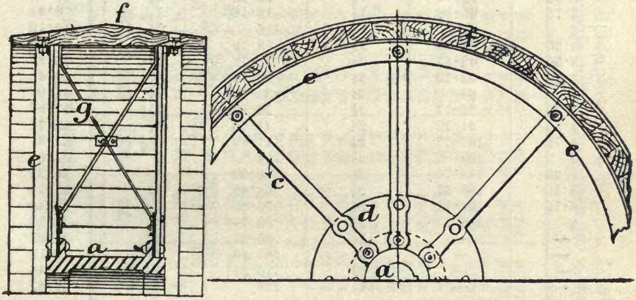


FIG. 15.—WOODEN PULLEY OVER 36 IN. DIAM.

(*b*) The diameter and revolutions of driver being given, to find the diameter of the driven that shall make any given number of revolutions in the same time. *Rule*: Multiply the diameter of the driver by its number of revolutions, and divide the product by the number of revolutions of the driven; the quotient will be the diameter.

(*c*) To ascertain the size of the driver. *Rule*: Multiply the diameter of the driven by the number of revolutions you wish to make, and divide the product by the revolutions of the driver; the quotient will be the diameter of the driver.

In ordering pulleys, the *exact* size of the shaft on which they are to go must be given; and the finish on the face should be flat for shifting belt, rounding for non-shifting belt.

Transmitting Capacity of Shafting. (Webber.)

The following table gives the number of horse-power which may be safely transmitted by wrought-iron shafting, properly supported, at 100 rev. per minute:—

First Movers, Carrying Main Pulley or Gear.		Second Movers, or Line Shafting, 8 ft. Spans.		Third Movers, or Short Counter Shafts, with Bearings near Pulleys.	
Diameter. in.	H.P.	Diameter. in.	H.P.	Diameter. in.	H.P.
1	1	1	2	1	3
1·25	1·95	$1\frac{1}{8}$	2·85	$1\frac{1}{16}$	3·59
1·50	3·37	$1\frac{1}{4}$	3·90	$1\frac{1}{8}$	4·27
1·75	5·36	$1\frac{3}{8}$	5·19	$1\frac{3}{16}$	5·02
2	8	$1\frac{1}{2}$	6·74	$1\frac{1}{4}$	5·85
2·25	11·39	$1\frac{5}{8}$	8·58	$1\frac{5}{16}$	6·78
2·50	15·62	$1\frac{3}{4}$	10·72	$1\frac{3}{8}$	7·79
2·75	20·80	$1\frac{7}{8}$	13·18	$1\frac{7}{16}$	8·91
3	27	2	16	$1\frac{1}{2}$	10·12
3·25	34·33	$2\frac{1}{8}$	19·19	$1\frac{9}{16}$	11·19
3·50	42·87	$2\frac{1}{4}$	22·78	$1\frac{5}{8}$	12·87
3·75	52·73	$2\frac{3}{8}$	26·79	$1\frac{11}{16}$	14·41
4	64	$2\frac{1}{2}$	31·24	$1\frac{3}{4}$	16·07
4·25	76·77	$2\frac{5}{8}$	36·17	$1\frac{13}{16}$	17·86
4·50	91·12	$2\frac{3}{4}$	41·60	$1\frac{7}{8}$	19·77
4·75	107·17	$2\frac{7}{8}$	47·53	$1\frac{15}{16}$	21·81
5	125	3	54	2	24·00
5·25	144·70	$3\frac{1}{8}$	60·92	$2\frac{1}{16}$	26·32
5·50	166·37	$3\frac{1}{4}$	68·66	$2\frac{1}{8}$	28·78
5·75	190·11	$3\frac{3}{8}$	76·89	$2\frac{3}{16}$	31·40
6	216	$3\frac{1}{2}$	85·74	$2\frac{1}{4}$	34·17
—	—	$3\frac{5}{8}$	95·27	$2\frac{5}{16}$	37·09
—	—	$3\frac{3}{4}$	105·46	$2\frac{3}{8}$	40·18
—	—	$3\frac{7}{8}$	116·37	$2\frac{7}{16}$	43·44
—	—	4	128	$2\frac{1}{2}$	46·87

For other velocities, multiply by the number of revolutions and divide by 100.

Diameter and Horse-power of Shafting. Revolutions per Minute.

Diam. of Shaft.	1	50	75	100	110	120	130	150	175	200	225	250	300	400	500	1000
in.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.	H.P.
1	·0099	·495	·742	·990	1·039	1·118	1·287	1·485	1·732	1·980	2·229	2·475	2·97	3·960	4·990	9·9
1¼	·0175	·875	1·3125	1·75	1·925	2·10	2·275	2·625	3·062	3·5	3·937	4·375	5·25	7·0	8·75	17·5
1½	·03	1·50	2·25	3·0	3·30	3·60	3·90	4·50	5·25	6·0	6·75	7·50	9·0	12·0	15·0	30·0
1¾	·045	2·25	3·375	4·5	4·95	5·4	5·85	3·75	7·875	9·0	10·125	11·25	13·5	18·0	22·5	45·0
2	·07	3·5	5·15	7·0	7·7	8·4	9·1	10·5	12·5	14·0	15·75	17·5	21·0	28·0	35·0	70·0
2¼	·1	5·0	7·5	10·0	11·0	12·0	13·0	15·0	17·5	20·0	22·5	25·0	30·0	40·0	50·0	100·0
2½	·130	6·5	9·75	13·0	14·3	15·6	16·9	19·5	22·75	26·0	29·25	32·5	39·0	52·0	65·0	130·0
2¾	·165	7·5	11·25	15·0	16·5	18·0	19·5	22·5	26·25	30·0	33·75	37·5	45·0	60·0	75·0	150·0
3	·225	11·25	16·875	22·5	24·75	27·0	29·25	33·75	39·375	45·0	50·6	56·2	67·5	90·0	112·5	225·0
3¼	·275	13·75	21·62	27·5	30·25	32·0	34·75	41·25	47·75	55·0	61·875	68·75	82·5	110·0	137·5	275·0
3½	·33	16·5	24·75	33·0	36·3	40·20	42·9	49·5	57·75	66·0	74·25	82·5	99·0	132·0	165·0	330·0
3¾	·412	20·6	30·9	41·2	45·32	49·44	53·56	61·8	71·10	82·4	92·7	103·0	123·6	164·3	206·0	412·0
4	·5	25·0	37·5	50·0	55·0	60·0	65·0	75·0	87·5	100·0	112·5	125·0	150·0	200·0	250·0	500·0
4¼	·6	30·0	45·6	60·0	66·0	72·0	78·0	90·0	105·0	120·0	135·0	150·0	180·0	240·0	300·0	600·0
4½	·725	35·25	54·27	72·5	79·75	86·0	94·25	108·70	126·87	145·0	163·0	180·0	217·0	280·0	350·0	725·0
4¾	·85	42·5	63·75	85·0	93·5	102·0	110·0	127·0	143·0	170·0	191·0	212·0	255·0	340·0	425·0	850·0
5	1·0	50·0	75·0	100·0	110·0	120·0	130·0	150·0	175·0	200·0	225·0	250·0	300·0	400·0	500·0	1000·0
5½	1·325	66·0	99·0	132·0	145·0	159·0	172·0	198·0	232·0	265·0	298·0	331·0	397·0	530·0	662·0	1324·0
6	1·725	86·0	139·0	172·0	189·0	207·0	224·0	258·0	301·0	345·0	388·0	431·0	517·0	690·0	862·0	1724·0
6½	2·175	108·0	163·0	217·0	239·0	261·0	282·0	326·0	380·0	435·0	489·0	543·0	652·0	870·0	1087·0	2174·0
7	2·7	135·0	202·0	270·0	297·0	324·0	351·0	405·0	472·0	540·0	607·0	675·0	810·0	1080·0	1350·0	2700·0

This table is calculated for general shafting, transmitting power by belt pulleys. For shafting carrying heavy weights, or transmitting power by gears, diameter should be increased accordingly.

ROPE DRIVING.

It has been found that 200 lb. on a 1-in. rope is a safe and economical working load, and when materially increased the wear is rapid. Test pieces of manila rope made at different works varied slightly in diameter, but when reduced to the equivalent of 1 in. diameter had an average breaking strength of 7140 lb. Expressed algebraically, the breaking strength, weight per foot, and the working strains are—

$$W = 720\ C^2\ (1);\ P = \cdot 32\ C^2\ (2);\ w = 20\ C^2\ (1)$$

In these and the following equations:—

C = circumference of rope in inches.
D = sag of the rope in inches.
F = centrifugal force in lb.
g = gravity.
H = horse-power.
L = distance between pulleys in ft.
P = lb. per ft. of rope.
H = force in lb. doing useful work.
S = load in lb. on rope at pulley.
T = tension in lb. on driving side of the rope.
t = tension in lb. on slack side of the rope.
v = velocity of rope in ft. per second.
w = working load in lb.
W = ultimate breaking load in lb.

This makes the normal working load equal to $\frac{1}{36}$ the breaking strength, and about $\frac{1}{25}$ of the strength at the splice. The actual loads are ordinarily much greater, owing to the vibrations in running, as well as from imperfectly adjusted tension mechanism.

Assuming that the load on the driving side of a rope is equal to 200 lb. on a rope 1 in. diam., and that the rope is in motion at various velocities of 10 ft. to 140 ft. per second. Then we will have in all cases a fibre load of 200 lb. on the driving side of a 1-in. rope, and an equivalent load for other sizes. The centrifugal force of the rope in running over the pulley will reduce the amount of force available for the transmission of power. The centrifugal force of the rope is computed by the formula—

$$F = \frac{P\,v^2}{g}. \qquad (2)$$

At a speed of about 80 ft. per second, the centrifugal force increases faster than the power from increased velocity of the rope, and about 140 ft. per second equals the assumed allowable tension of the rope. Computing this force at various speeds and then subtracting it from the assumed maximum tension, we have the force available for the transmission of power. The whole of this force cannot be used, because a certain amount of tension on the slack side of the rope is needed to give adhesion to the pulley. What tension should be given to the rope for this purpose is uncertain, as there are no experiments which give accurate data, and at the present time a decision must be made partly from analogy and partly from experience.

If the rope be considered as a belt on a plain pulley, the friction would be substantially the same as a leather belt at the same tension; but as ropes are frequently lubricated to reduce the wear,

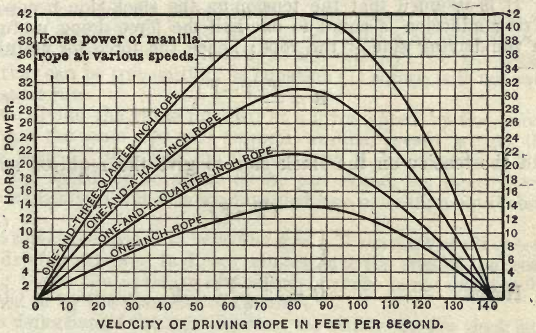


FIG. 16.

the coefficient of friction must be materially reduced. There have been no experiments to decide with accuracy what this reduction is, but it is known from considerable experience that when the rope runs in a groove whose sides are inclined towards each other at an angle of 45° there is sufficient adhesion when the ratio of the tension is

$$\frac{T}{t} = 2. \qquad (3)$$

For the present purpose, T can be divided into three parts: Tension doing useful work, tension from centrifugal force, tension to balance the strain for adhesion. The tension t can be divided into two parts: Tension for adhesion, tension from centrifugal force. It is evident, however, that the tension required to do a given work should not be materially exceeded during the life of the rope.

There are two methods of putting ropes on the pulleys; one in which the ropes are single and spliced on, being made very taut at first, and less so as the rope lengthens, stretching until it slips, when it is respliced. The other method is to wind a single rope over the pulley as many turns as needed to obtain the necessary horse-power, and put a tension pulley to give the necessary adhesion, and also to take up the wear. The total tension T on the driving side of the rope is assumed to be the same at all speeds. The centrifugal force, as well as an amount equal to the tension for adhesion on the slack side of the rope, must be taken from the total tension T to ascertain the amount of force available for the transmission of power.

It is assumed that the tension on the slack side necessary for giving adhesion is equal to one-half the force doing useful work on the driving side of the rope; hence the force for useful work is—

$$R = \frac{2(T-F)}{3}, \tag{4}$$

and the tension on the slack side to give the required adhesion is—

$$\frac{(T-F)}{3}. \tag{5}$$

Hence,

$$t = \frac{(T-F)}{3} + F. \tag{6}$$

The sum of the tensions T and t is not the same at different speeds, as the equation (6) indicates.

As F varies as the square of the velocity, there is, with an increasing speed of the rope, a decreasing useful force, and an increasing total tension t on the slack side. With these assumptions of allowable stresses, the horse-power will be—

$$H = \frac{2v(T-F)}{3 \times 550}. \tag{7}$$

Transmission ropes are usually 1–1¾ in. diam. A computation of the horse-power for four sizes at various speeds and under ordinary conditions, based on a maximum load equivalent to 200 lb. for a rope 1 in. diam., is given in Fig. 16. The horse-power of other sizes is readily obtained from these. The maximum power is transmitted, under the assumed conditions, at a speed of about 80 ft. per second.

The first cost of the rope will be smallest when the power transmitted by it is greatest, and, under the assumed conditions, will be a minimum for a given power when the velocity of the rope is about

80 ft. per second. The ratio of the first cost of the rope running at any other speed will be—

$$\text{Ratio of first cost} = \frac{\text{H at 80 ft. per second}}{\text{H at required speed}}. \quad (8)$$

The wear of the rope is both internal and external; the internal is caused by the movement of the fibres on each other, under pressure in bending over the sheaves; and the external is caused by the slipping and the wedging in the grooves of the pulley. Both of these causes of wear are, within the limits of ordinary practice, assumed to be directly proportional to the speed. Hence, if we assume the coefficient of the wear to be k, the wear will be $k\,v$, in which the wear increases directly as the velocity; but the horse-power that can be transmitted, as equation (7) shows, will not vary at the same rate.

If we divide the value for wear at a given speed by the horse-power that the same rope will transmit at other speeds, we get the relative wear of the rope in transmitting 1 horse-power. The higher the speed up to about 80 ft. per second, the more power will be transmitted, but it is accompanied by a more than equivalent wear.

The rope is supposed to have the strain T constant at all speeds on the driving side, and in direct proportion to the area of the cross-section; hence the catenary of the driving side is not affected by the speed or by the diameter of the rope.

The deflection of the rope between the pulleys on the slack side varies with each change of the load or change of the speed, as the tension equation (8) indicates. The deflection or sag of the rope may be computed for the assumed value of T and t by the parabolic formula—

$$S = \frac{P\,L^2}{8\,D} + P\,D \quad (9)$$

S being the assumed strain T on the driving side, and t, calculated by equation (6), on the slack side.

It is to be regretted that accurate data are not available to determine the constants needed in the equations for wear and for friction on the pulley. (C. W. Hunt.)

The American or "continuous" system—that of winding a long rope round and round the pulleys, and then carrying it from side to side as it completes the circuit, by means of a jockey pulley fixed at the required angle, necessitates a series of deflections from the straight driving path, causing the rope to assume the form of an elongated spiral, and setting up a one-sided pressure in the grooves. If the weight upon the jockey pulley is so arranged as to merely take up the slack, and balance the driving tension of that part of the rope which it controls, then the frictional loss due to carrying the rope across the pulleys is not a very damaging factor; but when it is overloaded with a view to levelling up the entire drive, the strain is very materially increased, without accomplishing its object. Doubtless the greatest hindrance to adoption of the American system for all purposes is the fact that dependence has to be placed upon one rope; should that fail, the driving must stop until it is replaced.

With the separate rope system, an excess of the actual power required being usually provided, the replacement of a rope may await a convenient season, and that without detriment.

But there are circumstances where continuous rope driving may be adopted with advantage over other methods. An advocate of continuous driving recommends pulleys of not less than 60 times the diameter of the ropes.

The absolute point of detraction in power from the employment of relatively small pulleys cannot, of course, be determined with mathematical accuracy, by reason of the elasticity of the rope, which varies with almost every make. We must, therefore, be content to declare a position between the extreme limit where the bending faculty of a rope ceases to exert its influence, and the radius controlling the highest capability. By reason of the great disparity in elasticity between cotton and manila, what is known as "permanent set" (i.e., where elasticity altogether ceases) being reached at a very much earlier stage in the tension of the last mentioned, it has been found necessary to fix the smallest pulley diameter at 50 % greater than that of a cotton rope. For the efficient transmission of power, the smallest pulley used with cotton ropes should not be less than 30, and with manila 45 diameters, unless extra rope power is added to make up for loss of grip. Nothing is so detrimental to driving ropes as pulleys too small in diameter; the internal compression caused by bending quickly around a small pulley very soon breaks up any rope.

At one time it was an accepted theory with many engineers that grooves should bear some resemblance to the rope itself, and therefore curved sides were introduced in the belief that they afforded a necessary easement to the rope when leaving the grooves. When grooves of this description are employed, it is generally necessary to increase the diameter of the rope to the utmost limit, not only to make up for loss of power, but to prevent, as far as possible, the rolling action often induced.

The angles at which driving-rope grooves are constructed vary as much as from 54° to 15°, the latter being applied to driving with small bands; and from the results obtained it would appear that every diameter of rope has its most appropriate angle. While this need not be carried to the extent of providing a range of templets to cover all sizes, some line of demarcation between the acute and obtuse is advisable.

One firm of engineers works to four different angles, beginning with 30° for small ropes $\frac{5}{8}$ in.–$\frac{7}{8}$ in. diam., 36° for 1–1$\frac{1}{4}$ in., 40° for 1$\frac{3}{8}$–1$\frac{5}{8}$ in., and 45° for 1$\frac{3}{4}$–2 in. Good average practice is 30° for all under 1$\frac{1}{4}$ in., and 40° for all above that. (E. Kenyon.)

The true V-shape is not desirable, because the rope, after working for a time, would become wedge-shaped, losing power and shortening its life. The groove should be so constructed as to avoid all possible chance of the rope reaching the bottom, which is generally rounded to a curve, though in Continental practice the grooves are often flat-bottomed. If a rope rubs on the bottom, it has a strong tendency to slip.

Jockey Pulleys.—Carrier or jockey pulleys should be introduced very judiciously; the life of many a rope has been shortened by their injudicious introduction. No general rules can be laid down as to whether it is wise or not to introduce a carrier, or the best place to put a carrier, so much depends upon circumstances. It is usual to make the grooves of carrier pulleys U-shape (quite unlike those used for driving pulleys), so that the rope will not wedge at the sides, but ride on the bottom. Sometimes in fixing carriers to change the direction of travel of a rope, the pulleys have to be set in such a position that the rope will run to some extent over the flanges. In such a case, the groove may be made to an angle of 70°.

Forward and Backward Driving.—The slack or idle side of the ropes should be the top side, with the driving side at the bottom. In this way, the arc of contact on the driven pulley is considerably increased by the "sag" of the rope; this is known as "forward driving." If the ropes have to work under the reverse conditions (the slack side at the bottom), known as "backward driving," they have a tendency to hang off the pulleys, thus lessening the grip, and reducing the effectiveness of the drive. In this case, a considerable margin of power should be arranged for, and plenty of room left under the ropes to allow them to "sag" freely.

Angle of Drive.—Best results are obtained with the ropes working either horizontally or at an angle up to 45°; above this angle, they hang off the pulley. Where compelled to work at above this angle, it is advisable to have a margin of power. Vertical drives are not to be recommended, though several are working successfully with a good margin of rope power. They are sometimes arranged on the "continuous" system.

Length of Drive.—Though ropes are specially adapted for long drives, cotton ropes have been fixed on drives, with both large and small pulleys, where the pulley faces have only been 1–4 ft. apart. On the other hand, there are many drives with shafts 40–80 ft. apart, giving very satisfactory results; but for such long drives it is almost imperative that the pulleys be large. For long drives with only small pulleys, carriers can be successfully, if judiciously, introduced. In one drive which has come under notice, power is transmitted by a single rope over 300 ft., with a two-grooved carrier pulley every 30 ft.

Thickness of Ropes.—To decide the most suitable diameter of rope for a drive, many points have to be taken into consideration, the chief being the diam. of pulleys that can be used. If pulleys are to be less than 3 ft. diam., it is best for the ropes not to exceed $1\frac{1}{4}$ in. diam.; with pulleys 5 ft. diam., $1\frac{3}{4}$ in. ropes are generally used. Ropes above $2\frac{1}{4}$ in. diam. are rare, except for drives of one or two ropes only, where very little space is available for width of pulley, but plenty of space for a pulley of large diameter.

Power Transmitted.—A rope *will* transmit much more power than is advantageous or advisable. An instance might be quoted of 3 $2\frac{1}{8}$-in. ropes transmitting 600 h.p., but this is simply rope destruction. It is far better and more economical to err on the side of too little load. Following is a simple formula for finding h.p.—Let d = rope diam. (in.), v = speed (ft. per sec.), w = weight of 1 ft. rope (lb.), s = effective stress. For cotton rope, a safe stress is $160d^2$. Centrifugal stress $= \frac{w\,v^2}{32}$. Hence, effective stress $= 160d^2 - \frac{w\,v^2}{32}$; and h.p. $= \frac{v\,s}{550}$. The table (page 101) is reliable for ordinarily favourable conditions; with backward driving, excessive angle, small or unequal pulleys, a margin must be allowed.

Speed.—Perhaps the most economical speeds are 3600–5000 ft. per min.; in many cases it is impossible to attain even 3000 ft. For speeds of 2000 ft., it is advisable to use thicker ropes, provided the pulleys are of sufficiently large diameter. There are ropes working satisfactorily at speeds above 5200 ft. per min., though it is hardly advantageous, as the rope travelling straight, according to the first law of motion, will not easily lend itself to bending around the pulley. At such an extreme speed as 7000 ft. per min., 3–4 min. must be occupied in starting and stopping the drive; if started or stopped suddenly, the ropes would undoubtedly pull asunder. Also it must be remembered that at such a speed the rope cannot transmit as much power as at, say, 4800 ft. per min., owing to centrifugal action, so such high speeds are not economical.

Tightness.—This depends upon circumstances. When the slack side is at the bottom, the ropes must of necessity be kept tighter than when the slack is at the top. It is best to have them at such a tension as will just do the work without slipping; the tighter a rope has to be kept, the shorter is its life. When new ropes are being fixed, the splicer must use his own judgment as to the tension, bearing in mind the strength of shafts and bearings. It is often cheaper to have ropes tightened after they have been running a short time, than to risk bending a weak shaft in putting them on. Every rope should be well strained before being spliced up for a drive.

Slipping.—When a rope slips on the driven pulley, it means that it has not sufficient gripping force on the pulley to do its work. The gripping force is reduced if the rope is too small for the

Horse-power of Cotton Driving Ropes. (C. N. Pickworth.)

Velocity in ft. per min.	Diameter of Ropes (in.).									
	$\frac{3}{4}$	$\frac{7}{8}$	1	$1\frac{1}{8}$	$1\frac{1}{4}$	$1\frac{3}{8}$	$1\frac{1}{2}$	$1\frac{5}{8}$	$1\frac{3}{4}$	2
1000	2·5	3·5	4·5	5·7	7·1	8·6	10·2	12·0	14·0	18·2
1100	2·7	3·9	4·9	6·2	7·8	9·4	11·2	13·4	15·4	20·0
1200	3·0	4·3	5·4	6·8	8·5	10·3	12·2	14·4	16·8	21·7
1300	3·2	4·7	5·8	7·4	9·2	11·2	13·2	15·6	18·2	23·7
1400	3·5	5·0	6·3	8·0	9·9	12·0	14·3	16·8	19·6	25·5
1500	3·7	5·4	6·7	8·5	10·6	12·9	15·3	18·0	21·0	27·1
1600	4·0	5·8	7·2	9·1	11·4	13·8	16·3	19·2	22·4	29·1
1700	4·2	6·1	7·6	9·7	12·1	14·6	17·3	20·4	23·8	31·0
1800	4·5	6·5	8·1	10·2	12·8	15·5	18·4	21·6	25·5	32·8
1900	4·8	6·8	8·6	10·8	13·5	16·4	19·4	22·8	26·7	34·6
2000	5·1	7·0	9·1	11·5	14·2	17·3	20·5	24·1	28·0	36·5
2100	5·3	7·3	9·5	12·1	14·9	18·0	21·4	25·1	29·2	38·1
2200	5·6	7·6	9·9	12·6	15·5	18·8	22·3	26·2	30·4	39·7
2300	5·8	7·9	10·2	13·0	16·1	19·4	23·1	27·1	31·5	41·1
2400	6·0	8·1	10·6	13·4	16·6	20·1	23·9	28 1	32·6	42·5
2500	6·2	8·4	11·0	13·9	17·2	20·8	24·7	29·0	33·7	44·0
2600	6·4	8·7	11·3	14·4	17·8	21·5	25·5	30·0	34·8	45·4
2700	6·6	8·9	11·7	14·8	18·3	22·1	26·3	30·9	35·9	46·8
2800	6·8	9·2	12·0	15·2	18·8	22·8	27·1	31·8	36·8	48·1
2900	6·9	9·4	12·3	15·6	19·3	23·3	27·8	32·5	37·8	49·3
3000	7·1	9·6	12·6	16·0	19·8	23·9	28·4	33·3	38·7	50·4
3100	7·3	9·9	12·9	16·3	20·2	24·4	29·0	34·1	39·6	51·6
3200	7·4	10·1	13·1	16·6	20·6	24·9	29·6	34·8	40·4	52·7
3300	7·6	10 3	13·4	17·0	21·0	25·4	30·2	35·4	41·2	53·8
3400	7·7	10·6	13·7	17 3	21·5	26·0	30·8	36·2	42·0	54·6
3500	7·8	10·7	13·9	17·6	21·8	26·4	31·4	36·8	42·8	55·8
3600	8·0	10·8	14·1	17·9	22·1	26·8	31·8	37·4	43·3	56·5
3700	8·1	11·0	14·3	18·2	22·4	27·1	32·3	37·9	44·0	57·3
3800	8·2	11·1	14·5	18·4	22·7	27 5	32·7	38·4	44·5	58·2
3900	8·3	11·3	14·7	18 5	23·0	27·8	33·1	38·8	45·0	58·8
4000	8·4	11·4	14·8	18·7	23·2	28·1	33·4	39·2	45·5	59·4
4100	8·4	11·5	15·0	19·0	23 5	28·4	33·7	39·6	46·0	60·0
4200	8·5	11·5	15·1	19·1	23·7	28·6	34·0	39·9	46·3	60·4
4300	8·6	11·6	15·2	19·2	23·8	28·8	34·2	40·2	46·6	60·8
4400	8·6	11·7	15·3	19·3	23·9	28·9	34·4	40·4	46·8	61·2
4500	8·7	11·7	15·3	19·4	24·0	29·0	34·5	40·5	47·0	61·4
4600	8·7	11·8	15·4	19·4	24·1	29·1	34·6	40·6	47·1	61·5
4700	8·7	11·8	15·4	19·5	24·1	29·2	34·7	40·7	47·2	61·6
4800	8·7	11·8	15·4	19·5	24·2	29·2	34·7	40·8	47·3	61·7
4900	8·7	11·8	15·4	19·5	24·1	29·2	34·7	40·7	47·2	61·6
5000	8·7	11·8	15·4	19·4	24·1	29·1	34·6	40·6	47·1	61·5
5100	8·7	11·7	15·3	19·4	24·0	28·9	34·5	40·5	47·0	61·2
5200	8·6	11·7	15·2	19·3	23·9	28·8	34·3	40·2	46·7	61·0
5300	8·5	11·6	15·1	19·2	23·7	28·7	34·1	40·0	46·4	60·6
5400	8·5	11·5	15·0	19·0	23·5	28·4	33·8	39·6	46·0	60·0
5500	8·4	11·4	14·8	18·8	23·2	28·1	33·4	39·2	45·5	59·4

groove, but the most frequent cause of slipping is overloading the drive, such overloading being often caused by a tight bearing or a shaft being out of truth. If a rope be put on very tight, its gripping force is increased, but its life may be shortened in consequence of the strain.

To prevent slipping, it is very important that the rope be of such diameter as will best fit the working part of the groove. In drives where the driving pulley is very much larger than the driven pulley, or *vice versa*, there is a tendency for the ropes to slip, owing to the small arc of contact they have with the small pulley; in such drives, a large margin of rope-power should be provided, and every care taken that they fit the grooves perfectly.

Revolving.—It is difficult to say what is the cause of some ropes revolving and others not revolving. It often happens that of a set of ropes on one drive some will revolve and others will not. If a rope revolves, the wear is more even over its entire surface, and it can be tightened more easily than if it has not revolved.

Cross Drives.—Cross driving, though common with belts, has not until recently been successfully employed with ropes, the difficulty being that the forward and return ropes in a cross drive have to pass each other in a space designed only for one. Whilst open drives are to be recommended, ropes can be successfully crossed if fixed on the principle that, while each rope is separate (as in the ordinary open-drive system), the ropes work in pairs, one rope of each pair being crossed right-handed, the other left-handed, with the result that the two tight sides run in the centre and the two slack sides form the outside, of each pair. As the two tight sides run in the same direction, they cannot chafe each other; and the two slack sides, by reason of their sagging in work, readily open out and allow the tight sides to pass. Between each pair, it is advisable to leave an empty groove, thus allowing the necessary room for the sagging of the slack sides.

Splicing.—The following instructions refer to cotton ropes. They are issued by a first-class maker, T. Hart, Blackburn.

Ropes should be stored in a dry room, and never on an earthen floor. When wanted, uncoil them left-handed and stretch out with a pair of hand blocks, but do not allow the rope to revolve during this operation, or the "turn" or "twist" will be taken out of it. Everything depends upon the strength of shafting, etc., as to how tight the rope should be stretched, but, under ordinary circumstances, it is usual to have about 4 men pulling on a pair of hand blocks. Pass a string around the pulleys to get the exact running length of rope, and measure the rope whilst it is on the stretch, allowing 10 ft. (5 ft. at each end) for the splice. For ropes 2-in. diam., use 12 ft. for splice.

Cut off the rope, and tie a band around it 5 ft. from each end;

unlay the rope from the ends to these bands; it is also advisable to tie a band around the end of each strand, to prevent the "turn" or "twist" from coming out. Cut out the small centre cord on which the four strands have been laid, and butt the ends of the rope together. It should now be as in A, Fig. 17.

Cut one of the bands, then unlay one of the strands (1*a*), at the same time laying in its place the opposite strand (1). Do this for about 4 ft., and tie the two strands temporarily. The next strand to this is numbered 2, but do not touch this now; take the next strand but one, numbered 3*a*, and unlay it, laying in its place strand No. 3, but only for about 1 ft. 6 in.; tie these temporarily. It is very important in splicing that two strands next to each other should not be laid up in the same direction.

Next cut the other band, and proceed in the same manner with strands Nos. 2 and 4, but of course in the opposite direction. Care should be taken to keep the "turn" or "twist" in the strands, and they should be laid well and evenly down in their places. Shorten the strands to equal lengths of about 1 ft., and the splice should now be as in Fig. B.

Then begin at joint of strands 1 and 1*a*; untie the temporary knot, and remove the 10 friction bands (outside threads) from 1*a*, but do not cut them off; unlay strand No. 1 back two turns or laps, and remove the friction bands, but do not cut them off; lay in the tension strand (inside strand) of No. 1 one turn or lap, and reduce it by cutting out about one quarter; lay up the remainder the one turn or lap, thus bringing it up to tension strand of 1*a*, then tie the two with an over-hand knot. At this knot the rope should be of its original diameter, and the joint should appear as in Fig. C.

Take tension strand 1*a* and, with the splicing pin, work it under and over tension strand No. 1, as shown in Fig. D, passing it under and over spirally about 5 times; it will then have reached friction bands No. 1. Now reduce it by cutting out a portion, and pass the remainder between the two lots of 5 friction bands, and pass once or twice through the centre of the rope. This is shown in Fig. E. Five friction bands are also locked through the rope as shown. This complete, turn to three-quarters of tension strand No. 1; this is locked exactly as tension strand No. 1*a*, five of the friction bands are also locked as previously shown, and the splice should now be as in Fig. E.

All loose ends are cut off, and this portion is complete. Treat the three other joints in the same way, and the splice is finished.

With a 3-strand rope, one strand is run to the right, another to the left, the third strand remaining with its fellow strand at the butt joint; then follow the same process of tucking in as with a 4-strand rope.

Putting Rope on Pulley.—Put the rope on the small pulley, and as far around the large pulley as it will go; lash it there, but have the lashing so that the rope has freedom to slip through it. Then bar pulley slowly round by hand, and the rope will fall into its place. The groove of the small pulley, and the face of the large

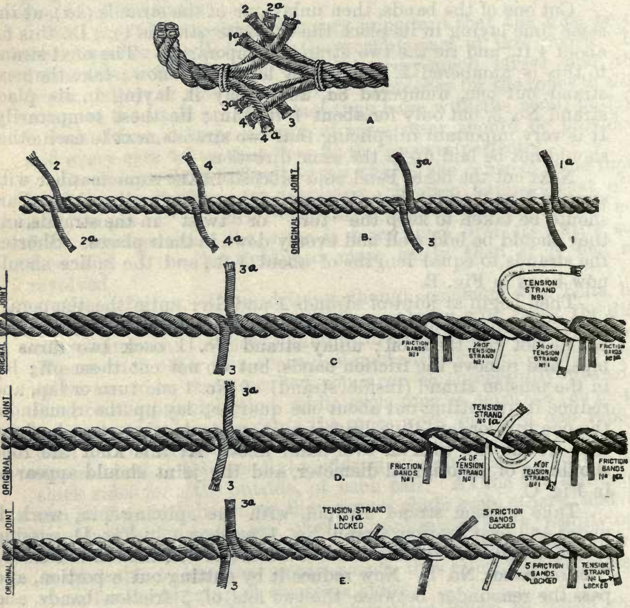


FIG. 17.—SPLICING COTTON DRIVING ROPES.

pulley where the rope will touch in barring on, should be well greased with tallow to prevent wedging. A piece of canvas may also be put on the rim of the large pulley to prevent it cutting the rope.

Wire Rope Drive for Mines. (G. D. Rice.)

Fig. 18 shows a section of the main jack-wheel and connections designed for operating five rope-drives from a central tower. The jack-wheel is keyed to the shaft *a*, and this shaft bears in journals which are bolted to the cross-beams of a wooden frame. The belt pulley *b* is fastened to the same shaft, and driven by the belt *c*.

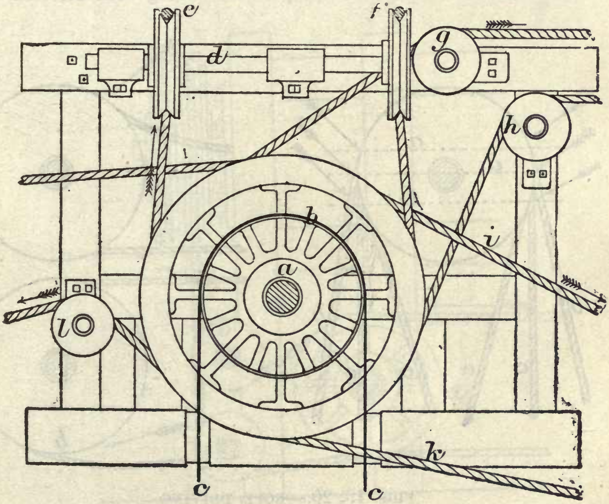
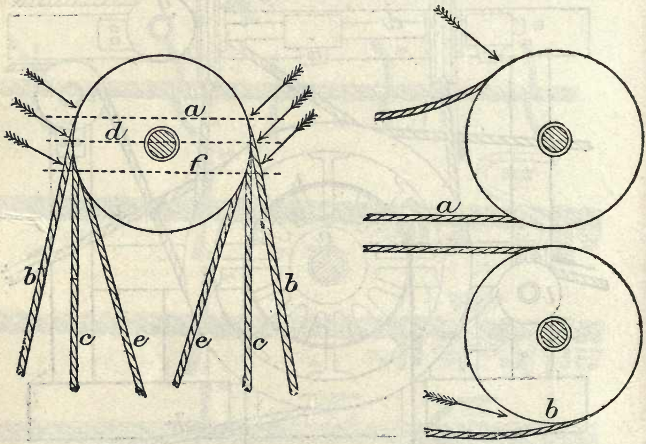


FIG. 18.—ROPE DRIVING.

The latter goes down to the driving-wheel of the engine below. Hangers are attached at either side of the jack-wheel to the top cross-beam of the frame, for shaft *d* and guide-wheels *e f* run on this shaft. These guide-wheels take the rope from one of the outside grooves of the jack-wheel, and deliver it from the tower at an angle. A rope may be run from the opposite side on same plan. Carrier-wheels *g h* are adjusted with the view of taking the rope almost straight from the jack-wheel. Ropes *i k* are calculated to go straight out from one of the middle grooves of the jack-wheel. A similar drive is provided for on the opposite side. Carrier-wheel *l* may be used if necessary. As in belt installations, it is best not to use idlers, carriers, or tighteners, when they can be avoided. Wheels in which the grooves are V-shaped are recommended, as then the rope gets a good grip by being forced against the sides of the groove. Oval grooves are advantageous under certain conditions.

Whatever the form of groove or type of rope used, the system will not give satisfaction unless the essential points of adjustment are attended to. Wheels of proper size having been selected for



FIGS. 19, 20.—ROPE DRIVING.

driving, as much of the arc of contact of the rope should be utilised as possible. If the arc of contact of the rope is only as much as indicated at *a*, Fig. 19, the service will very likely be weak and uncertain, owing to the slipping of the rope. The angle at which this rope extends from the wheel is shown by *b b*. The arc of contact of the rope *c* is increased to *d*, and much more effective service is obtained with the rope no tighter. The arc of contact of rope *e* reaches to *f*, and of course is more than can be obtained except in special cases. However, the arc of contact can be increased oftentimes by planning the lower side of the wheel to be the driving and consequently the tight side, as at *a*, Fig. 20, in which it is seen that the sag of the rope increases the diameter of contact to the point indicated by the arrow. If the conditions are reversed, the sag of the rope decreases the arc of contact, as indicated by the arrow at b, Fig. 20, and much of the surface contact is lost.

Rope drives intended for power transmission in mines must necessarily take many turns in underground passages. Fig. 21 is a simple design for this purpose. Wheel a is studded to a journal on one of the timbers as shown. This wheel has two grooves, one of which receives the rope from along the line of the passage, while the other carries the rope which passes under guide-wheel b, up

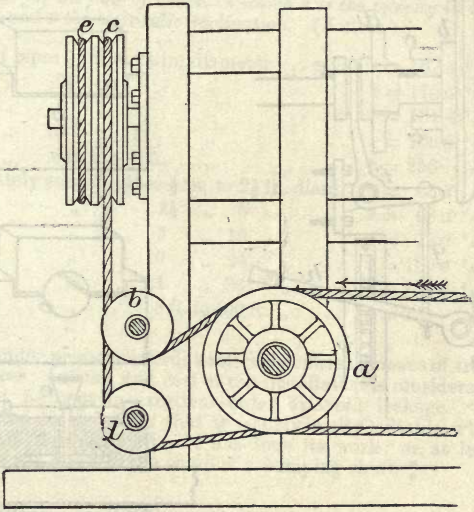
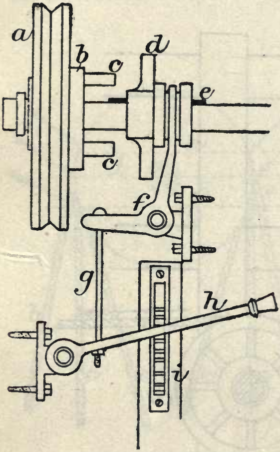
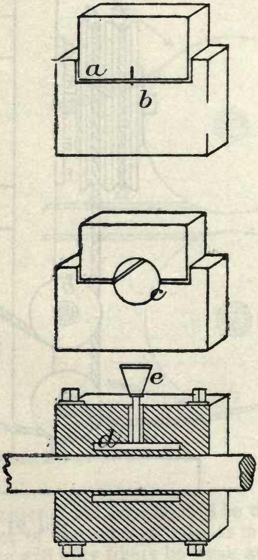


FIG. 21.—ROPE DRIVING.

over the upper wheel in groove c, and down under wheel d. The upper wheel is also two-grooved, and the rope in groove e runs along the line of the angle of the other passage. This requires three ropes, but the number may be reduced to two by having the drive rope go direct to the wheels b c, instead of around the wheel a first. For light work and short drives, one rope can be put around by using idlers.

There must be disconnecting devices at intervals, and the simplest form consists in running two grooved wheels together, one tight, the other loose. The flange of the loose pulley next the tight pulley should be very low. A gap should be cut in the flange of the tight pulley next the loose pulley, and the rope be guided through this gap with a stick. A lever disconnector is shown in Fig. 22. The wheel *a* is loose on the shaft, and is provided with hub *b*, in which the pins *c* are fixed. The pronged piece *d* is forged and bored out to fit the shaft loosely. Key *e* is firm in its seat in the shaft, but fits loosely in the key-way of the forged piece. This permits the latter to be shifted into clutch with





FIGS. 22, 23.—ROPE DRIVING.

the pins *c* and out again at will. A lever *f* is provided with rod *g*, and connection is made with the hand lever *h*, sustained in proper place by means of ratchet *i*.

As wood bearings are peculiarly serviceable for shafts of rope drives in underground work, the process of making them is appended. Good, dry, close-grained timber is selected, and the base and cap pieces are sawn out, with the latter sunk into the former, as shown at *a*, Fig. 23. A centre line is then drawn around both pieces, and a straight gauge is started for the boring tool by a saw-cut, about 1 in. deep in each, as at *b*, which the point of the auger will follow. When bored it appears as at *c*. If the box is to be used plain, bolt-holes only are needed; if to be babbited, the centre is cut out about ¼ in. deep, and to within 1 in. of the sides, as at *d*, and through an iron funnel *e* in the oil hole, metal is poured in. (En. & Min. Jl.)

HYDRAULIC.

Formulæ for the flow of water, in which v *is the velocity in inches per second, and* s *the hydraulic inclination.* (Kutter.)

For smooth pipes ½ in. to 2½ in. diameter	..	..	$v = 107\, d^{\cdot 9}\sqrt{s}$
„ „ 2½ „ 5 „	..	..	$v = 115\, d^{\cdot 8}\sqrt{s}$
„ „ 5 „ 10 „	..	..	$v = 134\, d^{\cdot 7}\sqrt{s}$
„ „ 10 „ 72 „	..	..	$v = 166\, d^{\cdot 6}\sqrt{s}$
„ „ 6 ft. to 400 ft. „	..	..	$v = 256\sqrt{d\,s}$
For moderately smooth pipes ½ in. to 2½ in. diam.			$v = 63\, d\sqrt{s}$
„ „ 2½ „ 5 „		..	$v = 68\, d^{\cdot 9}\sqrt{s}$
„ „ 5 „ 10 „		..	$v = 78\, d^{\cdot 8}\sqrt{s}$
„ „ 10 „ 24 „		..	$v = 100\, d^{\cdot 7}\sqrt{s}$
„ „ 24 „ 96 „		..	$v = 138\, d^{\cdot 6}\sqrt{s}$
„ „ 8 ft. to 400 ft. „		..	$v = 221\sqrt{d\,s}$

Water under pressure affords a tolerably efficient means of transmitting power; but the first cost of the installation is considerable, and it must be kept in excellent order to avoid leakage. The great objection to its use is that it is only really suitable where the water can flow away after it has done its work; or, at least, where it can run down to the sump of a pumping shaft.

PNEUMATIC.

Air Compression.—Air is an elastic fluid, which, when free from vapour, behaves as a gas; 13·09 cub. ft. at ordinary atmospheric pressure, and at 60° F., weigh 1 lb. According to Boyle's law, the volume of a gas varies inversely as the pressure affecting it, so long as the temperature remains constant; consequently, in doubling or trebling the pressure, the volume becomes ½ or ⅓ respectively. According to Charles' law, if the volume of a gas be kept constant, the pressure varies as the absolute temperature; and if the pressure be kept constant, the volume varies as the absolute temperature. By the law of the transmutation of energy, work performed on a body whether solid, liquid, or gaseous, is evidenced by a definite increase of temperature in that body. Consequently, when air is compressed it is heated; when heated, it expands, and the volume of air to be compressed is proportionately increased with a corresponding expenditure of the power required to compress it. Could the temperature of the air undergoing compression be kept constant (isothermal) during the process, and the heat taken up from it be returned to the air during its expansion in the motor while doing work, all loss from this source would be avoided. This, however, is impossible, and the aim of modern compressors is to prevent an increase in the volume of the air by keeping down the temperature during the period of compression; that is, by approximating to what is termed the isothermal process.

Compressors.—These may be primarily divided into two classes, according to the means adopted for reducing the temperature of the air. The "wet" class use a liquid piston in the cylinders; the "dry" class, a water-jacketed cylinder or an internal spray.

Wet compressors are of two types: (*a*) where the water-piston owes its energy to the fall of water from a height; (*b*) where the water-piston is actuated by a steam-driven piston.

At Mont Cenis, Sommeiller, by utilising the momentum of water falling 86 ft., was able to obtain an air pressure of 75 lb. per sq. in. Though extremely low in efficiency (not more than 6 %), and necessitating clumsy plant, the arrangement gave good results. This principle has been applied in other cases, as by Hathorn, Davey and Co., in Mexico; and where an abundant water supply exists, excellent results are obtained.

The second form of wet compresser has attained a wide application on the continent of Europe. As will be shown later on, the dead-space at the end of the air-piston stroke is undesirable, and it is largely to eliminate this defect, and to keep the air cool, that liquid pistons are so much in vogue on the Continent. The water, forced to and fro in the cylinder, and up the pipe at each end, carrying the necessary valves, fills the dead space. But there are a number of inconveniences attendant on the system. The cooling of the air is inefficient, being only on the surface of the water. The speed of the piston is extremely limited, and cannot exceed 40 to 50 ft. per minute, on account of the mass of water to be moved; consequently the number of compressors required for a given work is large. The water agitated by the motion is frothed, and causes excessive moisture in the air. Various devices, more or less successful, have been used to lessen these defects, but the fact remains that, in England and America, these compressors have not found favour.

Dry compressors have a cylinder and piston similar to those of a steam-engine, with suitable outlet and inlet valves at the cylinder ends. The temperature of the air is kept within reasonable limits by the constant flow of cold water through the water jacket of the cylinder from the bottom upwards. It is, however, doubtful if this process of cooling, even under the most favourable conditions, does more than keep the cylinder from becoming excessively heated, and so imparting heat to the incoming air. A more thorough method of cooling is obtained by injecting a fine spray of cold water into the cylinder near the outlet valves. To this objection has been strongly urged, that the presence of water, with its non-lubricating properties, causes an undue wear and tear in the cylinder, and loss in power.

The main difference in the various makes of compressor used in England, South Africa, and the United States, lies in the construction of the air-valves. This is not a matter of primary importance so long as efficiency is attained; there is no marked superiority of any one make. The great desideratum is maximum cooling of the air, and in that direction there is still room for improvement.

Usually the pressure of the air delivered by compressors is between 70 and 80 lb. per sq. in. above that of the atmosphere; but it may vary between 40 and 90 lb., and even the latter figure is sometimes greatly exceeded. There have been instances—as in running a tunnel in record time—where air has been used at 135 lb. and even 150 lb. But in these cases, the runners have been spurred on by a possible bonus. At ordinary day rates, the average drill man will not stand more than 100 lb. of air, and in fact, 75 lb. is rarely exceeded. The speed of compressors is about 65 strokes per minute for the large ones, up to 100 for the small. The "straight-line" compressor is a good machine in moderate sizes, for moderate pressures, under fairly constant loads, where the very highest steam economy is not a vital essential. For mining-plants of a semi-temporary nature, or pending the establishment of a permanent paying basis of operation, it is convenient, reliable and satisfactory. The "duplex" compressor is distinctly the machine for high-grade, permanent plants of large or moderate size and modern high pressure, working under wide variations of load, in conditions which make fuel and water economy imperative. Compressors which are to be electrically driven should have at least 2 cylinders, with their cranks at right angles; 3 cylinders are much to be preferred. Single-crank compressors are quite unsuitable.

Low Efficiency of Compressors.—The causes are: Heating of the air during compression; mechanical defects in the inlet and outlet valves; and leakage past the piston.

It has been already shown that air when subjected to compression is heated, and that as the volume is thereby increased, much power is uselessly expended in dealing with the heated air. The most efficient compressor, therefore, in this regard, must be the one presenting the best cooling arrangement for the air as it is being compressed. That form of compressor in which the piston is represented by the falling water supplying the power, such as Sommeiller's, permits of very thorough cooling, as the water-piston is renewed at each stroke, and the cylinder is kept perfectly cool. But in the second form of wet compressor, such as the Dubois, in which the water, only slightly renewed per stroke, becomes considerably heated, the cooling is not more perfectly effected than in the dry compressor. As the pressure to which the air is raised becomes greater, the losses from this source become serious; and as the efficiency of the motors increases with the pressure, and the size of the conduits can be correspondingly small, it is desirable, particularly in large installations, to use high-pressure air. The most satisfactory results in this direction have been obtained by stage compression, that is, by pressing the air to a certain pressure in one cylinder and further compressing it in a second ; and, if desired, in a third, or even a fourth. By this system the air is cooled at each stage, and the losses from this source are minimised. For low pressures it is doubtful if any practical economy would result from stage compression; but it is now fully demonstrated that for pressures above 60 lb. the advantages of stage compression are very marked.

To diminish losses caused by resistance to passage of air through the inlet outlet valves, many devices have been resorted to. In the ordinary valves held to their work by springs, the valves rattle or chatter if the springs are weak. On the other hand, if the springs are made very strong, a resistance to the passage of the air is set up, resulting in a loss of power which, in some cases, becomes serious. To obviate this defect, the valves are occasionally devised to open mechanically.

Stress is sometimes laid on the losses caused by the unavoidable dead space occupied by compressed air at the end of each stroke. It may be pointed out at once that the loss is not in power, but solely in the volumetric capacity of the compressor. To diminish this inconvenience, the air-piston is usually run as close to the cylinder ends as practicable; but care is requisite to avoid sailing too close to the wind in this direction, and damaging the mechanism. The best plan is to arrange trick passages, or grooves, on the inside of the cylinder, for a short distance back from each end, to allow the air in the dead space to pass the piston to the end in which compression is about to begin. The inside pressure against the suction valves is thereby relieved, and compression on the other side of the piston begins at once. To prevent knocking through the sudden relief caused at the end of the stroke, a certain amount of cushioning in the steam cylinder is required.

Loss of capacity, owing to necessary clearance between cylinder-heads and piston at the end of each stroke, is practically remedied by making the cylinder a little larger; the work performed in compressing the air into the clearance space is to a great extent given out during the succeeding stroke. The usual amount of clearance is about 3% of the cylinder volume; air compressed into this at 75 lb. pressure expands as the piston recedes, and air is taken in until it reaches about 15% of the cylinder volume, so that with single compression to 75 lb. pressure, the capacity of the cylinder is reduced 15% by the clearance.

Low efficiency due to leakage of pistons can be effectually reduced by carefully attending to their condition. Naturally, the higher the compression, the greater the leakage; but stage compression greatly lessens this evil.

Stage Compression.—The value of double-stage compression is shown below from actual trials in 1897.

Pneumatic Stage Compression. (K. Schweder.)

Plants.	Cub. ft. Air Compressed to 65 lb. per minute per I.H.P. at the Engine.	Rev. necessary to make up for the Losses in Compressor and Main Pipe.	% Loss in Main and Compressor when running 80 Rev.	Loss in cub. ft. per minute as Leakage per 1000 sq. ft. Surface of the Pipage.	Cub. ft. Compressed Air delivered to each Drill per minute.	Indicated H.P. at Engine per Drill running.
1. Langlaagte Estate—Rand	5·35	13·2	16·5	27·8[1]	58	13·7
2. Crown Reef—Ingersoll-Sergeant	5·65	37·0	46·25	20·0	77	24·5
3. Meyer & Charlton—Hirnant..	5·25	14·3	17·8	110·0[1]	46	13·5
4. George Goch—Allis	5·85	6·8	8·5	26·0[1]	98	19·5
Average	5·52	17·3	22·3	58·4	70	17·8

[1] Includes slight leakages in compressor.

Nos. 2 and 4 are double-stage compressors, and their superiority in volume of air compressed per i.h.p. is manifested, being a minimum of 5·6%, and a maximum of 11·4%, in the cases quoted. The gain is usually estimated to be 10 to 15%.

Obviously the intake for air to be compressed should be in the coolest place possible, and if the air is filled with dust, some means must be provided for eliminating it. A simple and effective air "washer" consists of a nest of tubes sufficient in number to give ample passage way, set vertically in a box with their lower ends immersed a few inches in water. The air, drawn downward through the tubes and bubbling up through the water, leaves in the latter its solid matter. Needless to say, the water should be changed frequently. If a continuous flow through the box can be secured, so much the better. Besides preventing scouring of the air cylinders by the grit in the air, this device will be further effective as an economiser, if the water is cool enough, by lowering the intake air temperature and increasing the efficiency of compression.

With double compression, owing to the lesser pressure (about 20 lb.) reached in the larger cylinder, the effect of 3% clearance is not much more than to reduce the capacity of the air cylinder by about 5%.

In estimating compressor capacity, altitude must not be lost sight of.

Air Efficiency according to Altitude.

Altitude (ft.)	0	1000	2000	3000	4000	5000
Barometric pressure (in mercury)	30·00	28·88	27·80	26·76	25·76	24·79
,, ,, (lb. per sq. in.)	14·75	14·20	13·67	13·16	12·67	12·20
Volumetric efficiency (%)	100	97	93	90	87	84
Loss of capacity (%)	0	3	7	10	13	16
Increased power required (%) ..	·0	1·8	3·5	5·2	6·9	8·5

6000	7000	8000	9000	10,000	11,000	12,000	13,000	14,000	15,000
23·86	22·97	22·11	21·29	20·49	19·72	18·98	18·27	17·59	16·93
11·73	11·30	10·87	10·46	10·07	9·70	9·34	8·98	8·65	8·32
81	78	76	73	70	68	62	63	60	58
19	22	24	27	30	32	35	37	40	42
10·1	11·6	13·1	14·6	16·1	17·6	19·1	20·6	22·1	23·5

Receivers.—After compression, the air is passed into a receiver of large capacity, so that the intermittent delivery from the compressor is rendered more uniform before entry into the pipes which conduct to the point of operation. Receivers are usually sheet-steel cylinders, 10 to 20 ft. long by 4 to 5 ft. diam., and fitted with safety-valve, pressure-gauge, and blow-off cock. Old boilers of the Cornish or Lancashire type are often utilised in this way, the fire tubes being blocked up with bricks laid in cement. It is rarely that any steps are taken to protect them from the sun's heat in summer, though it would seem to be well worth while. Any watery vapour introduced by the compressor should be condensed in the receiver, and drawn off. Supplementary receivers at various points underground, especially where heavy consumption takes place, and where pipes diverge, are of material assistance in equalising working pressures and supplies.

Pipes.—Two considerations are of importance in air distribution: the size of the pipes, and the character of the joints.

Frictional loss in passage of air through pipes increases very rapidly as the diameter decreases. Thus, if a volume of air at 60 lb. pressure, equivalent to 18,000 cub. ft. per hour at atmospheric pressure, be passed through 1000 ft. of pipes, the loss of pressure of air for $2\frac{1}{2}$- and 4-in.-pipes would be $5\frac{3}{4}$ and $1\frac{1}{2}$ lb. respectively.

For ordinary high pressures lap-welded steel tube is in general use; wrought-iron and butt-welded joints are cheaper, and permissible for low pressures.

Shaft mains range from 3 to 6 in. diam., according to supply needed. The smaller the pipe, the greater the velocity of the air in it, and the higher the friction against the walls of the pipe. With an air velocity of 25 to 30 ft. per second, the friction only amounts to 2 lb. per sq. in. per mile of main. The velocity should not exceed 50 ft. per second.

Along the levels, 2- to 4-in. pipes are used; and generally each machine is connected by a length of hose with a pipe 1 in. diam.

Leakage at joints, through expansion and contraction of pipes, is a fruitful and at times a serious source of loss of power. As shown in table on p. 113, the average loss in four prominent South African mines is 58·4 cub. ft. per minute for every 1000 sq. ft. of pipe surface.

Screw couplings are in many respects the most satisfactory joints. They will withstand the highest pressures, and tolerate much ill usage, besides being easily connected and disconnected, and occupying a minimum of space; the thread should be very coarse. But bolted flange joints, with rubber gasket rings, are sometimes favoured.

Thin sheet asbestos near to, and brown paper at a distance from, the compressor cylinders may be used for making all air-pipe flange-joints, the ordinary rubber packing, besides being more expensive, often proving entirely unsuited, the oil carried along the pipes by the air rapidly decomposing the rubber.

Bends should always be as gradual as possible, and never be made at joints.

A pipe line laid over an irregular surface is full of possibilities of lost power, and should be supported against sagging strains due to its own weight; it should be securely anchored as a precaution against "creeping," and strain of joints; and the consequences of expansion and contraction, should be everywhere provided for. These are indispensable safeguards against leakage, and the volume of good live air which a small leak will discharge is startling; 5% should be the utmost limit of leakage allowed. The power equivalent of a good quantity of coal may be wasted through a leak which can be detected only by the use of soapy water. It is well to have gauge connections on the line for frequent tests for pressure drop. Judgment should also be used in arranging the grade of the line. Pipes should grade toward secondary receivers at the lowest points, and drain cocks should discharge accumulated moisture.

The oil that escapes by the oil collectors near the air cylinders is carried, by the compressed air, along the air pipes, and the greater part is deposited on the inner surface of the pipe line; therefore, although this oil is lost at the collectors, it serves to prevent oxidation of the inner surface of the pipes, and answers the two-fold purpose of enabling a thin tube line to be used, and of preventing rust from being carried along, by the air in the pipes, to the various engine cylinders, which of course would be highly detrimental to the inner surface of the cylinders, valves, etc. It has been found that the oil collected from the compressor cylinders (heavy cylinder oil) may be advantageously used over again three times, each time being thoroughly strained before using. It should not be mixed with new oil, but used by itself, and fed into the cylinder with a sight-feed lubricator. (Short, Trans. Inst. M.M.)

Application.—When air is compressed, it is heated; when it expands, it is cooled. The latter fact gives rise to the inconvenience, so frequently met with in air-motors, of ice being formed in the ports, through the freezing of the moisture in the air. Where the air is admitted to the motor, practically during the whole stroke, there is little danger of ice being formed; but there is an excessive waste of power, for it is as important for economy to use air expansively as it is to use steam expansively.

Trouble may be caused by the drain pipe from the cooling chamber being connected into the main drain pipe in the engine room, which receives water from pumps, condensers and engine drips. The compressor being provided with a governor on the air intake pipe, when the ball on this governor is down, the compressor takes air, and discharges it until the desired pressure is obtained. Then the ball rises, the intake valve closes, and the compressor runs under a high vacuum. At such a time the valve on the drain pipe from the cooling chamber being open, it will draw all the water in the drain pipe into the compressor.

While a little moisture in air used expansively results in the formation of ice in the ports, aqueous vapour has a specific heat nearly double that of air, and consequently cools less rapidly under expansion than dry air, and the tendency of an excess of moisture is to reduce the cooling. The specific heat of water being still greater, a spray of water may be effectively used in the motor cylinder to prevent cooling to the freezing point.

Re-heating the air is, however, the most effective method of allowing air to be used expansively without the formation of ice in the ports, and this can best be done by passing the air near the motor through a coil of pipes heated by a small furnace; and a further elaboration, permitting the highest degree of expansion, is effected by introducing a small quantity of water into the heater, where it is converted into steam. A move in the latter direction was made years ago by the use of a jet of steam in the air pipe near the motor.

In practice it is found that re-heating the air not only prevents freezing, but results in a very great economy in the use of compressed air, at a small cost both for plant and fuel. Thus at the North Star mine, California, the power generated by air at 60° F. was 128 i.h.p.; heated to 300° F., it was 187 i.h.p., or a gain of 46 %; and at 350° F., it was 199 i.h.p., or a gain of 55 %. (P. R. Robert.) And at Glen Lyon, Pennsylvania, re-heating by passage through pipes at the temperature produced by steam at 90 lb. pressure, showed an economy of 50 %. The cost of re-heating to 350° F. at the North Star is 1 cord of wood daily at 18*s.* 9*d.* for 200 i.h.p., or 1⅛*d.* per i.h.p.

Experiments by Prof. Riedler showed that the fuel used in heating compressed air to a temperature of 300° F. was five times as effective as when used for making steam.

The following system is adopted by H. L. Short in a fly-wheel pumping engine, running on compressed air, used expansively in a 10 by 20 in. cylinder, with expansion valve. A super-heated jet of steam is admitted at either end of the cylinder during the entrance of air, through fine perforations directly opposite to the air-admission ports; the steam is generated in a small vertical boiler close to the engine, the steam pressure being a little above the air pressure, and both air and steam can be cut off independently of each other at any part of the stroke. Many tests show that the system is very economical and well worth adopting.

Utility.—The use of compressed air seems at first sight extremely simple, and consequently the principles surrounding it are seldom inquired into. The results obtained from it are in consequence, at times, exceedingly poor, and its reputation as a means of transmitting power suffers proportionately. In the worst forms of machine, 10 % only of the power expended may be obtained; while in the best installations, about 55 % without re-heating, and 75 % with re-heating, should be realised. (Prof. Goodman.)

Practical limit of efficiency, 50 % (Dr. Siemens). Usual efficiency, 25–35 % (Prof. Rankine). General efficiency rarely exceeds 30 % (D. K. Clark). Coal consumption to produce given power is 70 % greater for compressed air than for steam (H. Robinson). Coal consumption of only 1·54 lb. per i.h.p. per hour, at Rio Tinto, in compressors by Walker Bros., Wigan, is highly creditable.

For underground transmission, compressed air is one of the most satisfactory agents, and in particular at the working faces of coal mines—as for actuating coal cutters—it has the special advantage of ventilating and cooling the workings, and of being free from danger of fire. So too in many metalliferous mines, where large quantities of high explosives are used, a jet of compressed air is very beneficial in clearing away the fumes, and enabling men to resume work within a reasonable time after firing.

On the other hand, it is pointed out by Prof. H. Louis, an air-driven coal cutter is very noisy; and in this special case, the noise is more than a trifling objection, as it prevents the men hearing cracks of the roof, which often give the first warning of impending danger. The cost of an air-compressing installation complete "is probably rather less than that of an electric plant for very low powers; but it rapidly loses this advantage as the power increases, and soon becomes the greater of the two for even comparatively moderate amounts of power. Air pipes are more cumbersome than electric cables, require much more time and care in laying, and are far more difficult to keep in good order, especially when the ground is liable to movement. Apart from the question of leakage, which is always greater in the case of compressed air, the loss of power in transmission is far greater with compressed air than with electricity. By using the best type of modern stage compressor, and air pipes of ample size, an efficiency of 50 % may be obtained, but this is an unusually high figure, 30 % being far more usual. Electric installations rarely give an efficiency as low as 65 %, and occasionally go up to 80 %, so that the efficiency of electric transmission may fairly be taken as being twice as good as that by compressed air. As matters stand just now, it may be claimed for compressed air that any ordinary mechanic is quite equal to dealing with the machinery it requires, whilst an electrical plant generally needs a special electrician to look after it. In time, however, this objection will lose its force; and at the same time, part of the additional outlay thus incurred may be recouped by using the electric current as a source of light, as well as of power, an advantage that is presented by no other means of transmission." (Min. Jl.)

A suggestion has been made by A. L. Stevenson, that electric transmission might be used to work air compressors situated underground, and the compressed air might then be carried relatively short distances into the working faces; but in most mines, the air supply for compression would have to be specially conveyed, and not allowed to mix with the mine atmosphere, on account of temperature and impurity. It is being done quite successfully in the new installations on the Comstock, Nevada.

STEAM.

Steam transmission is in no case suitable when the distance between the generator and the engine is great; it is always associated with considerable loss, due to friction in the pipes, to condensation, and to leakage, all of which causes are operative in spite of every care that can be taken in covering the steam pipes with a non-conductor of heat, and in making the joints. Underground the use of steam is objectionable for many reasons; it cannot be used at all at the working face, or in confined situations; the heat given off is a source of trouble; the escape of waste steam is usually inadmissible; whilst condensers not only introduce additional complications, but require that the condenser water shall be got rid of somehow. Steam pipes take up much space in a shaft, besides being a source of danger and discomfort, when steam is taken underground from boilers at the surface; whilst the location of boilers underground gives rise to even more serious objections. It must not be forgotten that the heat and moisture due to the use of steam affect some classes of ground seriously, causing it to heave or to crumble away, whilst the same agencies are most destructive to mine timber. In spite of these drawbacks, occasionally steam is used underground for pumping or hauling engines situated close to the shafts, where its injurious effects are least. Steam is less efficient economically than electricity. (Prof. H. Louis.)

Loss in transmission below ground from surface, even with the best lagging, may easily reach 10 to 20 lb. per sq. in. It varies generally between $\frac{1}{4}$ and $\frac{1}{2}$ lb., and averages about $\frac{1}{3}$ lb. per sq. ft. of pipe per hour with bare pipes in an external temperature of 60° F., and may be reduced by the undermentioned amounts with lagging $1\frac{1}{2}$–2 in. thick of the following substances wrapped in paper and canvas:—

Asbestos paper, 68 %; asbestos paste, 71 %.
Cork in various forms, 83–87 %.
Magnesia carbonate, 85 %; in moulded sections, 83–89%.
Fossil meal and hair plaster, 89 %.
Slag-wool or silicate cotton, 95 %.
Hair-felt and wood, 96 %.

Recessed planking, with the joining ends cut so as to shed water off, reduces condensation to less than ·5 % per 100 ft.

Another good lagging is 2 to 3 in. of magnesia carbonate wrapped in hair-felt, and finally covered with sheet tin, joints being soldered.

All absorptive substances gather and retain moisture, thus increasing condensation unless hermetically enclosed.

There seems to be no reason for the former practice of putting on different thickness of covering on different sized pipes, except the mechanical difficulty of applying a very heavy covering to a small pipe. This difficulty can be overcome by putting the covering on in two separate layers, and this should be done on all sizes, in order that the joints may be broken, as poor joints may reduce the efficiency of the best covering 6 % or more.

Condensation losses may be materially reduced by superheating. At Seghill colliery, 3 boilers supplying superheated steam did the work of 4 with saturated steam, burning the same quality of coal.

A 1400 ft. steam transmission is quoted as the practical limit, pressure drop being 13 lb., and condensation loss about 21 %.

Pipes in Shafts.—A steam pipe in a shaft is a fruitful source of repairs and anxiety, the difficulties arising mainly from expansion and the strains that occur in consequence. Various expansion joints have been devised, relying for their action upon the flexibility of coiled or looped sections, or the movement permitted by "fitted" sections, or the ordinary gland and stuffing-box. Coiled or looped sections, though performing their office admirably if they contain enough length of pipe, have the disadvantages of being expensive, and of taking up valuable room in the shaft. "Fitted" joints or sections are cheap, and usually home-made, but they soon become leaky, and do not admit of ready repairs. The gland and stuffing-box expansion joint is the very worst, needing constant attention to keep it tight. It accomplishes a complete section of the pipe, the portions of the pipe above and below, being discontinuous, are thrust apart upon the admission of the steam by a stress equal to the product of the area of the pipe and the pressure of the steam. This puts the pipe under a buckling strain which it is not fitted to stand. In spite of the pipe being held in line by staples or U-bolts at each timber, it usually yields to the strain, with the result that the parts of the stuffing-box joint separate rather than close up as desired to take up the expansion of the pipe. This style of joint aggravates the trouble it is designed to prevent. The pipe, subjected to frequent recurrence of buckling strains, becomes leaky at every point where there is opportunity.

The following arrangement, employed in the shafts of the St. Louis Smelting and Refining Co., Missouri, overcame the difficulties. (Fig. 24.)

The central ideas of the scheme were to throw all the expansion to one point, the bottom; to keep the pipes at all times under a moderate tension; and to relieve the pipes, when hot, of all stress other than that due to the pressure of the steam.

The pipes were made up in 60-ft. sections, fitted with an extra heavy flange on each end. The flanges were turned, faced, and scraped, so that when bolted together they were steam-tight without the use of gaskets of any kind. The 60-ft. sections were made up complete on the surface, ready to be lowered into place. The pipes were covered with a high-grade magnesia covering, over which was slipped a tight casing of galvanised spirally-riveted pipe. At the flanges, the casing was crimped over the covering, and all spaces in the crimping, or points where the clamps came through the openings, were filled with plaster of paris and painted with tar, to prevent any water getting access to the covering. This means of protecting the covering is much superior to the usual one of wrapping with strips of canvas and painting, and, considering the labour saved, is not much more expensive.

A plumb line is dropped down the shaft at the desired point, the timbers intended to support the pipe are marked accordingly, and the counter-balancing apparatus is placed. The bottom section is next lowered into place, and temporarily supported until the next section is placed, when the lowest counter-balance is attached. The succeeding sections are then lowered,and the counter-balances are attached until the top is reached. The top section is then solidly clamped to the timbers of the gallows frame.

The counter-balances are placed at every 100 ft. Where the

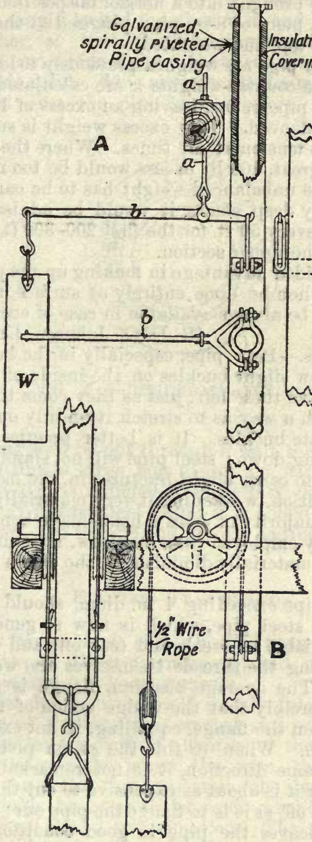


FIG. 24.—STEAM-PIPE EXPANSION.

expansion, or the movement due to expansion, does not exceed 5 in., the style of counter balance shown at A is used; where that amount is exceeded, the kind shown at B is advisable. When the pipe is hot and fully expanded, the rod *a* should be lowered, so that the lever arm *b* is brought into a horizontal position. The weight *w* is made up of a piece of 8-in. pipe plugged at the bottom, fitted with a bail at the top, and weighted.

The sections of pipe are weighed previously to lowering to place in the shaft. The counter-weights *w* are calculated to balance the 100 ft. of covered pipe below, leaving an excess of 100 lb. in favour of the pipe unbalanced. This excess weight is sufficient to keep the pipe in slight tension at all times. Where the total length of the pipe is very great, 100 lb. excess would be too much to be left unbalanced, as the unbalanced weight has to be carried by the top sections. In very deep shafts, it would be advisable to use the counter-balances every 50 ft. for the first 200–300 ft., to prevent too great tension on the upper section.

There is a decided advantage in making up the pipe in sections, as the work can then be done entirely at surface, and a complete spare section can be always available in case of accident.

(R. D. O. Johnson, En. & Min. Jl.)

Bending Pipes.—Bent pipe, especially in the larger sizes, will nearly always show slight buckles on the inside of the bend; but it is better to have these left just as they come than to have the pipe bent in such a way as to stretch it unduly on the outside of the bend to obviate buckles. It is better practice to leave them without hammering down: steel pipe will not stand this usage very well; it is liable to cause slight fractures in the metal, or produce a laminated condition, weakening it very materially.

In the great majority of cases a home-made bend can be much more satisfactorily employed than an elbow, lessening the length of of pipe used, and materially diminishing the strain and loss due to sharp turns.

Joints.—No pipe exceeding 4 in. diam. should have a screwed flange joint. On steel pipe, which is now so generally used, the threads are very liable to break and tear off, and give trouble in many ways, making the threads themselves the weakest point in the entire line. The average cast-iron flange is screwed on the taper thread so forcibly that the wedge action of the thread will produce a strain on the flange, equalling, if not exceeding, its safe limit of strength. When to this the steam pressure is added, pulling in the same direction, it is not remarkable that flanges break. Moreover it is about as expensive to cut threads, put on a flange, and face it off, as it is to flange the pipe out; and, if properly done, the latter leaves the pipe in good condition, and makes a joint that will stand more pressure than the body of the pipe itself. The flanged-out portion of the pipe, reaching out to the bolt holes, gives just as much surface to carry packing as any flange does.

With the flanged-out joint, it is best to use a divided flange (Fig. 25), especially where it is difficult to get pipe into position with flanges on, as this flange can be fixed after the pipe is in place. It is simply locked together at *a*, and the two flanges are bolted together, so that the dividing lines cross each other, thus making companion flanges that are just as strong as though they were made solid. This obviates all the trouble with ordinary flanges placed over the pipe loosely before being flanged out.

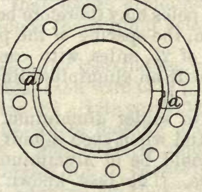


FIG. 25.—DIVIDED FLANGE JOINT.

Again, if piping is to be put through a wall, the divided flange does away with cutting a large hole to get the flanges through. The divided flange also is indispensable in very close quarters. It is made preferably in cast-steel; after it is bolted together, all strain is removed from the tongue and groove, as the bolts hold them in position.

ELECTRIC TRANSMISSION.

In electric transmission, choice lies between high and low tensions, and between direct (continuous) and alternating (single-phase, di-phase, and tri-phase) currents.

Cables.—Tension resolves itself mainly into cost of cable. A certain portion of the tension is wasted in overcoming the resistance of the line, and a certain quantity of the current is lost by defective insulation; the larger the wire, the less pressure is wasted; and the better the insulation, the less loss by leakage. There are reliable bases on which each can be estimated beforehand with very close approximation. The principal material available for cables is copper. Overhead wires are usual, and for these the copper is best "hard-drawn," in order to increase its tensile strength, and to enable it to resist the strain caused both by its own weight and by external causes, such as accidental blows and other interference, underground, and storms above ground; but the strain of its own weight may be relieved by suspending it with leather thongs from an iron supporting wire, and this practice is certainly to be commended.

Electric transmission to a long distance would certainly not be possible in the immediate future by the employment of underground cables, because the cost of an underground cable is between 4 times and 10 times the cost of a bare copper wire, and the cost of the copper in a long transmission line is a serious item of expense.

Aluminium is becoming a serious rival to copper, because, for equal conductivity, it is only about half the weight of copper, and the span between the poles may therefore be greater. On a line of 142 miles transmission, the spacing of the poles is 132 ft. with aluminium; on another of 25 miles, with copper, the spacing of the poles is 75 ft., whilst with an aluminium line which has been added later, the spacing is 112 ft.

As it is difficult to solder aluminium, a mechanical joint is generally employed, and when such a joint is made, it is necessary that all the parts should be of aluminium, otherwise electrolytic action may take place. The best known joint is the McIntyre, consisting of an aluminium tube 9 in. long and $\frac{1}{16}$ in. thick, large enough to snugly enclose two wires. The whole is given several complete twists by clamping rods, and the joint is finished by turning back the ends of the wires.

A very recent American transmission line of 154 miles consists of three $\frac{5}{8}$-in. 37-strand aluminium cables supported on special high-tension insulators for a potential of 60,000 volts. The most notable feature of the transmission is a river crossing, where the aluminium cables are supported on four rectangular steel towers 150 ft. high and resting on piles. The longest span is 618 ft.

On the Kolar transmission line, where crossing rivers, railways, and telegraph wires, a cable of silicon bronze is substituted for the ordinary copper wire. Transposition of the main line wires is made three times in the total length of 92 miles, in order to obviate induction. Each line of three wires is of sufficient capacity to transmit the full power at increased potential, to meet emergencies in the way of breakdown of lines, or repairs; but ordinarily both lines will be used for transmission, each carrying half the current.

For underground work, bitumen or rubber cables are very useful, as lead sheaving is heavy, and easily damaged when hanging down the shafts, or in the roads underground, and a fault upon a lead-sheathed cable would shut down a mine without warning, where a similar fault on a leadless cable would give a warning reading upon the switchboard instruments long before it took such a drastic step. But bitumen cables are mechanically weak, the conductor decentralises when overheated, and certain alkalies attack the bitumen. Still bitumen cables are the most suitable for low-tension single cables laid in troughs; and for multiple conductor cables, as the three-phase type, paper-insulated, bitumen-sheathed cables are best. For extra high-tension cables, it is necessary to have a lead sheath, and, in addition, an armouring of (preferably) galvanised iron wires over the lead, to ensure ample conductivity for leakage currents.

A very frequently omitted precaution with lead-covered cables is the covering with a padding of jute. This jute serving is cheap, and is a most useful protection in a number of ways, and a sound method of provision against trouble when laying the cable and afterwards.

It should be noted that where armoured cable runs through ground composed of cinders, the sulphur acts upon such armouring very disastrously.

In the new Comstock mine installation, current is taken down the shaft in a 3-wire cable, each wire of 500,000 circular mils area. These wires are protected by a jute covering, a lead covering, and a steel wire armour, the whole making a cable of $2\frac{3}{4}$ in. diam., weighing 10 lb. per ft. This cable is supported by brackets at every 30 ft., and, at intervals of 6 ft. between, is secured by staples to the shaft timbers. Underground, the 2200-volt current is transformed to 440 volts for motors under 50 h.p., and to 110 volts for the lighting circuit.

Personal contact with any cable carrying more than 500 volts may be fatal to human life.

According to Lord Kelvin's law, in transmitting any given current to any given distance, the most economical cross section of cable is such that the electric energy annually dissipated in it shall equal in value the annual interest on the first cost, plus allowance for depreciation. The table on p. 126 (dimensions and resistances of copper cables) relates to wire calculated to carry 1000 amperes per sq. in. of sectional area, which is the maximum amount admissible for ordinary cables covered with insulating material, as a greater current would heat the wires to such an extent as to endanger insulation. Cables for long-distance transmission are best, however, arranged as bare overhead wires, supported on insulators, and for such wires 2000 or even 3000 amperes per sq. in. of sectional area is not too much; there is no insulating coat to destroy, and heat is conducted away far more readily from a bare than from a covered wire. This is an important matter, because any notable increase in temperature of conducting wire greatly increases the resistance which it offers to passage of current. Only exceptionally is a heavier wire than No. 5 covered with insulating material. When a covered conductor of greater sectional area is needed, the requisite number of wires are twisted together into a cable. By this means, greater flexibility is obtained than would be the case with a single wire, a cable of given cross section being more flexible the larger the number of wires of which it is made up; it is, however, at the same time more expensive, and its tensile strength is somewhat less. The weight and resistance of such a compound cable are practically equal to those of a wire of equal sectional area. Thus a No. 14 wire can be replaced by a cable made of 3 No. 18 wires, or of 7 No. 21 wires; these would be technically designated as 3–18 and 7–21 cables respectively.

Dimensions and Resistances of Copper Cables. (Louis.)

Standard wire gauge.	Diam.	Area.	Weight.		Resistance at 60° F.	
			per 1000 yd.	per mile.	per 1000 yd.	per mile.
No.	in.	sq. in.	lb.	lb.	ohm.	ohm.
0	0·324	0·0824	958·1	1685·8	0·288	0·507
1	0·300	0·0706	821·0	1444·3	0·336	0·591
2	0·276	0·0598	695·3	1223·6	0·397	0·698
3	0·252	0·0499	580·3	1024·0	0·476	0·837
4	0·232	0·0423	481·8	865·3	0·561	0·987
5	0·212	0·0353	410·5	722·3	0·672	1·18
6	0·192	0·0290	336·0	591·2	0·828	1·44
7	0·176	0·0243	282·3	496·1	0·977	1·71
8	0·160	0·0201	232·5	409·2	1·18	2·07
9	0·144	0·0163	188·3	331·5	1·45	2·56
10	0·128	0·0129	148·8	261·9	1·84	3·24
11	0·116	0·0106	122·2	215·1	2·24	3·94
12	0·104	0·0085	98·22	172·9	2·78	4·89
13	0·092	0·0066	76·86	135·3	3·57	6·28
14	0·080	0·0050	58·12	102·3	4·71	8·29
15	0·072	0·0041	47·08	82·85	5·81	10·23
16	0·064	0·0032	37·19	65·47	7·36	12·97
17	0·056	0·0025	28·48	50·12	9·61	16·91
18	0·048	0·0018	20·92	36·82	13·10	23·06
19	0·040	0·0013	14·53	25·57	18·86	33·19
20	0·036	0·0010	11·77	20·71	23·26	40·94
21	0·032	0·0008	9·29	16·37	29·45	51·83
22	0·028	0·0006	7·12	12·53	38·46	67·69

Usually insulators are of stout glass or porcelain (often old bottle-necks are utilised), but in very wet climates an oil-bath is sometimes preferred. They are supported on iron or wooden poles at a height ensuring freedom from contact with ground or snow level, and out of reach of falling trees; the supports are best as far apart as may be, each being a point for leakage of current.

Wooden poles are used almost exclusively for over-head transmission lines. The kind of wood depends on the country or district; in Europe, fir, spruce, and pine are commonly employed; in the States, harder woods, such as chestnut, cedar, and sawed redwood, are also used; in Australia and India, jarrah; in Malaya, chengai and mirbau. In favourable soils, spruce and pine will last five years, chestnut 12, and cedar 20, and the harder woods even longer, the butts of the poles being, of course, treated with some preservative, such as hot tar, pitch, asphalt, carbolineum, salt, or Jodelite. The last-named is particularly valuable for transmission lines in tropical countries, where the wood is liable to be attacked by white ants or other insects. As a further protection against atmospheric influences, some of the American pole lines are painted throughout.

Whereas cross arms are usually made of oak in England, they are generally of yellow pine, treated with asphaltum or linseed oil, in the States. A notch is cut in the pole to receive the cross arm, and it is held by one $\frac{3}{4}$ or $\frac{7}{8}$ in. bolt. If longer than 5 ft., they should be secured by braces. On the Butte line (50,000 volts), each brace is 3 ft. long, 3 in. wide, of maple wood fastened with wooden pegs.

In extra high-tension work, experience has shown that wood is the only suitable material for insulator pins. Oak, locust wood, pine, or eucalyptus is employed, after drying, being boiled for several hours in linseed oil or paraffin. On the Butte line the pins are of seasoned oak boiled in paraffin, and they measure $17\frac{1}{2}$ in. long by $2\frac{1}{2}$ in. diam. in the middle. The insulators are supported high enough to give 9 in. between the lower outside edges of the insulator and the top of the cross arm.

On the Kolar line, each jarrah post, 29 ft. long and 7 in. square, is put into a steel pipe, 13 ft. long, of which 7 ft. is imbedded in the ground. Squaring the posts is an unnecessary expense.

An ordinary cost for posts fixed and equipped is about 2*l.* each or 120*l.* a mile.

It is now the most common and probably the best practice on long transmissions at high voltages to mount each 3-phase circuit on a separate pole line, where wooden poles are used. For a 50,000 volt, 3-phase circuit, the conductors should be mounted on insulators at least 6 ft. apart, and the length of each pole above ground should be not less than 35 ft. The standard distance between the poles may properly be 100 ft., and a fair allowance for shorter spacing on curves and corners brings their number per mile up to about 60.

Spans of over 4000 ft. are successfully negotiated.

Some of the largest installations in operation have the following capacities:—

Miles.	Voltage.	Capacity (h.p.).
47	55,000	40,200
55	50,000	12,000
65	50,000	..
67	60,000	26,800
90	60,000	118,000
99	60,000	..
100	60,000	6,040
101	60,000	8,040
110	67,500	15,100
142	40,000	..
154	60,000	..
173	60,000	40,200

As a cable of given size will carry a current with definite loss of voltage, independent of the tension of the current, the percentage of current lost is less when the voltage is high than when it is low. And since the minimum cross section of cable is determined by the number of amperes it transmits, independently of the voltage, whilst the power that a current can generate varies as the product of the volts multiplied by amperes, a smaller wire can be used to transmit a given amount of power in proportion as the voltage of the current is higher. Tensions of 10,000 volts are now quite within the limits of regular work. Any form of current can be generated at low voltage, transformed to high voltage, transmitted in this form, and then transformed down again to any voltage that may suit the purposes to which the current is to be applied; and in the best modern practice, the amount of electrical energy wasted in such transformation is very small. The most suitable tension at which current can be transmitted must be worked out specially for each individual case. For a short distance, it may be cheaper to put in heavier wires; as the distance increases, a point is soon reached at which the cost of transformers falls below that of difference between wires. The question is entirely a commercial one. The cost of electrical energy in the first case determines how large a proportion it may be economical to sacrifice in order to lessen first cost of line.

Lightning protectors or arresters are indispensable to any outdoor installation. Quite the most reliable is the ingenious water-jet arrester, which is used at the Vizzola Power Station, N. Italy. In this apparatus, the minute spaces between the globules of water in the jet effectively prevent the station voltage going to earth, but will readily pass a lightning discharge. The apparatus has the advantage that it is always ready, and there is nothing to burn up.

Direct-Current System.—This is the simplest of the various systems of electric transmission; it gives a high efficiency, and is free from self-induction; the motors are self-starting, will start when fully loaded, and admit of being considerably overloaded; the current can readily be sub-divided as required, and is available for lighting and other uses. On the other hand, direct-current dynamos are not suitable for generation of high-voltage current. The commutator is an integral portion of a direct-current generator, and, as this consists of alternate strips of metal and of insulating material, as soon as the voltage is at all high, the difficulty of maintaining insulation between the conducting strips becomes so great as to be practically insuperable; on this account, 2000 volts is looked upon as about the practical limit for direct-current dynamos. To generate direct current at low voltage and transform to high voltage involves the use of rotary transformers, with additional complications and expense. As it is only in the case of comparatively long-distance transmission of considerable power that the superiority of high voltage is marked, it may be said that continuous-current transmission is best adapted for powers under 500 h.p., and for distances under 1 mile.

Examples.—(*a*) Dalmatia mine, California. Water power from 8-ft. Pelton wheel under 100 ft. head, running at 100 rev., and belted to 100-h.p. Brush dynamo at 900 rev., generating 30 amperes. Single insulated cable ·2294 in. diam. and 2 miles long. Motor runs at 950 rev., and affords 70 h.p. Efficiency, about 64%.

(*b*) Comstock mine, Nevada. Water power down 1630-ft. shaft to 6 Pelton wheels, 40 in. diam., through ⅜-in. nozzles. Each Pelton wheel drives direct at 900 rev. a 125-h.p. dynamo wound for 40 amperes. Stranded cable, ⅝ in. diam., conveys current up shaft to six 90-h.p. motors. Efficiency reported "very high."

Alternating-Current System.—This is adapted to very high tensions, as much as 130,000 volts having been successfully tried.

The single-phase variety is popular, but the motors require to be brought up to about their normal speed of running by external means before the electric current which maintains that speed is switched on to them; they can only be used as "synchronous" motors, which means that the speeds of generator and motor must be identical. The motor may be heavily overloaded.

Di-phase and tri-phase systems can be worked either synchronously or not; they are self-starting under light loads, but only exceptionally under full load. The tri-phase system admits of greatest economy in line construction, requiring only ¾ of the weight of wire to transmit the same power. A comparatively small overload will at once pull up a tri-phase motor. It only gives maximum efficiency at one particular rate of working, falling off when the speed rises above or falls below this rate. The di-phase system may be used with either a 4- or 3-wire circuit, and thus possesses a wide range of adaptability; but it is not favoured, the tri-phase being preferred.

Examples.—(*a*) Gold King mine, Colorado. Single-phase. Water power from 6-ft. Pelton wheel under 320-ft. head. Dynamo 100 h.p., working at 833 rev. and 3000 volts; 12-pole machine, giving 10,000 alternations per minute. Bare cable, 3 miles long. Plant now extended to 1000 h.p., working various motors of 50 to 250 h.p., at distances of 3 to 10 miles.

(*b*) Standard mine, California. Single-phase. Water power from four 21-in. Pelton wheels under 355-ft. head, running at 860 to 870 rev., and developing 60 h.p. each max. They are keyed on one driving shaft, coupled to armature shaft of generator, which is a 120-kilowatt 12-pole Westinghouse machine capable of generating 3530 volts, but usually working at 3390. Cable of bare soft-drawn copper, ·2893 in. diam., on double glass insulators attached to poles 100 ft. apart; 12½ miles long. Current averages 3400 volts, and 20 to 25 amperes, and the line loss is between 8 and 9%. Motor is a 120-h.p. synchronous machine, which is started by a 10-h.p. alternate-current Tesla motor, the operation occupying 4 to 7 minutes. Efficiency of entire plant 77 to 80%. Cost, 7700*l.* Saving effected, 400*l.* per month over wood fuel at 40*s.* per cord.

(*c*) Sheba mine, Transvaal. Di-phase. Water power from 30-in. and 25-in. Victor turbines under 32-ft. head, running at 232 and 279 rev. respectively, and developing 396 b.h.p. They rope-drive a countershaft at 300 rev., which belt-drives two 150-h.p. alternators at 400 rev. and 3300 volts. Pair of concentric cables, each having inner strand of 19 wires ·057 in. diam., and similar outer strand, whole lead-sheathed; laid in trench 3 ft. deep, and 5 miles long. The high-tension current is transformed at ratio of 30 to 1 in four single-phase 50-kilowatt machines, and is supplied at 100 volts to two 50-h.p. and 3 15-h.p. motors, running at 640 to 800 rev. The latter are of induction type, and are started by resistances under practically full load. Efficiency of entire system, 70%. Cost, including reserve generator, cable, and transformer, about 35,000*l.* Saving effected, some 10,000*l.* a year.

(*d*) Gold Estates, Transvaal. Tri-phase. Water power from Girard turbines. Tension of current, 3000 volts, transformed to 120 to 220 for various motors, largest being 100 h.p.

(*e*) Real del Monte, Mexico. Tri-phase. Water power from 5 Pelton wheels under 800-ft. head, each working a 300-kilowatt generator, supplying current at 700 volts. Current is transformed to 10,000 volts, and transmitted an average of 20 miles; then re-transformed to work motors of 20 h.p. to 150 h.p.

(*f*) Ogden, Utah. Tri-phase. Current at 36,000 volts, transmitted 66 miles.

(*g*) Fresno, California. Tri-phase. Current at 10,000 volts, transmitting 360 kilowatts 35 miles.

(*h*) Loddon, Victoria. Tri-phase. Steam-power; current at

6600 volts, transmitting over 3000 h.p.; distances range from $1\frac{1}{2}$ to 5 miles. Use three 400-kilowatt machines, and two 30-kilowatt exciters; former are of revolving field type, with 48 poles, and run at 150 rev. per minute. Current transformed to 440 volts; motors range up to 400 h.p., and operate pumps, hoists, puddlers, and stamps.

The principal advantage of the alternating-current system lies in the possibility of employing high voltages, since the cost of transmission copper varies inversely as the square of the voltage. The voltage of the direct-current system is practically limited to about 750, on account of commutation. The voltage of the alternating-current system, however, is limited only by the insulation of the transmission line. In the open air, voltages as high as 50,000 are in successful every-day use, transmitting power for commercial purposes through distances as great as 150 miles. In mining work, however, involving underground transmission lines, the limit may be placed at approximately 10,000 volts.

In an installation at Windber, Pa., a cable containing 3 No. 6 B. and S. wires transmits 350 h.p. with a line loss of barely 5%, and delivers direct current to the trolley circuit at 275 volts. On the other hand, if the direct-current system were used, with a voltage as high as 300 at the power house, and 250 at the centre of distribution, that is, 16·6% line loss, the transmission line would consist of 18 No. 0000 feeders on one side of the circuit and 6 No. 0000 feeders in addition to the bonded double track on the other side. With the alternating-current system, the copper for 5% loss weighs 2200 lb.; while with direct-current system, the copper for 16·6% loss would weigh 138, 500 lb., and for 5 % loss approximately 560,000 lb. (W. B. Clarke, Am. I.M.E.)

According to Californian experience (1896), the average first cost of plant for transmission of at least 1000 h.p. 13 to 25 miles was 20*l*. to 30*l*. per h.p.; and the working cost, including maintenance, 2*l*. 15*s*. to 6*l*. per h.p. per annum, using water power.

Transmission Plant for 5 h.p. (Kapp.)

Distance of Transmission in miles.	Annual Cost per H.P. delivered, if the Transmission is—		
	By Batteries.	Direct.	
		Overhead.	Underground.
	£	£	£
1	36·1	22·8	33·6
2	37·6	25·6	47·2
3	39·1	28·0	60·0
4	40·6	30·6	74·0
5	42·1	33·0	87·0

Formulæ for Electric Transmission of Power (Kapp.)

$$\text{Volts} = \text{H}\, v\, l\, 10^{-8}$$

$$\text{Kilogrammes} = \frac{\text{H}\, c\, l}{9{,}810{,}000}$$

C Total current through armature.
c Current through single armature conductor.
e_a E.M.F. in armature in volts.
τ Number of active conductors counted all round armature.
p Number of pairs of poles ($p = 1$ in a two-pole machine).
n Speed in revolutions per minute.
F Total induction in C.G.S. lines.
Z " English lines.

Electromotive force

$$\left.\begin{aligned} e_a &= \text{F}\,\tau\,\frac{n}{60}\,10^{-8} \\ e_a &= \text{Z}\,\tau\,n\,10^{-6} \end{aligned}\right\}\text{ for two pole machines.}$$

$$\left.\begin{aligned} e_a &= p\,\text{F}\,\tau\,\frac{n}{60}\,10^{-8} \\ e_a &= p\,\text{Z}\,\tau\,n\,10^{-6} \end{aligned}\right\}\text{ for multipolar machines with series wound armature.}$$

Torque

$$\left.\begin{aligned} \text{Kilogramme-metres} &= 1{\cdot}615\ \text{F}\,\tau\,\text{C}\,10^{-10} \\ \text{Foot-pounds} &= 7{\cdot}05\ \text{Z}\,\tau\,\text{C}\,10^{-6} \end{aligned}\right\}\text{ for two-pole machines.}$$

$$\left.\begin{aligned} \text{Kilogramme-metres} &= 3{\cdot}23\ \text{F}\,\tau\,c\,p\,10^{-10} \\ \text{Foot-pounds} &= 14{\cdot}10\ \text{Z}\,\tau\,c\,p\,10^{-6} \end{aligned}\right\}\text{ for multipolar machines.}$$

Most Economical Current for Electric Power Transmission. (Kapp.)

D	Distance in miles.
a	Section of conductor in sq. in.
E	Terminal volts at generator.
e	Terminal volts at motor.
HP_g	Brake horse-power required to drive generator.
HP_m	Brake horse-power obtained from motor.
c	Current in amperes.
	Efficiency of generator, 90 per cent.; efficiency of motor, 90 per cent.
.. .. ..	Cost in £ per electrical horse-power output of generator.
m	Cost in £ per brake horse-power output of motor, including regulating gear.
$G = \cdot 9g\ HP_g$ ..	Cost in £ of generator.
$M = m\ HP_m$ ..	Cost in £ of motor and regulating gear.
$t = 18 \cdot 2\ Da$ ..	Weight in tons of copper in line.
K	Cost in £ per ton of copper, including labour in erection.
s	Cost in £ of supports of line per mile run.
p	Cost in £ of one annual brake horse-power absorbed by generator.
q	Percentage for interest and depreciation on the whole plant.

$$\text{Capital outlay} = g\frac{Ec}{746} + m\ HP_m + Ds + \frac{1 \cdot 6\ K\ D^2\ c^2}{Ec - 830\ HP_m} = A.$$

$$\text{Annual cost per brake horse-power delivered} = q\frac{A}{HP_m} + p\frac{HP_g}{HP_m}.$$

$$\text{Put } B = \frac{Ep}{670} + q\frac{Eg}{746}.$$

$$j = \frac{830}{E} HP_m,$$ the current which would be required if the line had no resistance,

and $\beta = j^2 \dfrac{E\ B}{1 \cdot 6\ q\ K\ D^2 + E\ B}$; then the most economical current at the given voltage E is—

$$c = j\left\{1 + \sqrt{1 - \frac{\beta}{j^2}}\right\}$$

$$c = j\left\{1 + \sqrt{\frac{1 \cdot 6\ q\ K\ D^2}{1 \cdot 6\ q\ K\ D^2 + B\ E}}\right\}$$

For very long distances the term under the square root approaches unity, and the most economical current the value $2j$; from which it follows that under no circumstances will it be economical to lose more than half the total power in the line.

Cost of Transmission of Power Plant. (Kapp.)

Distance in Miles.	H.P. Delivered.	Speed of Machines.	Cost in £. Gen.	Cost in £. Mot.	Cost in £. Line.	Total Cost.*	Cost per H.P.
						£	£
1·870	85	450	640	560	440	1,880	22·2
·280	195	500	760	680	132	1,800	9·7
·280	51	600	320	280	60	720	14·1
·375	90	550	520	480	80	1,240	13·8
·560	71	600	440	400	60	1,040	14·6
·280	40	700	260	240	20	640	16
·375	75	600	480	440	68	1,120	15
·500	87	500	520	480	100	1,260	14·5
1·560	150	600	760	720	330	2,050	13·7
·220	93	450	440	420	232	1,270	13·7
6·250	11	900	132	110	480	960	87
2·200	51	600	360	320	300	1,140	22·4
·187	60	900	240	220	18	600	10
5·000	41	750	240	200	344	1,020	24·8
3·750	220	600	1,040	960	640	2,960	13·5
·002	15	600	112	104	8	252	16·8
·250	19	700	160	160	20	390	20·5

* This includes regulating apparatus, instruments, posts, insulators, lightning arresters, erection, and supervision.

Schaffhausen Electric Power Transmission Plant. (Kapp.)

	Generators.	Twin Motor.	Small Motors.
Number of machines	2	1	2
Normal horse-power	300	380	60
Number of poles in magnet fields	6	6	2
Revolutions per minute	300	300	350
Terminal voltage	624	600	600
Normal current, amperes	330	500	81
Diameter of armature, inches	47½	42½	23⅝
Length of armature core, inches	20	20⅜	22½
Radial depth of armature core	8	7	4¾
Section of armature conductor, square inches	·103	·078	·0287
Number of armature conductors	316	316	540
Number of commutator segments	158	158	90
Loss in armature resistance per cent.	1·46	1·52	2·7
Induction in armature, C.G.S. measure	7,500	7,600	15,800
Shunt resistance, ohms	140	143	295
Loss in shunt excitation per cent.	1·35	1·68	—
Main turns per magnet	6	4	—
Loss in main excitation per cent.	·3	·2	—
Type of armature	Drum	Drum	Cylinder

The power is transmitted from a turbine station across the Rhine to the Schaffhausen spinning mills, a distance of about 750 yd. At the generating station are four turbines, each of 350 h.p., of which only two are at present in use, and the energy delivered at the turbine pulleys is sold to the mill owners at the rate of 2*l.* 16*s.* per h.p. per annum. From the turbine pulleys, two 6-pole dynamos are driven by cotton ropes, each dynamo having an output of 330 amperes at 624 volts when running at 300 rev. per minute. These machines are over-compounded, so as to give a constant potential of 600 volts at the motor-end of the line, and, in ordinary working, are coupled in parallel. At the receiving station is a twin motor of 380 h.p., and two other motors each of 60 h.p. The twin motor is 6-pole, and the smaller motors 2-pole, and the power is transmitted from them to the mill shafting by cotton ropes. There are four main conductors, all overhead, each having a sectional area of ·437 sq. in., and being supported by iron towers 46 ft. high; the span across the river is 330 ft., and each of the remaining spans is 430 ft. The guaranteed commercial efficiency reckoned from the turbine pulleys is 78 per cent., the variation of speed of the motors between full and no load not more than 3 per cent., the life of a set of brushes not less than 2000 hours, and of a commutator not less than 20,000 hours. The total cost of the electrical part of the plant, including iron towers and erection, was 6800*l.*

The use of alternate-current generators and motors for large transmissions has much in its favour; there are no commutators, and, by having transformers at both ends of the line the working potential may be 10,000 or 25,000 volts, or even higher still, while the potential at the machines is only a few hundred volts. An ordinary alternate-current dynamo can easily be run as a motor, but, as such, it is not self-starting, and, if overloaded to the extent of 50–100 per cent., may get out of step with the supply current and come to a standstill. There is another method of alternate-current transmission, known as the "three-phase current" system, which has neither of the above objections; it is a development of the Ferraris two-wire and of the Tesla three-wire systems, but with it the transmission and plant efficiencies are much greater than with the Tesla system. An objection to the Ferraris system is that the speed of the motor is not self-regulating, but may vary from zero to the synchronising speed, according to the load.

Electric Accumulators in Mines.

The use of accumulators in mines is not far off, and it will be of interest to many to see how far this reservoir of power will bear filling and drawing upon at the present time, and what the relative cost of the two electrical systems may be expected to be, not so much in first outlay as in the running expenses of the plant.

The first thing to be decided is the weight of these accumulators, and the easiest way to define this, is the weight per h.p. Salom says it takes 25 lb. of battery to give 1 h.p.-hour, and that to give 100 h.p.-hours, or 10 h.p. for 10 hours, requires 5500 lb. of battery, or 220 elements; but 25 lb. is the net weight and 32 lb. the real total upon which we must base our calculations, so that we have a total of 7040 lb. as the weight of this battery. Then as to the room this will take upon a mine-locomotive. A mine-locomotive of 10 h.p. should not be more than 9 ft. long, and 2 ft. of this will be taken up by the bumpers, leaving 7 ft. in length for the battery. Then for a 3-ft. track it should not be more than 5 ft. wide. Allowing 1 in. all round a cell, it will be possible to set 96 of these on the floor-space of the locomotive; but we must have 14 more than this, which will make the width 68 in. The height of this cell is 8 in.; and allowing 2 in. space above it, and 1½ in. of plank, the top of the second tier will be 19½ in. high above the floor.

If the motor is to be placed below this floor (and there is no other place for it), then the bottom of the floor will be 2·6 in. from the track; and, allowing the floor to be 3 in. thick to stand the weight, the top of this car will be 4 ft. 4½ in. above the track. Remembering that the height of the 40-h.p. motor at Lykens Valley is only 4 ft., and that the one at Erie Colliery, also 40-h.p., is 4 ft. 4 in. high, and that they are both narrower and of the same length as the proposed storage-motor, and of four times the power, there is certainly one point established against the present use of accumulators.

Weight, etc., of Electric Mine Locomotives. (Pocock.)

Location.	Horse-power of Motor.	Weight of Locomotive. lb.	Largest Load. Tons.	Speed. Miles per hour.
Zankerode	4·5	3000	13½	6
Paulus	5 to 6	4200	..	6
Lykens	40	12,000	165	6·8
,,	40	12,000	150	6·8
Shawnee	..	4500	21	5
Buckingham	..	7000	60	8
,,	..	4000	30	8
Bear Run	60	18,000	150	6
Erie	40	13,500	107	6

To the weight of accumulators, 7040 lb., must be added that of motor, say 1300 lb., wheels and axles, say 1100 lb., and frame of machine, say, for strength alone, 1400 lb., giving a total weight of 10,840 lb.

From this it appears that the small German motors weigh about 700 lb. per h.p., and that the large American motors only weigh

300 lb. per h.p., whereas the accumulator-motor would weigh at least 1000 lb. per h.p. It is true that weight is necessary to traction, but it is also true that unnecessary weight will entail loss of power, and it appears that 700 lb. per h.p. is found to work satisfactorily with small motors, and that the weight per h.p. decreases as the h.p. increases. Consequently for a 10-h.p. motor, 1000 lb. per h.p. appears to be too high. There will be a waste of power in moving this weight, and, if we wished to operate a 40-h.p. locomotive with batteries, the number of cells would be 880, and their weight 28,160 lb. They might be divided into two batteries, and towed in a tender, but even then each would weigh 14,080 lb., and this would absorb at least 500 lb. of the total pull of the motor, when running on the level, and considerably more where grades were to be overcome.

Dr. Lewis Bell has pointed out that, while a good roadbed is necessary for electrical traction by overhead wire, it is even more imperatively so when storage-batteries are used. We have found in practice that a 25 lb.-per-yd. steel rail is too light for a locomotive of 13,000 lb. weight to be run upon; therefore, there is no saving to be looked for in this direction by the use of accumulators. On the contrary, when we begin to put in a roadbed heavy enough to stand the weight of a locomotive weighing, say, 23,000 lb., there are many disadvantages that the colliery-manager will be the first to see. Besides the first cost of the track, the keeping of this weight of track in good repair in a mine where the floor is for ever moving (as it is in many of our mines), would be a work of no slight expense in itself. It appears, therefore, that not much is to be expected from accumulators as a means of haulage.

However, this is one side of the question only. There is another side which should be considered, and that is the use of the accumulator-motor in collecting the cars to a point where the heavy haulage-motor can reach them. The prospect, as viewed from this point, is more encouraging. From what I have seen of the work, a motor of about 5 h.p. would do the work of 3-4 mules, and the weight would be about 5000 lb., as follows:—

	lb.
Accumulators (110 cells)	3500
Motor	600
Wheels and frame	1000
Total weight	5100

This machine could be built low, so as to take up little height. It is only possible to make this form a commercial success, in my opinion, where the gangways are low, and the roofs would have to be cut to gain height enough for mules to work. Then, the cost of the cutting saved by the use of the motor would counterbalance the repairs on the battery, and the extra care and expense in laying and keeping the track in order. The cost of the system may be estimated as follows—assuming the first cost of the locomotive complete to be 460*l.*, and allowing that the mine can afford to charge these cells for 8*l.* per h.p. per year, the generator and engine being already installed and doing work during the daytime, and this sum representing fuel and interest on machinery. The attendance should not be more than 60*l.* per year, as the pump-man and night engineer can do the work:—

Estimate of Expense of 5 H.P. Accumulator-Motor.

	£	s.
Interest, at 6 per cent., on 460*l.*	27	12
Repairs to battery	100	0
Repairs to motor, etc.	30	0
Cost of power, at 8*l.* per H.P.	40	0
Attendance in charging at night	60	0
Engineer, at 8*s.* per day, for 260 days	104	0
	361	12

Cost of Running Three Mules and Drivers.

	£	s.
Interest and depreciation (26 per cent.) on 90*l.* ..	23	10
Feed, shoeing, harness, and attention, at 1*s.* 4*d.* per day	72	5
Three drivers, for 260 days, at 8*s.* per day ..	312	0
Total	407	15
Less	361	12
Annual saving on 3 mules ..	46	3
Or, for 4 mules: total expense	543	10
Less	361	12
Annual saving on 4 mules ..	181	18

Or, about 44 per cent. on the investment, under the circumstances assumed.

The rail and track being an important item in the economy of this method, I think that perhaps the cheapest, and at the same time the best method would be to use 20–25 lb. steel rail, and, in laying the track, to place the ties first about 3 ft. apart centre to centre, and on these, and under each rail, to place a string-piece of wood 1½ by 3 in., nailed to the ties, and spike the rails on the top, keeping the stringer-joints and the rail-joints from coinciding. The combination makes a solid track, and a very smooth-running one, and it has the advantage of not lifting easily into very uneven points; the ends of the rails do not jump, as when only laid on the ties, and the track has not the spring to it which is so injurious. (Pocock.)

Electric Mining Motors.

H.P. of Motor.	Speed.	Volts.	Amperes.	Approximate Weight.	Price of Motor.	Size of Lead-covered and Double insulation Cable recommended.	Approximate Cost of Cable per Mile.	Price of Dynamo.
					£	B.W.G.	£	£
2	1200	200	9·5	3 cwt.	30	$\frac{7}{18}$	60	34
4	1200	200	18·5	4½ „	40	$\frac{7}{17}$	80	45
6	1200	200	27·5	6 „	50	$\frac{7}{16}$	100	57
9	1000	250	31·5	10 „	75	$\frac{19}{18}$	120	84
12	850	300	35·0	15 „	100	$\frac{19}{18}$	120	112
16	800	350	39·0	1 ton	125	$\frac{7}{14}$	150	142
20	750	400	41·5	1½ „	150	$\frac{7}{14}$	150	170
25	700	450	45·5	2 „	175	$\frac{19}{17}$	175	200
30	650	500	49·5	2½ „	200	$\frac{19}{17}$	175	225

In mines where the haulage is continuous and the pauses are short, something can be done, as proposed by Ilgner, to average out the load by means of a fly-wheel. This, however, is impossible in many instances, and the only means by which economy can then be improved is by electric storage—that is to say, the employment of accumulators.

The advantages of accumulators under such circumstances are, of course, well recognised. But hitherto their employment has only been contemplated in direct current installations. Where a system of 3-phase distribution is in use, it is essential that a converter be inserted between the accumulators and the distribution network. When this is done, the converter puts back current into the accumulators when the current consumption of the motor is less than the normal output of the generator. On the other hand, when the motor requires an amount of current which would overload the generator, it is provided by the accumulators through the converter. Apparatus of this kind is so perfected that the battery of accumulators can be employed as a true "buffer" battery. In other words, we may say that the 3-phase direct-current converter acts at the same time as a reversible booster—a most important feature. The difficulties inherent in the problem, such as preventing the synchronism in the different circuits from being upset, and avoiding abnormal sparking, have been successfully solved.

The diminution in size and capacity of the mechanical portion of the plant enable the extra cost of the accumulators to be more than covered, and their cost, when the advantages of their employment are considered, is not worth thinking about. These advantages for mining purposes, and specially in connexion with haulage plant, may be summarised as follows: (1) Diminution in size of plant, the accumulators supplying the additional current necessary for starting motors, etc.; (2) Cheapening of working expenses, the accumulators enabling the generators to be equably loaded; (3) Diminution of repair bill from this cause; (4) Instantaneous reserve, the accumulators at once taking up the load without any switching in; (5) Greater reliability, as the generators are less liable to injury through being more regularly loaded; (6) At night, and when work is slack, the haulage motors can be driven by the accumulators; (7) If the generator stops for awhile, the accumulators can be employed to run the motors for pumping, ventilation, etc.

Efficiencies.

The efficiencies of any given electric transmission, from water power to the motor spindle at the further end of a transmission line, may be roughly estimated as follows:—

	Full load Efficiency.
Pipe line	96%
Nozzle of tangential wheel	97 ,,
Tangential wheel	82 ,,
Three-phase alternator generating high-tension current	94 ,,
Transmission line	90 ,,
Step-down transformer	98 ,,
Three-phase motor	92 ,,

Overall efficiency, 58 %.

That is to say, of the potential energy of the water, 58 % is turned into useful work on the electric motor spindles. It would be a poor installation indeed to fall below 50 %.

The Nevada County Electrical Co., California, secure with their plant (transmitting 1000 h.p.) an efficiency of 89·87 % compared with the force actually generated at the water-wheel shaft, the transmission being about 8 miles, and No. 3 copper wire being used to convey it.

On the Kolar goldfield, India, the various motors are distributed over an area of about 6 miles by 2, and the total loss in transmission between the generators at one end and the motors at the other end of the system is approximately 20%: an output of nearly 5000 h.p. at the Cauvery falls power station being requisite for the delivery of 4000 h.p. at the mines. The loss of power in line transmission along the 92 miles between the falls and Kolar is calculated at 13%, when all 6 wires carry the current. If only 3 wires are used, the loss is doubled. (Mervyn Smyth.)

Electrical Equipment of a Mine. (A. W. K. Pierce.)

Following are the power requirements for an average, moderately deep-level mining proposition to be equipped on a 200-stamp basis: Crusher and sorting stations, including belts or other conveyors to the mill, 150 kw.; cyanide works, power required for pumps, etc., in the extractor house, 50 kw.; slimes plant, sludge and solution pumps, etc., 120 kw.; return water pumps, 30 kw.; shops, 40 kw.; miscellaneous small motors on the surface for various purposes, 20 kw.; mine pumping, allowing for pumping 200,000 gal. per day 2000 ft. high in actual running time, 100 kw.; lighting underground, 15 kw.; a total day load of 525 kilowatts. This will require the installation of two 3-phase 50-cycle, 2000-volt, 500-kw. alternators, one machine to be kept as a reserve. The lighting would be done by single-phase, from one selected phase of the machine, and an automatic voltage regulator would be used for maintaining constant voltage on this phase. The motors would be 200-volt, 3-phase induction machines, fed from transformers of the same ratio as the lighting transformers, and the feeders for lighting and power purposes would be separately controlled.

This estimate does not embrace power for driving the battery itself.

Breakdowns. (A. C. Cormack.)

An analysis of the breakdowns occurring during four years to several thousand motors and dynamos, ranging in size from ·5 h.p. to 800 kw., showed the following salient features.

Shaft breakdowns were caused generally by the armatures coming out of centre through wear of the bearings. Bearings frequently wear upward, this sometimes occurring in cases where the armature has been placed nearer the top of the race than the bottom, in order that the upward pull of the magnets may relieve the pressure on the bearings. Owing to the liability of this wear to escape attention, it is somewhat dangerous. Commutators were responsible for 14·9% of the breakdowns. Commutator breakdowns are of two classes. The first, in which the insulation has failed, occurs most frequently between the bars and supporting rings, and results often in the burning out of the armature windings. Defective fastening and keying of the commutator constitute the second class of accidents to which this part of the machine is subject. When the keying is defective, the commutator is driven by the armature wires, and sometimes the commutator and the armature have a slight relative motion on the shaft. This causes breakages of the wires joining the armature to the commutator.

There is a tendency to design machines having higher rises of temperature than experience has shown to be compatible with reasonable durability of the machine. Defects from dust occur most frequently in semi-enclosed motors, for which the provision for cleaning is usually insufficient.

Comparative Cost of Plants per H.P. transmitted. (Beringer.)

Maximum H.P. transmitted.	Distance transmitted.	Cost per H.P. received.			
		Electric.	Hydraulic.	Pneumatic.	Wire Rope
	ft.	d.	d.	d.	d.
5	300	·35	·29	·40	·11
	1,500	·36	·38	·47	·19
	3,000	·37	·48	·58	·30
	15,000	·45	1·40	1·28	1·26
	30,000	·52	2·53	2·43	2·53
	60,000	·85	4·85	4·50	4·92
10	300	·27	·25	·35	·09
	1,500	·28	·30	·38	·17
	3,000	·29	·37	·45	·25
	15,000	·36	·96	·89	·97
	30,000	·47	1·56	1·44	1·93
	60,000	·72	3·21	4·02	4·05
50	300	·23	·15	·22	·09
	1,500	·24	·18	·24	·11
	3,000	·26	·22	·28	·13
	15,000	·29	·46	·44	·38
	30,000	·31	·77	·65	·73
	60,000	·55	1·44	1·09	1·63
100	300	·20	·16	·22	·08
	1,500	·22	·17	·23	·10
	3,000	·23	·19	·24	·11
	15,000	·26	·43	·36	·28
	30,000	·32	·73	·48	·48
	60,000	·50	1·15	·84	1·20

Comparative Cost per Steam Power H.P. received. (Beringer.)

Maximum H.P. transmitted.	Distance of transmission.	Cost per H.P. received.			
		Electric.	Hydraulic.	Pneumatic.	Wire Rope.
	ft.	d.	d.	d.	d.
5	300	2·3	2·55	2·75	1·15
	1,500	2·35	2·9	3·0	1·45
	3,000	2·45	3·2	3·35	2·9
	15,000	2·9	6·6	5·3	5·5
	30,000	3·35	10·65	9·65	10·5
	60,000	5·25	19·25	16·95	23·0
10	300	2·0	2·4	2·55	1·15
	1,500	2·1	2·6	2·7	1·4
	3,000	2·15	2·85	2·9	1·75
	15,000	2·55	5·65	4·55	4·55
	30,000	3·65	7·8	6·35	8·6
	60,000	4·9	14·5	10·55	19·35

Comparative Cost per Steam Power H.P. received—contd.

Maximum H.P. transmitted.	Distance of transmission.	Cost per H.P. received.			
		Electric.	Hydraulic.	Pneumatic.	Wire Rope.
	ft.	*d.*	*d.*	*d.*	*d.*
50	300	1·9	1·65	2·05	1·1
	1,500	1·95	1·7	2·15	1·2
	3,000	2·0	1·8	2·2	1·3
	15,000	2·3	2·95	2·9	2·55
	30,000	2·8	4·25	3·6	4·55
	60,000	4·3	7·9	5·35	11·25
100	300	1·8	1·65	2·0	1·1
	1,500	1·85	1·7	2·05	1·15
	3,000	1·95	1·8	2·1	1·25
	15,000	2·2	2·9	2·65	2·25
	30,000	2·65	4·2	3·15	3·9
	60,000	4·15	6·95	4·55	9·85

Comparative Cost per Water Power H.P. received. (Beringer.)

Maximum H.P. transmitted.	Distance of transmission.	Capital outlay per H.P.			
		Electric.	Hydraulic.	Pneumatic.	Wire Rope.
	ft.	£	£	£	£
5	300	73	40	71	6
	1,500	76	64	94	30
	3,000	79	94	204	59
	15,000	105	348	584	296
	30,000	138	594	1,060	740
	60,000	204	1,206	2,000	1,188
10	300	50	29	58	5
	1,500	53	44	70	22
	3,000	55	63	86	46
	15,000	75	214	208	225
	30,000	100	406	360	448
	60,000	150	784	662	910
50	300	39	16	30	2
	1,500	40	20	35	7
	3,000	41	30	41	14
	15,000	54	89	86	67
	30,000	67	166	143	132
	60,000	97	316	258	265
100	300	31	14	25	1
	1,500	32	20	29	4
	3,000	34	27	33	8
	15,000	44	86	65	40
	30,000	57	160	106	79
	60,000	85	302	187	158

SYSTEMS COMPARED.

Broadly there are two cases to be considered: (*a*) when the source of energy cannot well be itself moved, e.g. when a waterfall is utilised; (*b*) when the source of power (fuel) can be conveyed to any central point. The first is usually a case of long-distance transmission; the second rather of distribution. In the latter case, there is hardly any limit to the number, variety and disposition of the points at which power is to be applied; and similarly, there are many ways in which the several engines may be driven. Thus each engine may have its own boiler or boilers; this necessitates an engineman and a fireman to each, coals and clean water conveyed to each point, and ashes to be got rid of. Many of the engines will be small, and steam cannot be utilised to best advantage in a small engine any more than it can be generated economically in a small boiler. Small engines can hardly be worked otherwise than at high-pressure; it is out of the question to supply each little 10- or 20-h.p. engine with its own condenser, even when a suitable supply of water is available; and the problem of a central condenser station has not received any thoroughly satisfactory solution as yet, although several large Continental mines have erected such. Still more unsatisfactory is the case when some of these smaller engines are required to work at irregular intervals, but always to be ready for work at a moment's notice. The system of supplying each engine with its own boiler, although perhaps the simplest, is certainly the most costly, both as regards labour and fuel, and probably quite as expensive as any in the matter of first cost.

A more satisfactory method, when the various engines are close together, is that of supplying steam from a central battery of boilers; but this has various drawbacks when any considerable distance separates the engines from the boilers. A long range of steam pipes always involves condensation and loss of pressure, however well the pipes may be covered, and the numerous joints kept tight by the help of expansion joints. Conveyance to long distances of very high pressure steam, say at 200 lb. or more per sq. in., is not often attempted. The objection to the uneconomical small steam engine still remains.

Without doubt the best system is a central power-generating station (whether derived from water or steam), and hydraulic, pneumatic or electric distribution. Where steam is necessary, the plant can be erected on the coal-field, and further economy be obtained by utilising the lowest-class coal in Meldrum furnaces. Moreover, the central station can be equipped with large and economical boilers, and heat economisers or feed-water heaters; and the engines can be large, of compound or multiple expansion type, condensing, fitted with variable cut-off gear of the best kind, and, therefore, as economical as they are made.

Preferably such an engine or engines can be employed to work dynamos, needing but the minimum of attention, and utilising the thermal energy of the fuel to the utmost. Cables can be easily and comparatively cheaply carried to the point where power is required; suitable motors installed there are always ready for work, need practically no attention, and start by the simple throwing-in of a switch. Moreover, this system enables the distribution of energy to be carried to a point of subdivision which is impossible with steam.

A few such installations have been erected in England, and many on the Continent, all in connection with mines, and mostly at collieries. Their work embraces both surface and underground operations.

While electric transmission has been chiefly applied so far to conveying power from waterfalls (natural or artificial) to mines and mills situated somewhat inaccessibly and away from fuel, its extension to fields of competition with rail and road transport of coal is not far distant. (Prof. H. Louis.)

Comparative Efficiency of Systems. (Beringer.)

Distance of Transmission.	Electric.	Hydraulic.	Pneumatic.	Wire Rope.
300 ft.	·69	·50	·55	·96
1,500 „	·68	·50	·55	·93
3,000 „	·66	·50	·55	·90
15,000 „	·60	·40	·50	·60
30,000 „	·51	·35	·50	·36
60,000 „	·32	·20	·40	·13

It appears from this table that wire rope is most efficient up to about 3 miles, beyond which electric and pneumatic transmission are most efficient.

Ores and Rocks.

	Lb. per cub. ft.	Cub. ft. per ton.	Cub. ft. per short ton.		Lb. per cub. ft.	Cub. ft. per ton.	Cub. ft per short ton.
Antimony oxide	281	8	7	Lead concentrates, B. Hill	250	9	8
Asphalt	62–106	21–36	19–32	Lignite	75	30	28
Barytes	250	9	8	Limestone	100–200	11–22	10–20
Basalt	170–185	12–13	11–12	Loam	65–100	22–34	20–32
Blende	250	9	8	Marble	165–170	13½	12
Calcite	170	13½	12	Marl	100–140	16–22	14–20
Chalk	95–175	13–23	11–21	Millstone	80–155	14–28	13–26
Charcoal	18	125	111	Mispickel	356	6¼	5½
Chrome ore	274	8	7	Nickel ore, N. Caledonia	80	28	25
„ N. Caled.	125	18	16	Nickel pyrites	468	4¾	4¼
Clay	100–125	18–22	16–20	Nitrate of soda	137	16½	14½
Clayslate	180	12	11	Peat	20–80	28–112	25–100
Coal	50–100	22–44	20–40	Petroleum	44–56	40–50	36–48
Cobalt ore, N. Caledonia	77	29	26	Pitchblende	437	5	4⅔
Cobalt pyrites	310	7¼	6½	Porphyry	170	13½	12
Copper carbonate	237	9½	8½	Potash salts	135	16½	14½
„ grey	300	7½	6½	Pumice	50–60	37–44	33–40
„ native	556	4	3½	Pyrolusite	300	7½	6½
„ pyrites	262	8½	7½	Quartz	165–170	13½	12
Earth	75–100	22–30	20–28	„ broken	95	23½	21
Emery	243	9¼	8¼	Salt	133–137	16½	14½
Felspar	150–168	13½–15	12–13	Sand	90–120	18–26	16–24
Fluorspar	198	11	10	Sandstone	130–160	14–17	13–15
Galena	468	4¾	4¼	Shale	160–170	13½	12
Granite & gneiss	164–172	13½	12	„ decomposed	100	22	20
Gravel	80–105	21–28	19–26	Shingle	90	24	22
Greenstone	199–218	10–11	9–10	Silver, native	625	3½	3¼
Grit	130	17	15	Slate	165–180	12½–13½	11–12
Gypsum	75	30	28	Slimes	100–120	18–22	16–20
Iron ore, hematite	250	9	8	Stibnite	410	5½	5
„ magnetic	338	6½	6	Sulphur	125	18	16
„ pyrites	304	7½	6½	Traprock	170	13½	12
„ specular	304	7½	6½	Zinc oxide	337	6⅔	6
Lead carbonate	400	5½	5				

Wood, 8 × 4 × 4 ft. = 128 cub. ft. = 1 cord.

Avoirdupois and Troy Equivalents.

Avoir.	Troy	Avoir.	Troy	Avoir.	Troy	Avoir.	Troy
oz.	dwt.	oz.	oz.	lb.	oz.	lb.	oz.
¼ =	4·55	9 =	8·20	5 =	72·91	17 =	247·92
½ =	9·11	10 =	9·11	6 =	87·50	18 =	262·50
¾ =	13·67	11 =	10·02	7 =	102·08	19 =	277·08
1 =	18·23	12 =	10·92	8 =	116·66	20 =	291·64
	oz.	13 =	11·84	9 =	131·25	21 =	306·25
2 =	1·82	14 =	12·76	10 =	145·82	22 =	320·82
3 =	2·73	15 =	13·67	11 =	160·41	23 =	335·41
4 =	3·64	lb.		12 =	175·00	24 =	350·00
5 =	4·56	1 =	14·58	13 =	189·58	25 =	364·58
6 =	5·46	2 =	29·16	14 =	204·16	26 =	379·16
7 =	6·38	3 =	43·75	15 =	218·75	27 =	393·75
8 =	7·29	4 =	58·33	16 =	233·33	28 =	408·33

Grains and dwt. in decimals of 1 oz.

gr.	oz.	gr.	oz.	gr.	oz.	dwt.	oz.
1 =	·0021	12 =	·0250	23 =	·0479	10 =	·50
2 =	·0042	13 =	·0271	24 =	·05	11 =	·55
3 =	·0062	14 =	·0292	dwt.		12 =	·60
4 =	·0083	15 =	·0312	2 =	·10	13 =	·65
5 =	·0104	16 =	·0333	3 =	·15	14 =	·70
6 =	·0125	17 =	·0354	4 =	·20	15 =	·75
7 =	·0146	18 =	·0375	5 =	·25	16 =	·80
8 =	·0167	19 =	·0396	6 =	·30	17 =	·85
9 =	·0188	20 =	·0417	7 =	·35	18 =	·90
10 =	·0208	21 =	·0438	8 =	·40	19 =	·95
11 =	·0229	22 =	·0458	9 =	·45	20 =	1·0

Lb., qr., and cwt. in decimals of 1 ton.

	Short Ton	Long Ton		Short Ton	Long Ton		Short Ton	Long Ton
lb.			lb.			lb.		
1 =	·0004	·0004	12 =	·0060	·0054	23 =	·0115	·0103
2 =	·0010	·0009	13 =	·0065	·0058	24 =	·0120	·0107
3 =	·0014	·0013	14 =	·0069	·0062	25 =	·0126	·0111
4 =	·0020	·0018	15 =	·0075	·0067	26 =	·0130	·0116
5 =	·0024	·0022	16 =	·0079	·0071	27 =	·0135	·0121
6 =	·0030	·0027	17 =	·0085	·0076	qr.		
7 =	·0035	·0031	18 =	·0089	·0080	1 =	·0140	·0125
8 =	·0039	·0035	19 =	·0095	·0085	2 =	·0280	·025
9 =	·0044	·0040	20 =	·0099	·0089	3 =	·0420	·0375
10 =	·0050	·0045	21 =	·0105	·0094	cwt.		
11 =	·0055	·0049	22 =	·0109	·0098	1 =	·0560	·05

Mexican Mining Weights.

1 grano	=		·7716	gr.
12 granos	= 1 tomin	=	9·2592	,,
6 tomines	= 1 ochava	=	·12685	oz.
8 ochavas	= 1 onza	=	1·0148	,,
8 onzas	= 1 marco	=	8·1184	,,

The *marco* is the unit for weighing bullion.

Mexicans value an ore at so many *marcos* of silver per *carga*. instead of oz. per ton.

The *carga* = 12 *arrobas* = 304·332 lb.

To reduce *onzas* per *carga* to oz. per ton, multiply by 6·078; and to convert oz. per ton to *onzas* per *carga*, multiply by ·165.

The *monton* in Zacatecas	= 20 quintals	= 2000 lb.	
,, ,, Guanajato	=	3200	,,
,, ,, some places	=	3000	,,
,, ,, others	= 4 cargas	= 1200	,,

Surveying Measures.

Circle, diameter	×3·1416	= circumference.
,, ,,	× ·8862	= side of an equal square.
,, ,,	× ·7071	= side of an inscribed square.
,, area = diameter²	× ·7854.	
,, circumference	× ·31831	= diameter.
Circular inches	× 183,346	= 1 sq. ft.
Sphere, diameter³	× ·5236	= solidity.
,, diameter	× ·806	= dimensions of equal cube.
,, ,,	× ·6667	= length of equal cylinder.
Cylindrical inches	× ·0004546	= cub. ft.
,, ,,	× ·002832	= gal.
,, ,,	2200	= 1 cub. ft.
,, feet	× ·02909	= cub. yd.
Square, side	×1·128	= diameter of an equal circle.
,, root of acre	×1·12837	= diameter of an equal circle.
,, inches	× ·00695	= sq. ft.
,, feet	× ·0000229	= acres.
,, yards	× ·0002066	= acres.
Cubic inches	× ·00058	= cub. ft.
,, feet	× ·03704	= cub. yd.
,, ,,	× ·6232	= gal.
Lineal feet	×1·51515	= links.
,, ,,	× ·00019	= miles.
,, yards	×4·54545	= links.
,, ,,	× ·000568	= miles.
,, links	× ·66	= ft.
,, ,,	× ·22	= yd.
,, chains	× ·0125	= miles.

Surveying Measures—continued.

Lineal miles	× 5,280	= ft.
" "	× 1,760	= yd.
" "	× 80	= chains.
Acres	×43,560	= sq. ft.
"	× 4,840	= sq. yd.

Area of circle = diameter² × ·7854.
" parallelogram = base × height.
" trapezium: divide into two triangles and find area of each.
" trapezoid = height × ½ the sum of the parallel sides.
" triangle = base × ½ height.

Latitude (northing or southing) = cos. of angle of bearing × distance.

Departure (easting or westing) = sin. of angle of bearing × distance.

Level, difference of = sin. of angle of inclination × hypothenuse.

Horizontal measurement = cos. of angle of inclination × hypothenuse.

Inclination, rate of (ratio of base to perpendicular) = cotan. of angle of inclination.

Slope, rate of (ratio of hypothenuse to perpendicular) = cosec. of angle of inclination.

Areas of Circles.

Diam.	Area.	Diam.	Area.	Diam.	Area.	Diam.	Area.	Diam.	Area.
in.	sq. in.	in.	sq. in.	in.	sq. in.	in.	sq. in.	in.	sq. in.
⅛	·012	7½	44·17	20	314·16	32½	829·5	45	1590·4
¼	·049	8	50·26	20½	330·06	33	855·3	45½	1625·9
⅜	·110	8½	56·74	21	346·36	33½	881·4	46	1661·9
½	·196	9	63·61	21½	363·05	34	907·9	46½	1698·2
¾	·441	9½	70·88	22	380·13	34½	934·8	47	1734·9
1	·785	10	78·54	22½	397·60	35	962·1	47½	1772·0
1⅛	·994	10½	86·59	23	415·47	35½	989·8	48	1808·5
1¼	1·227	11	95·03	23½	433·73	36	1017·8	48½	1847·4
1½	1·767	11½	103·87	24	452·39	36½	1046·3	49	1885·7
1¾	2·405	12	113·10	24½	471·43	37	1075·2	49½	1924·4
2	3·141	12½	122·71	25	490·8	37½	1104·4	50	1963·5
2¼	3·976	13	132·73	25½	510·7	38	1134·1	50½	2002·9
2½	4·908	13½	143·13	26	530·9	38½	1164·1	51	2042·8
2¾	5·939	14	153·94	26½	551·5	39	1194·6	51½	2083·0
3	7·06	14½	165·13	27	572·5	39½	1225·4	52	2123·7
3¼	8·29	15	176·71	27½	593·9	40	1256·6	52½	2164·7
3½	9·62	15½	188·69	28	615·7	40½	1288·2	53	2206·1
3¾	11·04	16	201·06	28½	637·9	41	1320·2	53½	2248·0
4	12·56	16½	213·82	29	660·5	41½	1352·6	54	2290·2
4½	15·90	17	226·98	29½	683·4	42	1385·4	54½	2332·8
5	19·63	17½	240·52	30	706·8	42½	1418·6	55	2375·8
5½	23·75	18	254·46	30½	730·6	43	1452·2	55½	2419·2
6	28·27	18½	268·80	31	754·7	43½	1486·1	56	2463·0
6½	33·18	19	283·53	31½	779·3	44	1520·5	56½	2507·1
7	38·48	19½	298·64	32	804·2	44½	1555·2	57	2551·7

Thermometer Scales.

Fahrenheit.	Celsius or Centigrade.	Réaumur.	Fahrenheit.	Celsius or Centigrade.	Réaumur.
212°·0	100°	80°·0	141°·8	61°	48°·8
210·2	99	79·2	140·0	60	48·0
208·4	98	78·4	138·2	59	47·2
206·6	97	77·6	136·4	58	46·4
204·8	96	76·8	134·6	57	45·6
203·0	95	76·0	132·8	56	44·8
201·2	94	75·2	131·0	55	44·0
199·4	93	74·4	129·2	54	43·2
197·6	92	73·6	127·4	53	42·4
195·8	91	72·8	125·6	52	41·6
194·0	90	72·0	123·8	51	40·8
192·2	89	71·2	122·0	50	40·0
190·4	88	70·4	120·2	49	39·2
188·6	87	69·6	118·4	48	38·4
186·8	86	68·8	116·6	47	37·6
185·0	85	68·0	114·8	46	36·8
183·2	84	67·2	100·4	45	36·0
181·4	83	66·4	113·0	44	35·2
179·6	82	65·6	111·2	43	34·4
177·8	81	64·8	109·4	42	33·6
176·0	80	64·0	107·6	41	32·8
174·2	79	63·2	105·8	40	32·0
172·4	78	62·4	104·0	31	31·2
170·6	77	61·6	102·2	38	30·4
168·8	76	60·8	98·6	37	29·6
167·0	75	60·0	96·8	36	28·8
165·2	74	59·2	95·0	35	28·0
163·4	73	58·4	93·2	34	27·2
161·6	72	57·6	91·4	33	26·4
159·8	71	56·8	89·6	32	25·6
158·0	70	56·0	87·8	31	24·8
156·2	69	55·2	86·0	30	24·0
154·4	68	54·4	84·2	29	23·2
152·6	67	53·6	82·4	28	22·4
150·8	66	52·8	80·6	27	21·6
149·0	65	52·0	78·8	26	20·8
147·2	64	51·2	77·0	25	20·0
145·4	63	50·4	75·2	24	19·2
143·6	62	49·6	73·4	23	18·4

Thermometer Scales—continued.

Fahrenheit.	Celsius or Centigrade.	Réaumur.	Fahrenheit.	Celsius or Centigrade.	Réaumur.
° 71·6	° 22	° 17·6	° 5·0	° − 15	° − 12·0
69·8	21	16·8	3·2	16	12·8
68·0	20	16·0	1·4	17	13·6
66·2	19	15·2	− 0·4	18	14·4
64·4	18	14·4	2·2	19	15·2
62·6	17	13·6	4·0	20	16·0
60·8	16	12·8	5·8	21	16·8
59·0	15	12·0	7·6	22	17·6
57·2	14	11·2	9·4	23	18·4
55·4	13	10·4	11·2	24	19·2
53·6	12	9·6	13·0	25	20·0
51·8	11	8·8	14·8	26	20·8
50·0	10	8·0	16·6	27	21·6
48·2	9	7·2	18·4	28	22·4
46·4	8	6·4	20·2	29	23·2
44·6	7	5·6	22·0	30	24·0
42·8	6	4·8	23·8	31	24·8
42·0	5	4·0	25·6	32	25·6
39·2	4	3·2	27·4	33	26·4
37·4	3	2·4	29·2	34	27·2
35·6	2	1·6	31·0	35	28·0
33·8	1	0·8	32·8	36	28·8
32·0	0	0·0	34·6	37	29·6
30·2	− 1	− 0·8	36·4	38	30·4
28·4	2	1·6	38·2	39	31·2
26·6	3	2·4	40·0	40	32·0
24·8	4	3·2	41·8	41	32·8
23·0	5	4·0	43·6	42	33·6
21·2	6	4·8	45·4	43	34·4
19·4	7	5·6	47·2	44	35·2
17·6	8	6·4	49·0	45	36·0
15·8	9	7·2	50·8	46	36·8
14·0	10	8·0	52·6	47	37·6
12·2	11	8·8	54·4	48	38·4
10·4	12	9·6	56·2	49	39·2
8·6	13	10·4	58·0	50	40·0
6·8	14	11·2			

Rules for Conversion of Scales:—

F to C $= (F^\circ - 32) \times \frac{5}{9}$.
− F to C $= (F^\circ + 32) \times \frac{5}{9}$.
C to F $= (C^\circ \times \frac{9}{5}) + 32$.
R to F $= (R^\circ \times \frac{9}{4}) + 32$.
C to R $= C^\circ \times \frac{4}{5}$.

F to R $= (F^\circ - 32) \times \frac{4}{9}$.
− F to R $= (F^\circ + 32) \times \frac{4}{9}$.
− C to F $= (C^\circ \times \frac{9}{5}) - 32$.
− R to F $= (R^\circ \times \frac{9}{4}) - 32$.
R to C $= R^\circ \times \frac{5}{4}$.

Standard Laboratory Screens. (Inst. M.M.)

Mesh (apertures per lin. in.)	Diam. of Wire.	Aperture.	Screening Area.
	in.	in.	%
5	·1	·1	25·00
8	·063	·062	24·60
10	·05	·05	25·00
12	·0417	·0416	24·92
16	·0313	·0312	24·92
20	·025	·025	25·00
25	·02	·02	25·00
30	·0167	·0166	24·80
35	·0143	·0142	24·70
40	·0125	·0125	25·00
50	·01	·01	25·00
60	·0083	·0083	24·80
70	·0071	·0071	24·70
80	·0063	·0062	24·60
100	·005	·005	25·00
150	·0033	·0033	24·50
200	·0025	·0025	25·00

Straits Weights and Measures.

Avoirdupois Weight.

10 Tee	=	1 Hoon.		
10 Hoon	=	1 Chee.		
10 Chee	=	1 Tahil.		
1 Tahil	=	$1\frac{1}{3}$ oz.		
16 Tahil	=	1 Kati	=	$1\frac{1}{3}$ lb.
100 Kati	=	1 Pikul	=	$133\frac{1}{3}$ „
3 Pikul	=	1 Bhara	=	400 „
40 Pikul	=	1 Koyan	=	$5333\frac{1}{3}$ „

In round figures, 1 long ton (2240 lb.) = 16·8 *pikul*; 1 short ton (2000 lb.) = 15 *pikul*. To convert *pikul* to long tons, multiply by 6 and cut off two decimals: thus, 684 *pikul* × 6 = 41·04 long tons.

Goldsmiths' Weight (used only by jewellers).

12 Saga	=	1 Mayam	=	52 gr.
16 Mayam	=	1 Bongkal	=	832 „ (2 Spanish Dollars).
12 Bongkal	=	1 Kati	=	9984 „ (1 lb. 8 oz. 16 dwt. Troy).

Measures of Capacity.

2 Gills	=	1 pau or $\frac{1}{4}$ Chupak.
2 pau	=	1 pint or $\frac{1}{2}$ Chupak.
2 pints or 4 pau	=	1 quart or Chupak.
4 quarts or Chupak	=	1 gallon or Gantang.
10 Gantang	=	1 Para.
800 Gantang	=	1 Koyan.

Cubic Measures.

1 Chang = 30 ft. × 30 ft. × $1\frac{1}{2}$ ft. = 50 cub. yd. (mining).
1 Pasong = 6 ft. × 6 ft. × length of sticks agreed on (firewood)

Equivalent Prices per Kati and per Lb.

Per Kati.		Per Lb.	Per Lb.		Per Kati.
1c.	=	·21*d*.	1*d*.	=	4·76c.
2c.	=	·42*d*.	2*d*.	=	9·52c.
3c.	=	·63*d*.	3*d*.	=	14·28c.
4c.	=	·84*d*.	4*d*.	=	19·04c.
5c.	=	1·05*d*.	5*d*.	=	23·80c.
6c.	=	1·26*d*.	6*d*.	=	28·56c.
7c.	=	1·47*d*.	7*d*.	=	33·32c.
8c.	=	1·68*d*.	8*d*.	=	38·08c.
9c.	=	1·89*d*.	9*d*.	=	42·84c.
10c.	=	2·10*d*.	10*d*.	=	47·60c.
11c.	=	2·31*d*.	11*d*.	=	52·36c.
12c.	=	2·52*d*.	1*s*.	=	57·12c.
13c.	=	2·73*d*.	2*s*.	=	$ 1·14
14c.	=	2·94*d*.	3*s*.	=	$ 1·71
15c.	=	3·15*d*.	4*s*.	=	$ 2·28
16c.	=	3·36*d*.	5*s*.	=	$ 2·85
17c.	=	3·57*d*.	6*s*.	=	$ 3·42
18c.	=	3·78*d*.	7*s*.	=	$ 4·00
19c.	=	3·99*d*.	8*s*.	=	$ 4·57
20c.	=	4·20*d*.	9*s*.	=	$ 5·14
30c.	=	6·30*d*.	10*s*.	=	$ 5·71
40c.	=	8·40*d*.	11*s*.	=	$ 6·28
50c.	=	10·50*d*.	12*s*.	=	$ 6·85
60c.	=	12·60*d*.	13*s*.	=	$ 7·42
70c.	=	14·70*d*.	14*s*.	=	$ 8·00
80c.	=	16·80*d*.	15*s*.	=	$ 8·57
90c.	=	18·90*d*.	16*s*.	=	$ 9·14
$1	=	21·00*d*.	17*s*.	=	$ 9·71
			18*s*.	=	$10·28
			19*s*.	=	$10·85
			£1	=	$11·42

Equivalent Prices per Pikul and per Cwt.

Per Pikul.		Per Cwt.	Per Cwt.		Per Pikul.
1c.	=	·23*d*.	1*d*.	=	4·24c.
2c.	=	·46*d*.	2*d*.	=	8·49c.
3c.	=	·69*d*.	3*d*.	=	12·73c.
4c.	=	·92*d*.	4*d*.	=	16·99c.
5c.	=	1·15*d*.	5*d*.	=	21·24c.
6c.	=	1·38*d*.	6*d*.	=	25·49c.
7c.	=	1·61*d*.	7*d*.	=	29·74c.
8c.	=	1·84*d*.	8*d*.	=	33·99c.
9c.	=	2·07*d*.	9*d*.	=	38·24c.
10c.	=	2·35*d*.	10*d*.	=	42·49c.
20c.	=	4·70*d*.	11*d*.	=	46·73c.
30c.	=	7·05*d*.	1*s*.	=	50·98c.
40c.	=	9·40*d*.	2*s*.	=	$ 1·02
50c.	=	11·76*d*.	3*s*.	=	$ 1·53
60c.	=	14·11*d*.	4*s*.	=	$ 2·04
70c.	=	16·46*d*.	5*s*.	=	$ 2·55
80c.	=	18·81*d*.	6*s*.	=	$ 3·06
90c.	=	21·17*d*.	7*s*.	=	$ 3·57
$ 1	=	1*s*. 11·52*d*.	8*s*.	=	$ 4·08
$ 2	=	3*s*. 11·04*d*.	9*s*.	=	$ 4·59
$ 3	=	5*s*. 10·56*d*.	10*s*.	=	$ 5·10
$ 4	=	7*s*. 10·08*d*.	11*s*.	=	$ 5·61
$ 5	=	9*s*. 9·60*d*.	12*s*.	=	$ 6·12
$ 6	=	11*s*. 9·12*d*.	13*s*.	=	$ 6·63
$ 7	=	13*s*. 8·64*d*.	14*s*.	=	$ 7·14
$ 8	=	15*s*. 8·16*d*.	15*s*.	=	$ 7·65
$ 9	=	17*s*. 7·68*d*.	16*s*.	=	$ 8·16
$10	=	19*s*. 7·20*d*.	17*s*.	=	$ 8·67
			18*s*.	=	$ 9·18
			19*s*.	=	$ 9·69
			£1	=	$10·20

Russian Mining Weights and Measures.

1 doli = ·6857 gr. Troy.

1 zolotnik = 96 doli = 65·8329 gr. = 2·74 dwt.

1 funt = 96 zolotnik = ·9028 lb. Av.

1 pood = 40 funt = 36·114 lb. Av.

62·025 pood = 1 long ton; 55·38 pood = 1 short ton.

1 cub. sagene = 343 cub. ft. = 12·7 cub. yd. = (of auriferous gravel) about 19·3 long tons or 21·6 short tons.

94½ pood wash dirt = about 1 cub. yd.

Doli per 100 pood.		Per cub. yd. Gr.	Doli per 100 pood.		Per cub. yd. Gr.
1	=	0·65	20	=	12·96
2	=	1·29	30	=	19·44
3	=	1·94			
4	=	2·59			Dwt.
5	=	3·24	40	=	1·08
6	=	3·89	50	=	1·35
7	=	4·54	60	=	1·62
8	=	5·18	70	=	1·89
9	=	5·83	80	=	2·16
10	=	6·48	90	=	2·43

(Mercer, Min. Jl.)

PROSPECTING.

Clearing Land.—It frequently happens that before anything can be done in the way of estimating the mineral contents and value of a piece of mining land, a preliminary proceeding will consist in clearing the surface of forest or jungle growth.

But few figures are available as to costs of this proceeding.

An ordinary figure in the Eastern States of America is about 10*l.* per acre, with wages ruling at 6*s.* 3*d.* per 10 hr. shift.

In one instance in New York State, contract labour worked out at 49*s.* per acre (1160 a.), each acre carrying 50–75 cords of wood, mostly small pines, etc. (no large timber), and one man cutting and partially piling (no burning) an average of $\frac{1}{5}$ acre per shift. (Eng. News.)

On the heavy timber of Washington and Oregon, contract prices reach as high as 100*l.* per acre. (Gillette.)

Contract work by Dyaks, in Sarawak (Borneo), is done at the following rates :—

					Per Acre. s.	d.
Clearing primary jungle,	$50	per 100 fathoms square		=	12	0
„ secondary „	30	„	„	=	7	3
„ 6-years' growth	25	„	„	=	6	0
„ 2-years' „	15	„	„	=	3	7

Cutting paths 30 ft. wide, in the respective classes of growth, 25c., 8c., 5c., and 3c. per fath. = 1*d.*, ·3*d.*, ·2*d.*, and ·1*d.* per ft. Similar rates prevail throughout Malaya, where the jungle is always dense, and, when of primary growth, contains many huge buttressed trees.

SUPERFICIAL DEPOSITS.

Superficial deposits may contain not only valuable accumulations of such metals as gold, platinum, and tin (as oxide), various gem stones, and monazite sands, but are often worthy of examination for indications of the existence of lode deposits of minerals which are never worked alluvially, e.g. copper, lead, silver, etc. In all preliminary prospecting, the water-courses first deserve attention. Thereafter come the flat lands (including sea-beaches), and terraces, benches and table lands (in connection with previous river systems).

While gold is the most universally sought mineral in alluvial deposits, and most of the following remarks apply expressly to it, the same general rules hold good with the other objects of search.

Streams.—As a rule those streams which cross the laminations of reefs at right angles, or nearly so, are the richest. Gold is found very rarely in those parts of streams where the current has been the strongest, but generally in the lee or under the shelter of projecting points of rocks, where beaches are usually formed; and wherever such beaches exist there also may gold be looked for with fair chances of success. But the straight courses of streams, especially where they cross the laminations of the bed-rock at right angles, are also often very rich in gold. As a rule, the gold in streams is deposited in the crevices of the bed-rock, which must be laid as dry as possible, and picked up to such depths as the sand descends between its laminations. Very rarely is the gold found mixed up in the gravel lying in the beds of streams.

Terraces.—Terrace prospecting requires more experience, labour, perseverance, and, consequently, more time, than creek prospecting. Terraces are shelf-like accumulations upon the hill-slopes flanking valleys and lakes, and are the remains of old river-beds; so that the rules above laid down for the deposition of gold in streams apply also to terraces. Sometimes a ridge of rock divides a terrace from the newer river-channel, but this is not always the case. The first thing to do in prospecting a terrace is to discover its inlet, and next to find its outlet. This found, the "wash" should be carefully examined for the prevailing indications of gold peculiar to the locality in which the terrace occurs, and also for gold. The whole width of the channel of the terrace from the edge to the high or back rim must be carefully prospected, both along the bed-rock and also through the whole depth of the "wash," from the surface down; for the terrace "wash," being often deposited at different geological times, not infrequently contains gold in layers one above the other. Such terraces are very numerous in some countries; and a study of the river systems of the present, as well as of the preceding, geological ages, will assist the prospector greatly in his choice of a likely spot where to set in. The general fall in the country lying along the main directions of the present streams indicates the leading course upon which alluvial gold has been deposited, the consideration of which is often a reliable guide to the level of terraces which are covered up completely by landslips, or in which either the inlet or the outlet is hid from view.

In looking for new deposits of this description, the prospector should not be deterred by elevation, for gold in really astonishing quantities has been got at Mount Criffel, upwards of 4000 ft., and Mount Pisa, nearly 6000 ft., above sea-level, both in New Zealand. The author has worked on gold gravels in Australia lying 600 ft. vertical above the present level of the river which deposited them, and has found rich alluvial (or more strictly detrital) deposits of tinstone on granite ranges in Malaya at more than 4000 ft. elevation.

Shallow bench gravels can occasionally be prospected by diverting a high stream, or water from a ditch, in a direction transverse to the gold-bearing channel. The water will ground-sluice a trench to bed-rock, thus cross-cutting the ground. Such prospecting is done in Alaska. (Purington.)

Bench gravels covered by heavy overburden are usually prospected by "drifts" (levels). Drifts require timbering, and are more expensive than shafts, but give a more satisfactory test of the ground. In rich paystreaks, in Alaska, the running of prospect-drifts often more than pays the cost. Owners of claims sometimes get their ground partly prospected by letting out the right to drift to two or more men, who pay a royalty on the gold they take out. (Purington.)

Boring.—To ascertain the general value per cub. yd. of the deposit, samples may be taken by means of boring.

If the material is not too bouldery or too sandy, the simplest implement for the purpose of testing the deposit is what is called in America a "post-hole digger." This can be made by a blacksmith out of a piece of sheet steel, say $\frac{1}{16}$ in. thick and 2 ft. wide, bent into a cylinder, 5 or 6 in. diam., and fastened to the end of a stout rod as a handle. With this contrivance, holes can be put down quickly to a depth of 10, 15 or 20 ft. through ordinary soil and fairly stiff clay; and if the mineral sought is soft or disintegrated, samples of the bed can be brought up from over 20 ft. A contrivance of this kind, devised by the superintendent of a Florida phosphate mining company, has come into general use for prospecting for phosphate rock beds in that State.

For reaching greater depths, or for work in a country where the soil contains boulders and sand, other methods must be used. The simplest of these is a spring-pole drill, a heavy drill 3 in. or more wide, suspended from one end of a pole made of a springy sapling. Two or three men with such a drill can do good work as deep as 50 ft. By means of a "sand-bucket" (a piece of pipe slightly smaller than the hole, with a valve in the bottom), samples of drillings can be brought up. In case a very hard boulder is struck, it is best to change the position of the drill; and if fine sand gives trouble, the hole is kept open by driving down a piece of pipe to follow the progress of the drill. In holes of moderate depth, this pipe can be pulled out again without much difficulty.

For putting down still deeper holes, a heavier drill of the same type is commonly used. Such drills get their percussive motion from a rope passing over a pulley, and a motor at the other end to give a stroke of varying length. The power may be from any source. A small rig will do good work for 100 ft. or more, while larger drills, with steam power, such as are used in the Pennsylvania oil-fields will work down to 2000 ft. Drills of this type have been used in testing the comparatively shallow iron-ore bodies in Staffordshire and in Minnesota, and are the regular devices employed in hunting for pockets of lead and zinc ore in Missouri.

As all percussion drills chip the rock into bits to be brought up by the sand-pump, their testimony may be misleading. In the case of an iron-ore body, the bits of steel worn from the drill itself may make the pulverised ore brought up assay too high in iron. With any deposits containing layers of high and low grade rock, the record of the drillings may prove very unreliable. For instance, in drilling for zinc ore, a run of rich ore, 1 ft. thick, occurred in broken ground with barren rock below it. The ore from this seam, through the churning of the water in the hole made by the drill, broke off and fell down the hole, and contributed bits of blende to the drillings brought up by the pump from a depth of several feet below. Instead of having an ore body 25 ft. thick, as the drillings indicated, there were only a few narrow runs, altogether too small to pay for working.

In Fig. 26 is shown a plant for hand boring, with a selection of the most useful tools. The shear-legs *a* may be of wrought-iron tube for strength and lightness, while the winch may be driven by hand or by motor, fast and loose pulleys being provided; *b* is an auger for working into clay or stiff soil; *c* is a flat V chisel for soft rock, gravel, or hard soil; *d* is a T V chisel for stone and other hard but fine-grained material; *e* is an X chisel for hard rock; *f* is a sand-pump for bringing up the debris cut by the chisels; *g* is a worm and *h* is a bell-box for recovering tools from holes.

Another type of rotary drill, used with success for several kinds of work, has for its cutting edge a series of steel teeth of peculiar shape, well tempered. These teeth, when pressed against the rock as the drill revolves, break off bits of the rock, which, like the diamond drill rock shavings, are removed from the circular furrow by a stream of water forced down from surface. In rock that is not too hard, this "calyx" drill, as it is called, has done some remarkable work. The cores that it brings up are of larger diameter than those made by diamond drills.

The churn drill used in the Mesabi iron-fields consists of a chisel bit attached to a hollow rod, which is extensible to any desired length by simply screwing on additional sections. A flexible coupling attaches the upper end of the hollow rod to a pump, and this drives water down the rod and out of the perforation at the bit. This water returns to surface inside the casing, clearing the bit of choppings, and bringing them to surface. The casing is pipe of 2–3 in. diam., according to depth of hole and character of material through which the drill is passing. An oscillating engine of 6–7 h.p. is used, and the churn motion of the drill is given by passing a rope two or three times around the drum, and pulling the rope tight or easing it off. The drill bit is rotated in the hole by

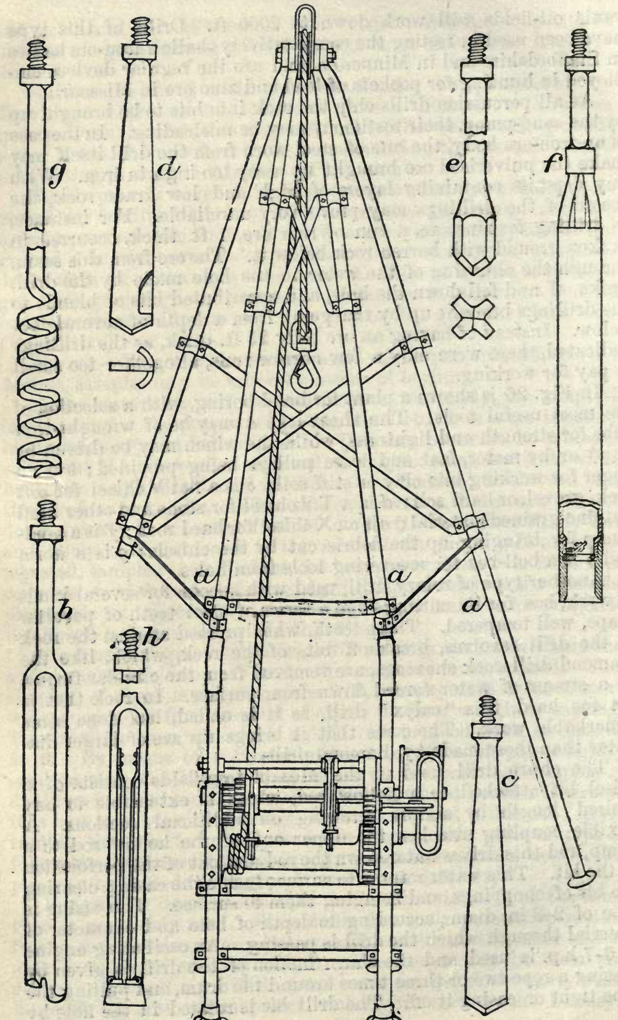


FIG. 26.—BORING PLANT.

hand, a man seating himself in the top hamper for this purpose. This drill is built by several local foundries, and costs, complete, about 150*l.*, with engine, pump, and a 36 × 84-in. upright boiler that can be hauled through the woods on skids. Such a drill drives its bit very rapidly through surface or soft ore, but is of less avail in chert or boulders. The latter are blasted out by dynamite, small sticks of which are let down the hole, and fired after the casing pipe has been drawn up out of the way. Often it is necessary to break out the hole by dynamite, in order to drive down the casing. If in ore of uniform grade and texture, it is not necessary that the casing shall follow the bit closely, but where there are sandy seams, or where grades change rapidly, the casing is kept close to the bottom of the hole. As the grade of ore driven through is estimated from the sludge that washes out of the hole with ascending water, the utmost care is necessary that no enlargement of the hole at bottom be permitted. (Woodbridge, E. & M. Jl., '05.)

A boring machine has been specially designed by Stanley B. Hunter for dealing with a class of deep basaltic lead country in Victoria, which is intermediate between shallow ground and that extremely deep ground (up to 600 ft.) where the overlying stratum is mainly hard basalt; its principal features are :—

Easy and cheap transport, either by road or rail.

Rapid starting of plant: unloading, erection, and commencement of boring from time of arrival on ground averages about 4 hours.

The derrick forms, when lowered down on to road wheels, the carriage, upon which all tools, rods, casing, and camp equipment are carried.

Ample room for men to work round the borehole, a very necessary requirement when casing is being worked down through the drift.

Rigidity of derrick when a great lifting strain is exerted on rods or casing.

The substitution of an oil engine for steam power. At machines now in use, "Simplex" oil engines of colonial manufacture are giving complete satisfaction.

The lessening of one man per shift, the staff comprising 4 men, who work in two shifts when boring in basalt or clay, the 4 men working on the one shift only when in drift, handling casing, or shifting from bore to bore.

The machine is capable of boring a hole to 8 in. diam., and, if necessary, calyx cutters and hollow rods can be used to produce a core. The total equipment, including engine for boring to 250 ft., is about 7 tons.

The chopping motion adopted gives an absolutely clear drop which can be regulated at 1–12 in.

The whole mechanism is simple, and requires only such skilled labour as would be necessary for a hand boring-plant. The machinery, apart from the oil engine, consists of the tripod or derrick, at the lower end of the two jointed legs or carriage frame of which are assembled the several parts for hoisting rods and pump, feed gear for chopping-rope, worm and tangent wheel, and cam for imparting the chopping or percussive motion to the rods and bit. At the top end of tripod are three grooved pulleys to receive the chopping, hoisting, and sand-pump ropes.

The chopping motion is imparted to rods by a flexible steel rope, one end of which is attached to the upper end of the surface rod. After passing over and being suspended by the centre pulley at tripod head, the other end of rope is wound upon a small feed drum (the groove of which is the width of rope used), carried on the same transverse shaft as a tangent wheel, to which the feed drum is attached, and which is rotated by a worm and hand wheel; the said chopping rope is pressed down by a revolving sheave placed at the outer end of a rocking knee-piece, and this knee-piece is pressed down by a cam wiping on to a roller situated about midway on the longer arm of the knee-piece. At the end of the shorter knee-piece is a buffer striking on to the base piece of frame, and preventing the roller from striking the cam when running at a maximum speed.

The rod-hoisting drum is stepped, or of two diameters, and this, in conjunction with a two-speed step-cone pulley on engine, gives four variations in speed, as occasion arises.

The winding drum slides in and out of a saw-tooth clutch, forming part of the main pinion wheel, and the pump-hoisting drum is operated by means of a friction clutch.

The tripod is made by strongly joining two legs (of channel steel) together, and having a wide movable sole-piece at bottom; the third leg is built up of two side-pieces of timber bolted together with distance-pieces between. At the bottom end are two rollers, and the tripod is hoisted by drawing the bottom end of this third leg towards the centre of the machine. To do this, a winch handle is placed on the small driving shaft, and a rope, fastened at one end to a shackle on the bottom end of third leg, is wound up on a small hoisting drum, until the base line between the two joined legs and the one in motion is about 16 ft. Rotating motion is imparted to the rods by a horizontal tangent and worm wheel. The former has a circular orifice in the centre, to admit of rods and casing being passed through. This is placed over the borehole, and so forms the deck head. Four vertical standards let into the upper surface of the horizontal tangent wheel press on to the rod or casing clamps, and so turn them round. Three speeds are obtainable with the mechanism, and the chopping and rotating motions can be used singly or together.

Boring Costs.

Cost of some trial borings for ironstone in the Barrow district made with the aid of a steam winch and free-falling tool :—

Depth of Hole.	Diameter of Hole.	Cost per yd.	Cost of Labour alone.	Time occupied.
yd.	in.	s. d.	£ s. d.	weeks.
126	6 to 2	7 10¼	49 10 0	15
124*	,,	9 7	59 8 0	18
50	,,	9 10¾	24 15 0	7½
63	,,	9 5	29 14 0	9
76½	,,	6 0½	23 2 0	7
88	,,	8 3	36 6 0	11
48	,,	11 0	26 8 0	8

* The strata passed through in this hole were as follows, proceeding from the surface downwards :—45 ft. pinder, 75 ft. red sand, 3 ft. white sand, 30 ft. red sand mixed with clay, 150 ft. red sand, 30 ft. red and white sand, 6 ft. white sand, 6 ft. shale, 4 ft. ore, 6 ft. clay, 1 ft. ore, 2 ft. stone, 9 ft. ore, 2 ft. black shale, 3 ft. stone —total, 372 ft.

According to recent figures, prospecting in California by Keystone driller (boring machine) ranges from 4*s.* to 10*s.* per ft.

The sinking of bore-holes is made quite a business in America; the price ordinarily charged for depths of 200 to 250 ft. is about 5*s.* per ft. The actual cost when the speed is 100 ft. per week (6 shifts of 10 hours) is quoted at 2*s.* to 2*s.* 6*d.* per ft. ; of this, labour (at 8*s.* to 14*s.* a day) accounts for 1*s.*; fuel, oil, rope and repairs, 1*s.*; the balance being for incidentals.

The cost of boring depends on price of labour and fuel; with a 6-in. core churn drill, it is about 10*s.* per ft. in California and in the Seward Peninsula, but rises to 15*s.* in the interior of Alaska. Unless a contract party can be found ready equipped for the work, drilling is more costly than pitting in Alaska. (Purington.)

Approximate cost per set of boring tools, including rigger, rope, shear legs, and windlass for depths of 300 ft. and upwards :—

	£		£
To bore 30 ft.	20	To bore 250 ft.	75
,, 50 ,, ...	36	,, 300 ,, ...	120
,, 100 ,, ...	45	,, 500 ,, ...	155
,, 150 ,, ...	58	,, 800 to 1000 ft. ..	195
,, 200 ,, ...	70		

Daily expense of running test hole driller (Calif.): drillman, 14*s.* 6*d.*; fireman, 10*s.* 6*d.*; teamster and team, 14*s.* 6*d.*; Chinese rocker man, 6*s.*; ½ cord wood, 12*s.* 6*d.*; repairs, 21*s.*; hire of drill, 21*s.*; total, 5*l.* Drilling 10 ft. per day, the cost would be 10*s.* per ft. Contracts are let at 10*s.* 6*d.* The number of feet drilled per day varies greatly, according to the character of the ground and the season of the year. In soft ground, 20–30 ft. per day can be drilled. In winter and spring, when the surface is wet and soft, the cost of moving the machine from hole to hole, and of bringing in wood and water, increases. (Knox, Tr. Inst. M.M.)

Computing Values.—The unit of ground tested per hole must vary with many conditions. Rarely is it more than 10 acres (in dealing with iron and zinc deposits). Commonly it ranges from 1 bore per acre to 1 per 5 acres, even in examining auriferous gravels; but it cannot under normal conditions be at all reliable at wider intervals than 50–100 ft.

The value of the ground is computed from the cubic contents of the hole and the amount of mineral obtained. In calculating the cubic contents in Californian practice for valuing gold-dredging ground, a factor called the "pipe constant" or "pipe factor" is applied. The inside diam. of the casing commonly employed is $5\frac{7}{8}$ in., the outside diam. $6\frac{1}{2}$ in. It is the practice to base calculations on the outside diameter. Figuring the cubic contents per ft. of pipe of $6\frac{1}{2}$ in. diam. gives ·23 cub. ft., but, in practice, it is found that ·23 is much too small, giving valuations too high, such valuations being not borne out by subsequent dredging. Some engineers use ·25 as a factor. The factor recommended by Radford is ·27. This factor was obtained by sinking a shaft 3 ft. diam. to a depth of 34 ft., using a drill-hole as the centre. The gold obtained from the shaft corresponded almost exactly with the gold obtained from the drill-hole when using ·27 as the factor. This question of factor is very important, since the difference in results between ·25 and ·27 may amount to 3–4*d.* per cub. yd. The proper factor having been determined, the computation of the results obtained from any hole is very simple. The depth of the hole multiplied by the factor gives the number of cub. ft. in the hole, and, the quantity of gold obtained being known, the value per cub. yd. is reckoned by simple proportion. The drillings, raised by means of a sand pump, are discharged into a wooden trough 12 ft. by 1 ft. by 1 ft., from which they are run into the riddle of a rocker, wherein they are rocked in the ordinary manner, great care being taken to save all the fine gold. The practice of cleaning up after each pumping (approximately after each foot of hole drilled), instead of making one final clean up, is sound; data as to the occurrence of rich streaks and lean streaks, fine gravel, depth of overburden, etc., are thus obtained. After the last clean-up, all the gold from the hole is collected by means of quicksilver. The amalgam is treated with nitric acid, leaving the gold, which, after a thorough washing with hot water, is dried, annealed, and weighed. (Knox, Tr. Inst. M.M.)

Common experience is that tests derived from boring are most unreliable. While in some cases 95 % of the bore values has resulted from actual subsequent mining operations, in other cases 35 % only has been obtained. No boring is really complete without some pitting.

Pitting.—While pitting is generally more costly than boring, it is infinitely more satisfactory, and can be done in all shallow deposits unless there is very heavy water. Pits show much better the character of the wash as apart from its richness, the nature of the difficulties to be encountered in the shape of boulders, trees, and water, and the condition of the bottom or bedrock. Pits need not be so numerous as bores usually.

Most rudimentary appliances may be used in pitting—baskets, skins and canvas all serve admirably for hoisting solids, and old kerosene tins make excellent water buckets. Labour may be minimised by utilising supple poles for lifting from shallow depths, and suitable windlasses can often be rigged on the spot at absurdly small cost. Examples are shown under Hauling and Hoisting. A notched or wooden spiked pole is an efficient ladder.

If water is very troublesome, and gets beyond baling, it is quite a simple matter to erect a small portable double-acting plunger pump, driven by rope from grooved wheels actuated by handles at surface; 6–8 ft. of suction hose (attached to the pump) and 6-ft. lengths of delivery pipe (socketed) answer all ordinary requirements. A 2-in. delivery pipe, with one flange for fastening to a short plank across the pit top, has sufficient rigidity to keep the pump steady, especially if supplemented by a second piece of planking below.

DEEP DEPOSITS.

Vein Outcrops.—Whilst working up the stream, attention must be paid to the banks on each side, especially where they show a section of the rocks, so that no outcropping vein may be overlooked. Should a quartzose or pyritous vein be discovered, specimens must be broken down for examination.

As all alluvial gold is the result of the breaking up of auriferous veins, an effort should always be made to trace the former to its source. Many valuable reefs have been found in this way. The occurrence of "float," or stray pieces of quartz, if of a promising appearance, should be taken as a guide also. Obviously, some judgment is necessary in estimating the best direction for search, and in determining the drainage area to which the float belongs. Sometimes there may be much difficulty in locating the reef, owing to disturbances or to accumulation of detritus hiding the nature of the subjacent rock. A knowledge of the neighbouring reefs and their gold is of great assistance. Where the gold specks and pebbles are much worn, they may represent a former riverine deposit, or its remains, scattered far from its previous resting-place.

Where the gold is heavy and the rock fragments are angular, the reef cannot be very far distant. Occasionally there may be a distinct feature in all the veins in a district, such as a peculiar band of defined colour, and the recurrence of this indicator should then stimulate search for the accompanying reef. Coarse alluvial gold is not always incompatible with fine reef gold as a source, because the reef gold may be so fine in general as to lend itself to very wide distribution when once liberated, while the rarer coarse grains would not be transported far.

Much intelligent prospecting for auriferous reefs is done by "loaming," as a preliminary to cutting experimental trenches or sinking trial shafts Loaming, long practised in searching for tin in Cornwall, consists in washing surface prospects from the bases and slopes of the ranges, until specks of gold or specimens are found to be obtainable with tolerable frequency within certain limits. The prospector then proceeds to trace the gold up-hill to its source, narrowing the limits of his work, as by slow and patient search he finds that he can obtain surface prospects of gold up to a certain point or line, but no farther; he then proceeds by means of trenching, etc., to search for the reef. As may be imagined, the work is one requiring patience, energy, and insensibility to fatigue, together with good memory for locality, as the prospector frequently has to work along a steep and scrubby mountain side, selecting his prospects, numbering them, and placing them in his "loam-bag," noting the localities, and then conveying his samples many hundred feet down the ranges to water. If he be fortunate in obtaining prospects of gold, he has to find his way back to the spots the samples were taken from, so as to continue his up-hill search, and trace the gold to its matrix. In many new discoveries made by means of the above system there was no surface indication whatever of the existence of a reef; nothing visible to the eye that would have led one passing over the ground to believe that it was there, the soil and debris completely concealing its outcrop, until by means of "loaming," the prospector was enabled so nearly to ascertain its position as to expose in it a trench of not many feet in length.

Having found a vein, the next essential thing is to determine its course or direction, and make sure that the area located really covers the vein. If the surface debris is, say, 10 ft., or less, it will not take long to dig some surface cross-cuts the whole width of the vein, and find out whether the ore is really in place, or whether it is slide or float-rock from higher up the hill-side. Having determined the general direction of the vein, and opened it, if the surface is not too deep, in at least 4 places on a claim 1500 ft. long, the next thing is to sink deep enough to determine accurately the dip or inclination. Then, careful tests of the rock from the bottom of the different pits or cross-cuts will show whether the vein grows richer in one direction than in another, or carries fairly uniform values. Having ascertained this, the prospector is in a position to go to work systematically. He can decide whether to sink his shaft deeper or run a tunnel, and where is the best point to locate it. Generally speaking, it is wise to "follow the ore." There is no better way by which the ground can be shown; and the ore taken out will go a long way toward paying expenses. Even if the ore is not very rich, it is tangible evidence of the value of the rock in the vein. Again, shafts, vertical or inclined, that follow the vein, will show up any changes in the ore, and give a large quantity "in sight." Thus 4 shafts 50 ft. down in a vein 4 ft. wide would expose 1680 sq. ft. of the vein. A tunnel driven 200 ft. through rock to cut the vein would expose but the cross-section, i.e. the height × width of the tunnel, at most 100 sq. ft. Of course, when an adit can be driven into the vein, nothing else is needed. And if the vein dips vertically, and the slope of the hill is 45°, a 200-ft. tunnel driven at right angles to the course of the vein, will cut the vein at a depth of 200 ft. It is, then, better to tunnel than to sink this distance. The rock removed can be trammed much easier than it can be hoisted, and the tunnel will drain itself. But the knowledge gained from an apparent outcrop or a single shaft may be very misleading. A common error is to mistake the slope of a hill—it is much more likely to be 25°–30° than 45°, and the tunnel to open ground at 200 ft. may have to be driven 400 ft. In a tunnel of this length, the chances are that the ventilation will become bad, long waits will be necessary after each blast, work will proceed slowly, and finally some sort of ventilating plant will be necessary. Again, the pay value may be confined to a shoot or chimney, the limits and inclination of which, in the plane of the vein, are altogether unknown. In such a case, a tunnel expected to strike the vein in 200 ft. (approaching from the foot-wall side), may fail to reach it in 500 ft. When finally cut, the vein may be absolutely barren. The section exposed is just the size of the tunnel; and whether there is rich ore to right, to left, or below, it is impossible to say, except by following the vein.

DIAMOND DRILLING.

For great depths, in hard rocks, and in cases where it is desired to secure sections of the ground passed through, a core drill is essential. The diamond drill is the best known of this class, and has become a necessary adjunct of mining all over the world. In this drill, a tubular bit or ring, of soft steel, has set in its lower face a varying number of diamonds. These are not gem stones, which are expensive and liable to break, but the tougher black diamonds, or borts. The rotating diamonds cut an annular hole, and a stream of water, forced down from surface, removes the cuttings: the cylindrical core left is broken off by a special contrivance, and hoisted, with the bit, from time to time. The smallest diamond drills are driven by hand power, and will take cores from a hole 200 ft. deep. The largest drills will cut cores of 4 in., and bring up sections of rock from a depth of 1 mile. The diamond drill cannot go through gravel or broken quartz without danger of injuring the cutting faces of the diamonds, or tearing them out of the bit altogether. Consequently, in prospecting it is usual to drive an iron stand-pipe through the surface soil or gravel to bed-rock, or, in case the soil is full of boulders, to sink a shaft to the solid rock before putting the drill to work. Owing to the tendency of diamond drills to follow joints or fissures in rock, it is difficult to maintain a truly straight bore, and thus an ore-body may be missed, or a core may be got which conveys a wrong idea of the thickness of the deposit, from having deviated in passing through it. Further, in gold-reef prospecting, the testimony they offer as to milling values of ore encountered is most unreliable, because the sample represented by the core is absurdly disproportionate to the mass under consideration. Nevertheless, the diamond drill has been widely availed of for locating reef continuations where great faults and slides occur, and for proving ground in depth, and determining sites of new shafts. It is particularly applicable to bedded deposits, such as coal seams, and is much employed in deep well boring.

Where igneous overflows hide outcrops, the existence of several gold reefs has been determined or verified on the Rand by diamond-drilling; and in prospecting the Black Reef, the diamond drill has been used extensively to define the shoots of better-grade ore. Although in many places where bore-holes have been put down the Black Reef could have been reached by prospecting shafts, the amount of water met with in the dolomite above this reef prevents the ordinary rate of shaft-sinking from being attained, and turns the balance in favour of prospecting by diamond drills. It is, however, in proving the continuity of reefs in depth that diamond-drilling has been of greatest use on the Rand; and in these deep bore-holes, sections of the formation are obtained which indicate almost the exact distances between various reefs.

It is, however, sometimes a very open question whether an ordinary cross-cut would not be cheaper in the end. If ore is encountered, this has to be driven as a rule, and the diamond bore is an additional cost. Its prominent advantage is in the matter of speed, 100 ft. a week being commonly driven where a cross-cut would not advance 27 ft. Many thousand feet were drilled by the author at Lucknow, New South Wales, searching for the irregular but rich ore bodies that there occur; but while in no case was a discovery made by it, on the other hand, several isolated "bonanzas," which the drill had failed to locate, were afterwards encountered in cross-cutting.

Without the diamond drill, the great lead deposits of Missouri would never have been discovered, and could not to-day be worked. The ore is galena, disseminated through horizontal strata of magnesian limestone. The ore-bodies are irregular in outline, in depth, and in elevation above the sandstone that invariably marks the lower limit of the ore. They are overlaid with horizontal strata of generally solid magnesian limestone, of uniform texture, and usually free from openings or chert. But it happens not infrequently that the water supplied to the bit fails to come to the surface, escaping at some porous stratum, or following a bedding plane. Many schemes are employed to remedy this. Bran, sawdust or manure (mixed with water to a thin consistency) is poured into the leaky holes, and is effective where the escape is through a porous stratum; but an "opening" is best filled with neat cement, poured in and allowed to set. No method is effective, however, where water flows along the "opening." Sand is apt to fill the space about the rods and bit as soon as the pump is stopped, and an opening filled with standing water is dangerous, as it allows the sludge to accumulate near the hole; part of this will slip into the hole, and the bit may be stuck, and lost.

As an adjunct in prospecting and developing the low-grade copper ore deposits of the Boundary, the diamond drill has proved an unqualified success. These great ore-bodies exist in irregular masses, with usually no very well defined walls (except in contact with limestone). Further, the deposits are frequently separated by barren zones, so that when the boundary of an ore-body is reached, it is quite impossible to predict whether more ore will be found beyond, or not. Commercially, the low grade of the ore prohibits cross-cutting barren ground in order to prospect it, the only allowable dead work being that necessary to reach known deposits.

On the subject of deviations of deep bore holes, there has been much recent literature (especially H. F. Marriott and J. Kitchin, in Tr. Inst. M. M.), without any marked conclusions being arrived at. On 235 Rand bore holes, 45 reaching over 3000 ft., surveys prove enormous deviations. Vertically, they average 26° up to 2000 ft., and reach 66° at 4000 ft.; and horizontally (at 4000 ft.), they range from a minimum of 700 ft. to a maximum of 2250 ft. Obviously, the diamond drill is incapable of going straight to any reasonable depth. It would seem that the flatter the strata the greater the deviation, but no other sound or guiding conclusion can be drawn from all the observations and surveys yet made.

In deciding geological questions, the diamond drill has a distinct value, as also in searching for deep-seated water, and proving bedded deposits. But in lodes it is of minor importance. In a new country, where there are no shafts, it is difficult to state whether any apparent flattening of a reef, as shown by a borehole cutting it sooner than expected, is really a flattening of the dip or an upthrow caused by a fault. Again, the percentage of core is never 100, and although this is a matter which can be easily allowed for in estimating the thickness of reef cut, yet some portion (rich or poor) is lost, the value of which can never be known. Moreover, the cost of boring horizontally is practically as great as that of driving a cross-cut.

Setting Bits.—The setting of carbons in the bit (Fig. 27) is a matter demanding no little skill and care.

After screwing the blank bit into the setting block, the first step is to divide the bit into as many equal parts as the number of diamonds to be used (generally 6 or 8, but sometimes up to 14),

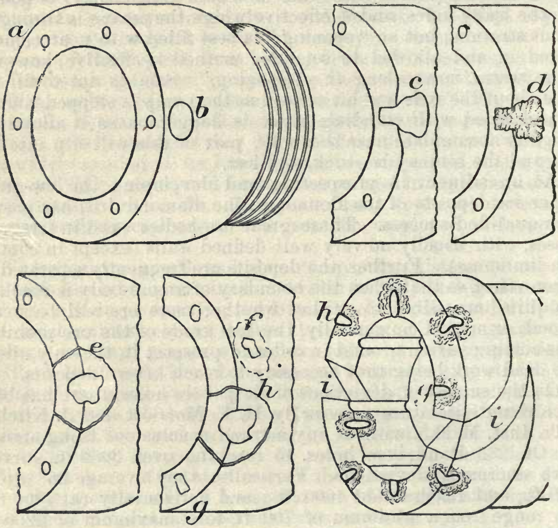


FIG. 27.—SETTING DIAMONDS IN BIT.

and mark with centre punch, as at *a*, where they are to be placed. Breast drill and twist bits are then used to bore a horizontal hole *b* in the side of the bit; each diamond should be studied separately, and a hole be bored in proportion to its size. As the outside diamonds can be more conveniently set than those on the inside, the largest should be selected for this purpose, and set first Horizontal holes are used for the outside diamonds, and vertical holes for those on the inside of the bit. After boring, the hole is chipped out by small chisels until the diamond fits very snugly in the metal, as at *c d*, and projects $\frac{1}{64}$ in. above the face, and the same distance from the outside and inside rim of the bit.

When the diamond is fitted in place, and the proper measurement is obtained, the metal is drawn up or closed round it, as at *e*; this is done by first making a cut, with a blunt-edged chisel, across the face of the bit, about $\frac{1}{8}$ in. from each side of the diamond, and all around it on the outer surface, then by using a dull-pointed chisel or caulking tool, the metal is gradually driven towards the diamond.

In order to get the diamond placed to the best advantage, it is often necessary to cut away more metal than it is possible to replace by driving up the original metal on the bit; in such cases, thin wedges made of horse-shoe nails, or copper wire hammered flat or wedge-shaped, should be used to fill up the space around the diamond before the caulking takes place; many operators prefer to make a bed of copper foil for seating the carbon, in any case. The setter should endeavour to place the diamond in such a position that it will have a sharp cutting edge on the face of the bit, and at the same time leave a broad strong side or surface for the clearance on the outside of the bit, as at *d*, which will obviate much reduction in size of the bit.

The diamond should be held in place by the third finger of the left hand, and the chisel or caulking tool be held between the thumb and first and second fingers. First drive up the metal on the face of the bit until it holds the diamond in its proper position; then the caulking on the sides can be done. Care should be taken that the diamond does not move from its proper position, thereby destroying the gauge or measurement. When the metal begins to bear on the diamond, a finer-pointed tool should be used; light blows are struck, and the metal is closed in carefully. It is possible to break the diamond by caulking the metal too tightly, and also by driving the metal to fill an opening near the corner of the diamond while the metal may be pressing hard on it at another point; it is, therefore, necessary to drive the metal so that it will be brought to press uniformly all around.

When the rock is extremely hard, two extra diamonds are set on the outside of the bit, and directly opposite each other, as at *f*; these assist those on the outer edge of the face in retaining the diameter or size as first set. All bits should be set so as to be of the same outside and inside diameter as the first one used.

The diamonds are set alternately inside and outside, as at g *h*; those on the outside cover the outer half of the face, and cut the outside clearance, while those on the inside cover the inner half of the face, and cut the inside clearance for the core to pass up freely.

Some makers fancy a bit with channels cut, as at *i*, which are intended to give greater freedom of exit for the mud produced by the machine in operation.

Whenever the drill is withdrawn from the hole, the bit should be carefully examined; if any of the diamonds are found to be loose, or the die is worn away so as to leave some of them unprotected, the metal should be re-caulked around them. When the bit is so badly worn that the diamonds are greatly exposed, they should be cut out and re-set in a new blank.

If, while drilling, some of the outside diamonds are chipped, so that the size of the hole becomes reduced, when the next bit is introduced that portion of the hole bored after the diamonds were broken should be re-bored, so as to be the full size of the standard bit, as any attempt to force the new bit down into the reduced hole, either by trying to turn the rods with tongs, or otherwise, will surely destroy the outside diamonds.

To remove diamonds from an old bit, file a cut across the face of the bit about $\frac{1}{8}$ in. from each side of the diamond; then chisel the metal back and chip it away until the diamond can be forced out by light taps of the hammer on a small copper rod.

Sometimes a carbon is dislodged from its setting, generally through applying too much pressure when passing through hard broken rock. This should be detected by an experienced drill hand by the sound produced. It must be recovered as soon as possible, because not only does it impede the work of the drill, and in itself constitute a serious loss, but it may easily cause unseating of the remaining diamonds. To recover lost carbons, a wad of wax or tenacious clay is placed on the end of the drill-rod, and it is gently forced into the hole to its extreme limit, and as gently withdrawn.

The best diamonds are the black amorphous "carbonados" of Brazil, especially those of compact form with well-marked corners. Size may vary between 1 and $3\frac{1}{2}$ carats, according to the bit in use; perhaps the most common is 2 to $2\frac{1}{2}$ carats. Sometimes pieces of corundum, and sapphires which are valueless as gems owing to opacity and bad colour, are coated with graphite and sold as carbons; they accomplish a double fraud, being both heavier (sp. gr. 4 as against 3·5) and less hard.

Drills.—Hand-power drills are made to work by handles on fly-wheels, similar to windlasses, the feed being produced either by weighted levers or a differential screw motion; they can attain a depth of 200 to 400 ft., and give a core 1 to $1\frac{1}{4}$ in. diam. A favourite machine on the Rand is made by A. Short, Durban. An English maker is R. Schram and Co.

Machines for heavy work, driven by steam, compressed air, or electricity, are mainly American—the Bullock, Sullivan, Ingersoll-Serjeant, American Rockdrill, and other companies, turning out a good article.

In deep drilling, it is of great importance to have the core-barrel of sufficient length to avoid frequent lifting as it fills. Height of derrick also influences rate of progress, and should not be less than 50 ft., in order that 40-ft. rods may be unscrewed at a time, this being a maximum convenient length with rods of 2 in. diam. Area of brake surface must be ample, or much delay will be caused by heating.

Electric motors present special advantages for working diamond drills, and have been largely used for that purpose, both at surface and underground. A drill making a 2 in. hole, and bringing up a $1\frac{3}{8}$-in. core, capable of drilling easily to a depth of 600 ft., can be driven by a $2\frac{3}{4}$ h.p. motor, the whole arrangement being compact in the extreme, and suitable for underground or awkward situations where steam could hardly be used. The rapid rotation of the diamond drill adapts it particularly to electric driving. But probably the majority operating in mines are run by compressed air.

Owing to the increasing cost of carbons for boring, the calyx drill is coming much into favour. A contrivance for adjusting the driving mechanism of the diamond drill to suit the calyx cutter, so as to make the one machine interchangeable, and save enormously in first cost of plant, has been invented by E. Williams, Superintendent of diamond drills, and adopted by the Victorian Government. It consists of a simple intermediate gear for reducing the speed in a ratio of 19 to 1, and can be thrown in or out as required.

In the calyx drill, a specially hardened steel shot is employed as the cutting agent. This drill has the advantage of bringing out a larger core than the ordinary diamond drill.

Steel bits are employed at Lake Superior, where masses of copper are likely to be encountered, which would tear the diamonds out of the bit.

In Victoria, a great deal of work has been done with steel bits, at a much slower speed—15 rev. a min., or thereabouts—and under many circumstances they work quite as well as diamonds. They are of no use on hard rock, such as quartzite, or on quartz itself, but on anything up to the hardness of an ordinary tough basalt, they can be used with great effect. The pressure allowed is a little higher than on the ordinary boring gear.

A light and handy diamond drill, actuated by a small petrol engine, would be exceedingly useful in many places where there is neither compressed air nor electricity.

Working Costs.—Official reports on diamond-drilling in New South Wales state the cost at 30*s.* 4*d.* per ft. in 1895, and only 11*s.* 5*d.* in 1896, the difference being due to shallower work and easier ground. The cost of carbons per ft. bored has varied remarkably, thus: 1883, 3*s.* 8*d.*; 1884, 2*s.* 1*d.*; 1885, 1*s.* $5\frac{1}{2}$*d.*; 1886, $8\frac{3}{4}$*d.*; 1887, 1*s.* 7*d.*; 1888, 1*s.*; 1889, 1*s.* 3*d.*; 1890, $7\frac{3}{4}$*d.*; 1891, 1*s.* 10*d.*; 1892, 2*s.* 2*d.*; 1893, 3*s.* $3\frac{3}{4}$*d.*; 1894, 9*d.*; 1895, 3*s.* $9\frac{1}{2}$*d.*; 1896, 2*s.* $1\frac{1}{4}$*d.* The actual working cost per ft. in 1895 for a 4-in. bore 299 ft. deep in porphyry was 15*s.* 2*d.*; the rate of progress was 9·34 in. per hour; and the core obtained was 87·6%.

Recent costs in the basalt-covered leads of Victoria are quoted at 9*s.* 4*d.* to 15*s.* 8*d.* per ft.; and in the softer sedimentary rocks at 1*s.* 7*d.* to 3*s.* 2*d.* per ft. (Lindgren.)

South African figures are quoted by several authors. Denny states the average at 18*s.* per ft., on an assumption of 100 ft. a week, and paying drill hands 20*s.* a day, labourers 2*s.* 6*d.*, fuel at 20*s.* a ton, and carbons at 150*s.* a carat. He says contractors charge 25*s.* per ft. for first 100 ft., rising 5*s.* per ft. for each 50 ft.

Truscott put down 8 holes, of an aggregate depth of 2686 ft., at a cost of 36*s.* 6*d.* per ft. One of these holes, having a depth of 597 ft., averaged 19·9 ft. a day, 1⅝-in. core, and used 8 h.p. motive force and 1440 gal. water daily; the contractor was paid 30*s.* a ft. for 500 ft., and 35*s.* for 97 ft., and the cost of water supply (74*l.* 18*s.* 6*d.*), core watcher (40*l.* 12*s.* 6*d.*), hire of drill (50*l.*), and sundries (29*l.* 5*s.*), was equal to 6*s.* 6*d.* a ft.

The Bezuidenville bore, sunk by Chalmers, occupied 212 days, with an average of 17·58 ft. per diem, external delays accounting for 12 days. For the first 2000 ft., the crown was 2¾ in. and core 1⅞ in. diam., and for final 1728 ft., 2 in. and 1⅜ in. Delays incidental to drilling, repairs, loose carbons, etc., totalled 55 days or 27% on 200 days. On 145 days, straightforward drilling, the rate was 25·7 ft. per diem. The time lost in raising and lowering rods was over ⅓ of the whole. There were used 360 carats of carbons, or between 8 and 9 carats per 100 ft., which, at 80*s.* per carat = 7*s.* per ft.; wages, including overseer, came to 7*s.* 7*d.* per ft.; coal, 200 t., at 20*s.* = 1*s.* 1*d.*; and sundries came to 9*d.*; or a total of 16*s.* 5*d.* per ft., plus interest on 3000*l.* worth of plant.

According to Wybergh, contract prices vary from 22*s.* 6*d.* to 40*s.* a ft., being usually constant for first 1000 ft., and rising 5*s.* per ft. for each 500 ft. Carbons ranging from 7*l.* to 13*l.* per carat. On 14 bore-holes put down by contractors, aggregating 7962 ft., the mean cost was 31*s.* per ft., the range being from 25*s.* 6*d.* to 40*s.*; in addition, water cost nil to 15*s.*, average 5*s.*; superintendence, 6*d.* to 7*s.* 6*d.*, average 2*s.* 6*d.*; and sundries, 4*d.* to 1*s.* 1*d.*, average 9¾*d.*; making the total 28*s.* 3*d.* to 51*s.* 10½*d.*, average 39*s.* 3¾*d.* per ft. The water consumption fluctuated between 1300 and 3200 gal. per diem. The rate of boring was 6·38 to 55·27 ft. per diem, and averaged 16·25 ft. per diem, or ·89 ft. per hour. With contractors, the wear of carbons cannot be ascertained; but in another bore of 1328 ft., in quartzite, somewhat more difficult than the average ground, the consumption was 6·92 carats per 100 ft. In this instance, the detailed cost was: Carbons, 9*s.* 9½*d.*; hire of drill, 3*s.* 1½*d.*; labour, 11*s.* 5½*d.*; coal, 1*s.* 5½*d.*; stores, 11½*d.*; superintendence, 1*s.* 3*d.*; sundries, 6*d.*; total, 28*s.* 6½*d.* per ft. In 3 holes put down by a hand drill, aggregating 318½ ft., through quartzite and diabase, the average rate was 2·03 ft. per diem, or ·309 ft. per hour, and the cost was: Hire of drill and wages of superintendent, 11*s.* 3*d.*; wear of carbons, 10¾*d.*; labour, 4*s.* 10*d.*; sundries, 1½*d.*; total, 17*s.* 1¼*d.* per ft.

West African figures, for holes in sandstone and quartzite 500–1000 ft. deep are 18–19*s.* per ft. (wages, 11–12*s.*; carbons, 3*s.*–5*s.* 6*d.*; fuel, 1*s.* 6*d.*–3*s.* 3*d.*), and $8\frac{1}{4}$–$9\frac{1}{2}$ ft. per diem; and, on bonus conditions, 12–14*s.* per ft. (Justice, Tr. Inst. M.M.)

In the United States, boring for copper ore in the Boundary district (1905) cost an average of about 9*s.* per ft. (carbons, 4*s.* 6*d.*), at a speed of 10 ft. per shift. (Keffer.) On the Mesabi iron-fields, 12*s.* 6*d.* per ft. is paid when in ore, and 25*s.* when in cherty rock (which is very hard on the stones, and allows of only 2·5 ft. per diem). (Woodbridge.) Boring through solid magnesian limestone, in the Missouri lead mines, where the depth ranges between 50 and 500 ft., and sometimes 160 ft. can be drilled in a 10-hour shift, costs between 1*s.* and 8*s.* 4*d.* per ft., the conditions varying so much. (Johnson.)

Costs per ft. of 1850 ft. of $1\frac{3}{4}$-in. hole at Zacatecas, Mex., are stated by Jordan thus: Carbons, at 8*l.* per carat, 1*s.* 10·6*d.*; coal, at 45*s.* per ton, 7·4*d.*; labour (2 drill-runners, 1 carbon-setter, 2 helpers), 3*s.* 2·7*d.*; candles, at 6*d.* per lb., 1·1*d.*; oil, 1·2*d.*; sundries, 2·7*d.*; total, 6*s.* 1·7*d.*

Some further figures quoted by Ihlseng are as follows:—

California, gold mine,	74 to 231	ft. deep,	4*s.* 7*d.* per ft.
Pennsylvania, coal measures,	367	,,	8*s.* 3*d.* ,,
do. do.	412	,,	9*s.* 3*d.* ,,
Lake Superior, iron mines,	400	,,	9*s.* 10*d.* ,,

In British Columbia, where suitable diamonds before the South African War cost about 70*s.* per carat, the contract prices paid for bore-holes ranged between 5*s.* and 6*s.* per ft.; but the supply of stones, even at 15*l.* per carat, is now limited, and over 16*s.* per ft. is being paid.

Precautions in Running.—To obtain useful cores of any friable crystalline or fissile rock or ore, recourse must be had to very large bores (4 in. or so), or the attrition of the bit and the core-barrel will break up the mineral, and perhaps destroy its identity.

When the deposit sought is that of a salt soluble in water, the flushing water must be constantly submitted to tests for its presence, or must be saturated with a similar substance so as to yield solid evidences.

Rocks which fracture obliquely afford wedge-shaped particles which are very apt to interfere seriously with operations, and must be removed by hoisting the rods, if rotating the rods backwards for a few turns and gently restarting does not effect a dislodgment. Extra force must on no account be used.

As diamonds below a certain size, say $\frac{3}{4}$ carat, are virtually useless for drilling purposes, it is true economy to use the largest stones which the drill will accommodate, and thus to reduce the unavoidable loss in rejected fragments to a minimum proportion.

On all occasions, the drill should be started at low speed and gradually hastened. Hence, when actuated by electricity, the motor should be provided with a rheostat.

Fragments of steel from any source are exceedingly destructive in the hole, even when small, and should be assiduously recovered, a magnet being sometimes very useful for the operation.

Clean water is eminently desirable for flushing purposes; where the supply is limited, and repeated use is compulsory, means should be provided for straining out sedimentary matter, and counteracting acidity, if necessary. Nothing must be allowed in the least to impede the water supply to the bit, and it is above all things essential that any choking of the pipes shall be prevented, if possible, and immediately indicated by the pump if it occurs.

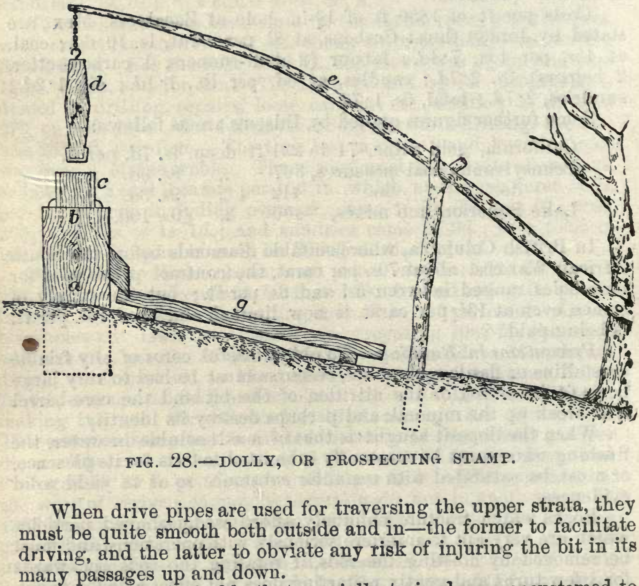


FIG. 28.—DOLLY, OR PROSPECTING STAMP.

When drive pipes are used for traversing the upper strata, they must be quite smooth both outside and in—the former to facilitate driving, and the latter to obviate any risk of injuring the bit in its many passages up and down.

It is highly desirable that specially trained men, accustomed to these machines, should use them whenever possible, rather than entrust the boring to more or less casual labour, even though that labour be most highly trained in other directions.

With regard to the recovery of bits, in case of accident, men who are using the drill daily, and doing nothing else but drilling, are far more expert than those who have not so much to do with it.

They lose fewer bits, and, if they happen to have a loss, they are far more likely to get that bit back, or to recover the stones, than men who are only using the machine periodically.

As to the pressure which may be allowed in a drill, it is governed by the power of the carbons to resist crushing. The heavier the pressure the greater the speed of boring, within limits. Present practice in Europe adopts about 170 lb. per sq. in. of hole area, while in America four or five times as great pressure is used, and even that, it would seem, is a most conservative figure compared with the resistance of the stones.

Prospecting Stamps.—In order to obtain proper samples of the vein-stuff for testing, it is necessary to reduce it to a fine state of subdivision, such as is ordinarily accomplished in the stamp battery.

A primitive yet efficient apparatus for this purpose, and such as may be erected by the prospector himself without the aid of much engineering skill, is shown in Fig. 28, and goes by the name of a "dolly." On the end of a solid log *a*, fixed in the ground and standing about 4 ft. high, is cut a square hole *b*, about 6 in. across, in which are firmly fitted wrought-iron bars about ½ in. apart, ⅝ in. thick and 3 in. deep, made thinner beneath, so that whatever enters above will fall through. A wooden box *c* is placed round this, to keep the ore from jumping away. A square block of wood *d*, about 3 or 4 ft. long, shod with wrought iron, and small enough at the lower end to work in the box, forms the stamper. It is hung on the end of a long pole *e*, the spring of which keeps it on the swing without too much labour. It is worked by laying hold with the hands of a wooden pin *f* on each side of the stamper, and pulling it down, its own rebound and the spring of the pole taking it up again. The iron bars might be replaced by an old stamp die. The gold is caught on the table *g*, which is covered with amalgamated copper plate or blanket.

N

MINE SURVEYING.

MEASURING INACCESSIBLE DISTANCES.

1. In measuring along the line *a b*, Fig. 29, a river intervenes, so that the distance *c b* is inaccessible to direct measurement with a chain. From the point *c*, set off a line *c d* at right angles to the line *a b*, and erect a staff at the point *d*. Then set off a line *d e* at right angles to the imaginary line *d b*, till it cuts the line *a c*. Measure the line *c e*. Then as *c e* is to *c d*, so is *c d* to *b c*. Say *c e* = 60 ft., and *c d* = 90 ft.; then 60 : 90 : : 90 : 135; thus *b c* = 135 ft.

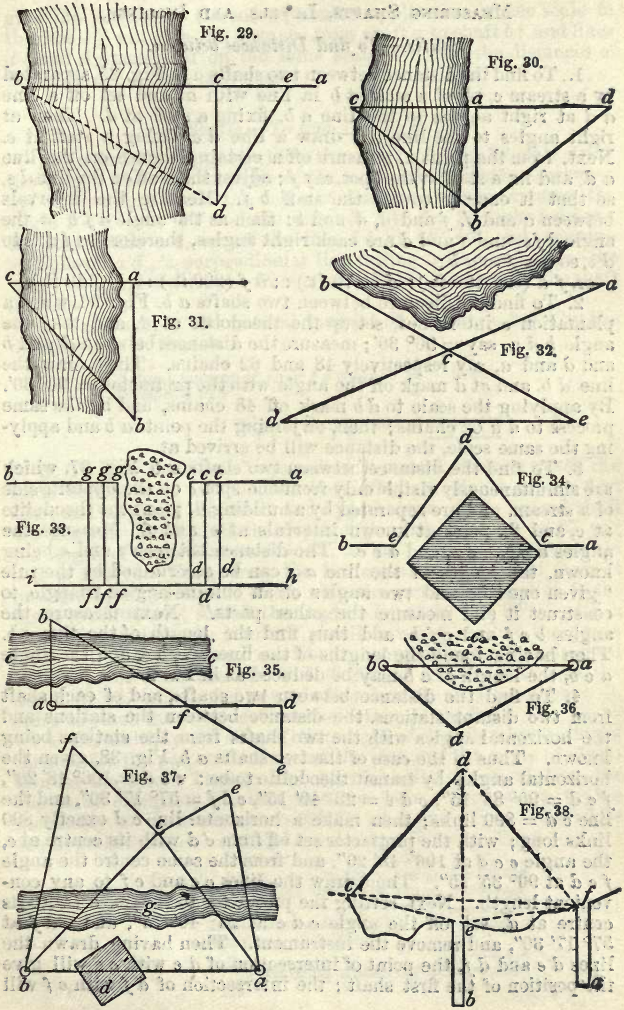
2. To apply the theodolite to the foregoing example, plant the theodolite at *a*, Fig. 30, and at any convenient distance raise a perpendicular *a b*, then remove the theodolite to *b*, and make the angle *a b c* = the angle *a b d*. The distance between *a c* will = the distance between *a d*.

3. Or, set up the theodolite at *a*, Fig. 31, and measure any angle, say 90°; walk along the line *a b* till the theodolite reads half the angle (45°); measure the distance *a b*, which will = the distance *a c*.

4. Or, plant the theodolite at *c*, Fig. 32, and prolong the line *a c* to *d*, and the line *b c* to *e*; measure the lines *a c* and *b c*, making *a c* = *c d* and *b c* = *c e*. The distance *d e* will then = the distance *a b*.

5. On the line *a b*, Fig. 33, an inclosed plantation intervenes. Then at points *c* on line *a b* plant staves, and from each point set off at right angles to the line *a b* lines *d*, and at right angles to these lines *d* set off a new line *h i*, which will pass the obstacle, repeating these operations with lines *f g* in order to regain the original line *a b*.

6. On the line *a b*, Fig. 34, a building interrupts the surveyor. To overcome this by means of a chain and angular instrument, the first step is to measure an angle which will clear the obstacle, viz. *a c d*, at say 140 °. At the point *d* the theodolite is then planted, and a second angle *d e b* is measured off = the supplement of the first, or 70°. Measure along the line *d e* till a point is reached exactly = the length of the line *c d*. Place the transit at *e*, and with the telescope pointing on *d*, get the angle exactly 140° or = the angle *a c d*. Then the true continuation of the line *a b* is found by bringing a staff into the line marked by the cross-hairs of the telescope.



MEASURING INACCESSIBLE DISTANCES.

MEASURING SHAFTS, LEVELS, AND INCLINES.

Shafts.—To find Distances between.

1. To find the distance between two shafts *a b*, Fig. 35, separated by a stream *c*, plant a staff at *b* in line with *a*, and set off a line *a d* at right angles to the line *a b*, fixing a staff at *d*. Then at right angles to the line *a d* draw a line *d e*, fixing a staff at *e*. Next, from the point *a*, measure off a certain distance on the line *a d*, and fix a staff at the spot, say *f*; adjust the staff on the line *d e*, so that it exactly covers the staff *b f*. Measure the intervals between *a* and *f*, *f* and *d*, *d* and *e*; then as the angle *a f b* = the angle *d f e*, and *a* and *d* are each right angles, therefore as *f d* is to *d e*, so is *a f* to *a b*;
or, *f d* (say 100 ft.): *d e* (70 ft.) : : *a f* (200 ft.) : *a b* (140 ft.).

2. To find the distance between two shafts *a b*, Fig. 36, when a plantation *c* intervenes, set up the theodolite at *d*, and take the angle *b d a*, say = 90° 30′; measure the distances between *d* and *b* and *d* and *a*, say respectively 48 and 62 chains. Then draw the line *d b*, and at *d* mark off the angle with the protractor = 90° 30′. By applying the scale to *d b* mark off 48 chains, and by the same process to *d a* 62 chains; then, on joining the points *a b* and applying the same scale, the distance will be arrived at.

3. To find the distance between two shafts *a b*, Fig. 37, which are simultaneously visible only from one spot *c* on the opposite side of a stream, and are separated by a building *d*, plant the theodolite at *c*, and fix poles at known intervals at *e* and *f*. Measure the angles *a c b*, *a c e*, and *a e c*. The distance between *c* and *e* being known, the length of the line *a c* can be ascertained by the rule "given one side and two angles of an oblique-angled triangle, to construct it and measure the other parts." Next measure the angles *b c f* and *c f b*, and thus find the length of the line *c b*. Then having found the lengths of the lines *a c*, *b c*, and the angle *a c b*, the length of *a b* may be deduced as in *Ex.* 2.

4. To find the distance between two shafts, and of each shaft from two distant stations, the distance between the stations and the horizontal angles with the two shafts from the stations being known. Thus in the case of the two shafts *a b*, Fig. 38, given the horizontal angles by transit theodolite to be: *e c d* = 106° 48′ 20″, *f c d* = 90° 35′ 15″, *c d e* = 23° 40′ 15″, *c d f* = 57° 17′ 30″, and the line *c d* = 890 links; then make a horizontal line *c d* exactly 890 links long; with the protractor set off from *c d* with its centre at *c*, the angle *e c d* at 106° 48′ 20″, and from the same centre the angle *f c d* at 90° 35′ 15″. Then draw the lines *c e* and *c f* to any convenient length. Next, setting the protractor to the line *c d* with its centre at *d*, set off the angle *c d e* at 23° 40′ 15″, and *c d f* at 57° 17′ 30″, and remove the instrument. Then having drawn the lines *d e* and *d f*, the point of intersection of *d e* with *c e* will give the position of the first shaft; the intersection of *d f* with *c f* will

give the position of the second shaft; application of the scale to the line $e\,f$ will give the distance from shaft a to shaft b; and lines $d\,e$ and $d\,f$ measured on the same scale will give the distances of the stations $c\,d$ from each shaft.

Shafts.—To find Depths for.

1. To find the depth at which the shaft $a\,b$, Fig. 39, will strike the vein c, when the underlie of the vein and the distance between the outcropping d and the top a of shaft are known, assume the angle $b\,d\,a = 44° 40'$ and the distance a to $d = 90$ links; draw a horizontal line $a\,d$, set off the line $d\,b$ at $44° 40'$, and measure 90 links on $a\,d$; a perpendicular line falling from a will strike the vein c at b, and the length from a to b will be the depth of the shaft in links.

2. Or, multiply the natural tangent of the angle $b\,d\,a$ by the distance a to d, thus—

Natural tangent of angle $b\,d\,a$ $44° 40'$	=	·988432
Distance a to d	=	90
Depth of shaft in links	=	88·958880

3. To find the depth required in a second shaft to reach the same bottom level as the first shaft, when the depth of the first shaft and angle of elevation and hypothenuse of the second shaft are known. In Fig. 40, the depth of the first shaft $a\,b = 87$ yd., the angle of elevation $c\,a\,d = 32° 25' 30''$, and the hypothenuse $a\,d = 220$ yd. Then draw the horizontal line $e\,f$, and from it measure up 87 yd. from b to a; with the protractor applied to the line $e\,f$, centre at f, set off the angle $e\,b\,c$ at $32° 25' 30''$; with a parallel rule draw the line $a\,d$ parallel to $b\,c$, so that the angle $c\,a\,d$ will = the angle $e\,b\,c$. Then a perpendicular line dropped from a point d on the line $a\,d$ at 220 yd. from a, will give the depth in yd. which the new shaft $g\,h$ sunk at d must have in order to reach the same level as the bottom of shaft $a\,b$.

4. Or, multiply the sine of the angle of elevation by the hypothenuse for the perpendicular, and the cosine of the angle of elevation for the base. Thus in Fig. 41, the shaft $a\,b$ is 80 yd. deep, the angles of elevation (in this instance being two, owing to the inequality of the surface) are $39° 31'$ at a, and $15° 11'$ at c, while the hypothenuse $a\,e = 240$ yd., and $c\,d = 160$ yd. Then—

Angle a $39° 31'$ = sine ·636303 × 240	=	152·712720
Angle c $15° 11'$ = sine ·261909 × 160	=	41·905440
Total difference of level $d\,e$	=	194·61816
Add depth of shaft $a\,b$	=	80
Total depth of shaft $d\,f$	=	274·61816

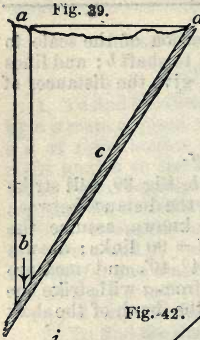


Fig. 39.

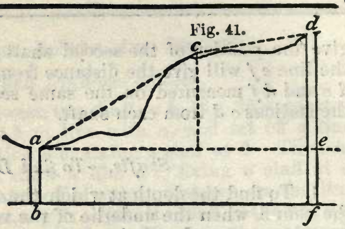


Fig. 41.

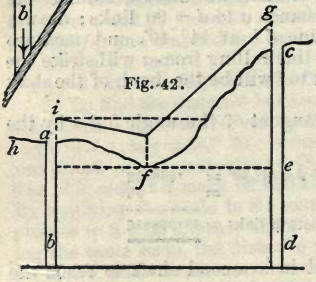


Fig. 42.

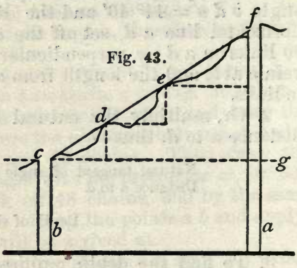


Fig. 43.

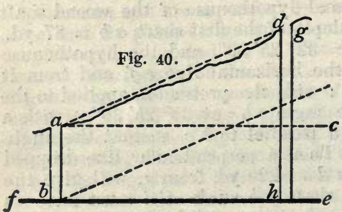


Fig. 40.

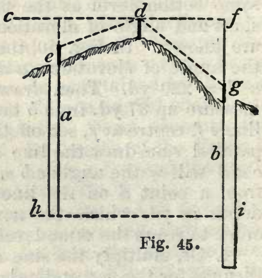


Fig. 45.

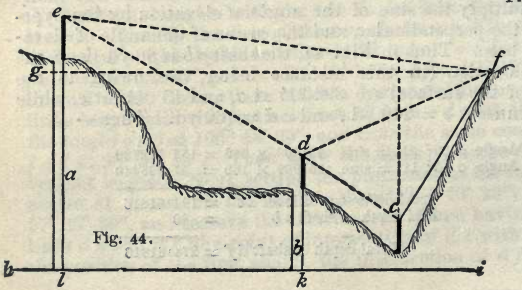


Fig. 44.

SHAFTS.

Shafts.—To find differences of Level of.

1. At top. The angles of elevation between the two shafts $a\,b$ and $c\,d$, Fig. 42, and the hypothenuse being known, their difference of level at the mouth is found in the following way. Suppose the angle of elevation $e\,f\,g = 44° 20'$, and the angle $h\,f\,i = 10° 35'$, while the hypothenuse $g\,f = 870$ yd., and the hypothenuse $i\,f = 630$ yd. Then draw the horizontal line $h\,e$, set off the above angles from f, and mark the distances named on $g\,f$ and $i\,f$. The difference of level between the intersection of the telescope of the theodolite at i, and a staff of similar height at g will give the difference of elevation between the mouths of $a\,b$ and $c\,d$; or it may be seen from the scale.

2. Or, supposing the angles of elevation in Fig. 43, between shafts a and b on the line $c\,g$, to be from $c = 42° 10'$, from $d = 41° 40'$, from $e = 43° 25'$, and hypothenuse $c\,d = 80$ yd., $d\,e = 85$ yd., $e\,f = 120$ yd., then multiply the sines of the angles by the hypothenuses, thus—

Angle from	c 42° 10′ = sine ·671290	×	80	=	53·7032
"	d 41° 40′ = sine ·664796	×	85	=	56·50766
"	e 43° 25′ = sine ·687299	×	120	=	82·47588
	Total difference of level in yd.			=	192·68674

3. Or, when the angles of elevation or depression from two distant stations are known. Suppose two shafts $a\,b$, Fig. 44, and that the angle from station c to point $d = 45° 10'$, that from c to $e = 44°$, and that from c to $f = 60°$, and the line $c\,f = 1000$ links. Take at f the angle of depression f to d, or $g\,f\,d = 30° 15' 22''$, and the angle of elevation f to e, or $g\,f\,e = 15° 9' 11''$. Draw the horizontal line $h\,i$, and from point c set off angle $h\,c\,d = 45° 10'$, angle $h\,c\,e = 44°$, and angle $i\,c\,f = 60°$. From point c draw the lines $c\,d$, $c\,e$, $c\,f$, making the length of $c\,f = 1000$ links, and $f\,g$ parallel to $h\,i$. At point f, set off the angles $g\,f\,d = 30° 15' 22''$, and $g\,f\,e = 15° 9' 11''$, and draw the lines $f\,d$, $f\,e$. Then the point where $f\,d$ intersects $c\,d$ will be the top of shaft b, and the point where $f\,e$ intersects $c\,e$ will be the top of shaft a. Applying the same scale as before to the lines $d\,k$ and $c\,l$, their difference will = the difference in elevation of the mouths of the two shafts.

4. At bottom. Suppose the depth of shaft a, Fig. 45, = 60 yd., and of shaft $b = 55$ yd., while the angle of depression $f\,d\,g = 43° 10' 15''$, angle $c\,d\,e = 21° 2' 10''$, the distance $g\,d = 500$ yd., and the distance $e\,d = 400$ yd. Then draw the horizontal line $c\,d\,f$, and with the protractor set off from d the angle $f\,d\,g$ at 43° 10′ 15″, and the angle $c\,d\,e$ at 21° 2′ 10″, and draw the line $g\,d = 500$ yd., and the line $e\,d = 400$ yd. The points $e\,g$ are thus the tops of the shafts $a\,b$, and perpendicular lines dropped from the horizontal line $c\,d\,f$ to the points $e\,g$ will give, when measured on the scale, the

differences in elevation at the mouths of the shafts. Then prolong these perpendicular lines and measure on a 60 yd., and on b 55 yd. in depth, and draw the horizontal line $h\ i$ from the bottom of shaft a, so that it intersects shaft b. The difference between the horizontal line $h\ i$ and the bottom of shaft i on the scale is the measure sought.

Shafts.—Surveying Deep.

The difficulties of surveying are not markedly increased by depth, except in the case of vertical shafts, when it becomes necessary to carry down an azimuth from the surface by means of two plumb lines hung in the shaft. It is almost impossible to free the lines entirely from disturbing influences, which displace them from their normal positions. If either lines or bobs are of magnetic material, the presence of iron pipes in the shaft may seriously disturb them. Falling water may be in such quantity, and so directed, as also to affect the position of the lines. But air currents, which cannot be wholly eliminated whatever the precautions taken, are the most serious cause of disturbance. The temperature at the bottom of the shaft is higher than that at the top, and in consequence convection currents are formed. The heat supplied from the surrounding rock keeps them up. Their effect on a plumb line may remain sensibly constant for hours at a time while deflecting the line from its vertical position. The stability of the currents is made possible by large cross section of the shafts giving a large air body, and by the constancy of the rock temperatures, and supply of heat through the shaft walls. When we take account of the fact that a force equivalent to a horizontal pressure of 10 gr. on a 60-lb. plumb bob suspended by a line 4000 ft. long will displace the bob over 1 in. from its normal position, it is easy to see how apparently slight causes may produce appreciable error in azimuth. While divergence alone would not affect azimuth, the question is whether the divergence may not be due to some cause which will also displace one or both of the lines in a direction perpendicular to their plane. The idea that the excess of attraction (gravitation) exerted on each bob horizontally by the mass of the wall towards which it hangs nearest is wrong. While existing qualitatively, its amount is insignificant—it does not account for more than ·001 ft. at the Tamarack No. 5, Lake Superior, while the convergence of vertical lines in that instance is more than 3 times as much. (McNair, E. & M. Jl.)

Levels.—To find Length required.

1. When the depth of the shaft, its distance from the vein, and the dip of the vein are known.

Suppose the shaft a, Fig. 46, is 50 yd. deep and 100 links distant from the outcrop b of the vein c, which dips at 40°. Connect the outcrop b with the top of shaft a by drawing the line d = 100 links, and from the line $b\,d$ at b, set off the angle $g\,b\,d = 40°$. A perpendicular line from d, 50 yd. deep, will represent the shaft. From the bottom of the shaft draw the horizontal line $e\,f$ parallel to $d\,b$ and measure it by the scale; the number at the point of intersection between $e\,f$ and $b\,g$ = length of level in links.

2. Or suppose the depth $c\,d$ of the shaft a, Fig. 47, = 50 yd., and the angle $c\,d\,e = 44°\ 10'$, being the dip of the vein b, then the

Natural tangent of $c\,d\,e$ 44° 10′	=	·971326
Multiplied by the depth $c\,d$	=	50
Length of level $c\,f$ in yd.	=	48·566300

3. Or, suppose the depth of shaft a, Fig. 48, = 50 yd., the distance $d\,e$ between the shaft a and outcrop of vein b = 120 yd., and the angle $d\,e\,f$ or dip of vein = 40°, then the

Natural tangent of angle $d\,e\,f$ 40°	=	·839100
Multiplied by distance $d\,e$	=	120
Distance $d\,f$	=	100·692000
Less distance $d\,g$	=	50
Then distance $g\,f$, in yd.	=	50·692000

The angle at $f = 90° -$ angle at e (which is, say 47° 10′), or f = 42° 50′, then the

Natural tangent of e 42° 50′	=	·927091
Multiplied by distance $g\,f$	=	50·5
Length of level c, in yd.	=	46·8280955

Levels.—To find Depth at which known Length shall strike the Vein.

1. Suppose the depth of shaft a, Fig. 46, = 50 yd., the distance from shaft a to outcrop b of vein c = 100 links, the dip of vein c = 40°, and the length of level $e\,f$ = 20 yd. Draw the horizontal line $d\,b$ 100 links long, and at b set off the angle 40°; drop a perpendicular line $d\,e$, and apply a parallel ruler to the line $d\,b$, running it down till the level $e\,f$ exactly fills the space between $d\,e$ and $b\,g$. This spot will indicate where the level will touch the shaft $d\,e$,

and the depth may be ascertained by applying the scale to the shaft *d e*.

12. When the shaft is continued below the vein and a returned heading is made. Suppose the shaft *a*, Fig. 49, to be distant 120 yd. from the outcrop *e* of the vein *b*, the angle of dip of vein *b* to be 47° 12′, and that the shaft *a* is continued in depth for 80 yd. below the point where it intersects the vein *b*. Draw a horizontal line *d e*, and from the point *e* set off with the protractor the angle 47° 12′; then a perpendicular line *d f* falling from *d* will intersect

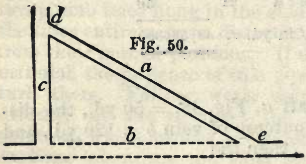


Fig. 50.

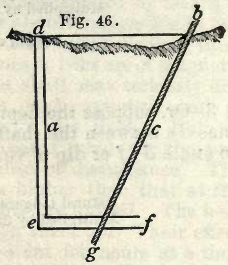


Fig. 46.

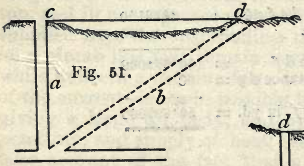


Fig. 51.

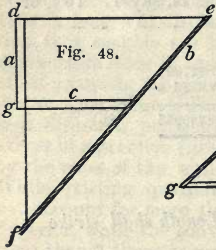


Fig. 48.

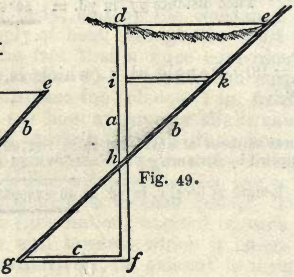


Fig. 49.

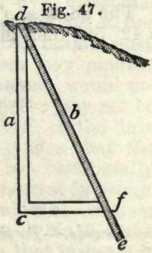


Fig. 47.

LEVELS AND INCLINES.

the vein *e g* at *h* (and by measuring on the scale will give the depth of shaft *d f* to that point), and extended 80 yd. deeper to *f* will indicate where the level is to be run; apply the parallel ruler to *d e*, run it down to point *f*, and apply the scale to line *g f*.

Levels.—To find depth of Shaft at Intersection of Vein, and Lengths of Levels, when one level is above and one below the intersection.

Let the distance $d\,e$, Fig. 49, = 120 yd., the angle of dip = 47° 12′, the depth of shaft a from d to first level $i\,k$ = 30 yd., and the depth from h to f = 80 yd.; then the

Natural tangent of angle at e 47° 12′	=	1·079902
Multiplied by distance $d\,e$	=	120
		21598040
		1079902
Depth at intersection, in yd.	=	129·588240
Less depth $d\,i$	=	30
Depth $i\,h$, in yd.	=	99·588240

The angle at h = 90° − 47° 12′ = 42° 48′; then the

Natural tangent of angle at h 42° 48′	=	·926010
Multiplied by distance $i\,h$	=	99·5
Length of level $i\,k$, in yd.	=	92·1379950

Also the

Natural tangent of angle at h 42° 48′	=	·926010
Multiplied by distance $h\,f$	=	80
Length of level $g\,f$, in yd.	=	74·080800

Levels.—To find Direction and Length when required to strike a fixed point at right angles.

Thus, in Fig. 50, it is required from the bottom of incline a to drive a level b so as to cut the shaft c at right angles. Suppose incline a to be 60 yd. long and shaft c to be 30 yd. deep, then divide the side c opposite the required angle of b by the hypothenuse, and the quotient will be the sine of the angle. Thus,

60) 300 (·5 = sine 30°
300

The angle at $d = 90° - 30° = 60°$. Then the

Natural cosine of angle at e 30°	=	·866025
Multiplied by a	=	60
Length of level b, in yd.	=	51·961500

Or, the

Natural tangent d 60°	=	1·732051
Multiplied by c	=	30
Length of level b, in yd.	=	51·961530

Inclines.—To find Direction and Length.

In Fig. 51, suppose the depth of shaft $a = 80$ yd., the distance $c\,d = 60$ yd., then find the length and angle of incline b so as to strike the surface at d. Divide the side a opposite the required angle on the other side $c\,d$; the quotient will be the natural tangent of the angle. Thus:—

$$
\begin{array}{l}
80\,)\,6000\,(\,\cdot 75 = \text{tangent } 36°\ 52' \\
560 \\
\hline
400
\end{array}
$$

The angle at $d = 90° - 36°\ 52' = 53°\ 8'$. Then the

Natural secant of 53° 8′	=	1·666792
Multiplied by $c\,d$	=	60
Length of incline b, in yd.	=	100·007520

Latching or Dialling. (Merrett.)

"Latching" or "dialling" is a term used by miners when an underground survey is required of the course that has been excavated from one shaft of the mine to another. This survey is then transferred to the estate plan, by the guidance of the position of the shafts, which are accurately shown on both surveys. By reference to Fig. 52, which is an actual survey, it will be seen that the lengths are taken by the chain or tape in links, and the angles are taken by the needle or magnetic meridian with a circumferator, called by miners a "dial." The legs of this instrument are made with a screw joint in the middle. It has an extra set of points to screw on when the mine workings are low-roofed. The survey commences at shaft No. 16, and is carried to

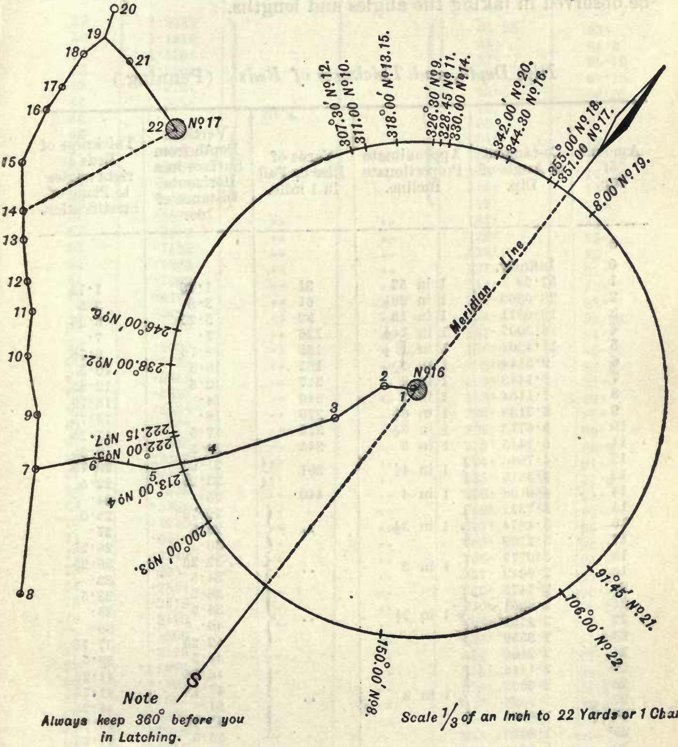


FIG. 52.—LATCHING OR DIALLING.

shaft No. 17. The chain lines and angles are all plotted from that point. The circle shows the protractor with the meridian line through the centre, and all the angles marked thereon for plotting. The "field-book" as it were, is on the margin, containing only the numbers, lengths, and angles. Great accuracy must be observed in taking the angles and lengths.

Dip, Depth, and Thickness of Beds. (Penning.)

Amount of Dip.	Co-tangent of Angle of Dip.	Approximate Proportionate Incline.	Yards of Rise or Fall in 1 mile.	Vertical Depth from Surface in a Horizontal Distance of 100.	Thickness of Beds at right angles to Plane of Stratification.
0	Infinity.				
1	57·29	1 in 57	31	1·75	1·75
2	28·6363	1 in 28½	61	3·5	3·5
3	19·0811	1 in 19	92	5·25	5·25
4	14·3007	1 in 14½	126	7·	7·
5	11·4301	1 in 11½	155	8·75	8·75
6	9·5144	1 in 9½	185	10·5	10·5
7	8·1443	1 in 8	217	12·5	12·25
8	7·1154	1 in 7	249	14·	13·75
9	6·3138	1 in 6½	279	16·	15·75
10	5·6713	1 in 5½	314	17·5	17·25
11	5·1445	1 in 5	345	19·5	19·25
12	4·7046	1 in 4½	391	21·25	20·75
13	4·3315			23·25	22·5
14	4·0108	1 in 4	440	25·	24·
15	3·7321			26·75	25·5
16	3·4874	1 in 3½	..	28·5	27·
17	3·2709			30·	28·25
18	3·0777	1 in 3	..	32·25	30·25
19	2·9042			34·5	32·
20	2·7475			36·5	33·5
21	2·6051			38·5	35·
22	2·4751	1 in 2½	..	40·	36·
23	2·3559			42·25	37·75
24	2·2460			44·5	39·5
25	2·1445			46·75	41·25
26	2·0503	1 in 2	..	47·75	41·75
27	1·9626			51·	44·25
28	1·8807			53·25	45·75
29	1·8040			55·5	47·25
30	1·7321			58·	49·
31	1·6643			60·25	50·5
32	1·6003			62·5	51·75
33	1·5399			65·	53·25
34	1·4826	1 in 1½	..	67 5	54·75
35	1·4281			70·	56·25
36	1·3764			72·5	57·75
37	1·3270			75·25	59·5
38	1·2799			78·	61·

Amount of Dip.	Co-tangent of Angle of Dip.	Approximate Proportionate Incline.	Yards of Rise or Fall in 1 mile.	Vertical Depth from Surface in a Horizontal Distance of 100.	Thickness of Beds at right angles to Plane of Stratification.
°					
39	1·2349			81·25	63·
40	1·1918			84·	64·5
41	1·1504			87·	65·75
42	1·1106			90·	67·75
43	1·0724			93·	69·25
44	1·0355			96·	70·25
45	1·	1 in 1	..	100·	71·
46	·9657			104·	72·
47	·9325			107·	73·25
48	·9004			111·	74·25
49	·8693			115·	75·5
50	·8391			119·	77·
51	·8098	..	..	123·	78·
52	·7813	..	..	128·	79·
53	·7536	..	..	133·	80·
54	·7265	..	..	137·	81·
55	·7002	..	..	143·	82·
56	·6745	..	..	150·	83·
57	·6494	..	..	154·	83·25
58	·6249	..	..	161·	84·5
59	·6009	..	..	166·	85·5
60	·5774	..	..	172·5	86·5
61	·5543	..	..	180·	87·
62	·5317	..	..	188·	88·
63	·5095	..	..	200·	89·
64	·4877	..	..	205·	90·
65	·4663	..	..	213·	90·5
66	·4452	..	..	224·	91·
67	·4245	..	..	235·	91·5
68	·4040	..	..	250·	92·
69	·3839	..	..	260·	93·
70	·3640	..	..	275·	94·
71	·3443	..	..	300·	94·5
72	·3249	..	..	308·	95·
73	·3057	..	..	327·	95·5
74	·2867	..	..	345·	96·
75	·2679	..	..	370·	96·5
76	·2493	..	..	400·	97·
77	·2309	..	..	433·	97·5
78	·2126	..	..	476·	97·5
79	·1944	..	..	515·	98·
80	·1763	..	..	570·	98·5
81	·1584	..	..	633·	98·5
82	·1405	..	..	714·	99·
83	·1228	..	..	813·	99·
84	·1051	..	..	1000·	99·5
85	·0875	..	..	1140·	99·5
86	·0699	..	..	1430·	99·5
87	·0523	..	..	1912·	100·
88	·0349	..	..	2865·	100·
89	·0175	..	..	5714·	100·
90	·0000	..	..	—	100·

Dip, Depth, and Thickness of Beds.

Ex. 1.—The outcrop, in a level district, of a definite bed is observed in a quarry, where it is found to dip E. at an angle of 14°. This bed would be found, if its dip remains unaltered, 100 yd. east from the quarry at a depth of 25 yd. = 75 ft.; the beds above it would have an actual thickness of 24 yd. = 72 ft.

Ex. 2.—The outcrop of the same bed with the same dip is observed in a district where the surface rises in the direction of the dip at an angle of 10°. The bed would be found, 100 yd. E. at a depth of 75 ft., as before, plus the rise in the ground between, which in 100 yd., at the rate of 314 yd. in a mile, is $17\frac{3}{8}$ yd. = 53 ft.: 75 + 53 = 128 ft. If the ground were falling instead of rising, at an angle of 10° in the direction of the dip, the bed would be met with at 75 − 53 = 22 ft.

Ex. 3.—A bed or formation is known to be 37 yd. in thickness, and to dip at an angle of 14°. In 100 yd. of outcrop it would, at that dip, have a thickness of 24 yd. only, therefore its outcrop is (24 : 36 : : 100) : 150 yd.

Ex. 4.—The thickness is known to be 36 yd. as before, and the level outcrop 150 yd. In 100 yd. ($\frac{2}{3}$ of 150), the thickness would be 24 yd., which corresponds to a dip of 14°.

Ex. 5.—The dip of the bed is 14°, its outcrop 150 yd.; in 100 yd. at 14° the thickness would be 24 yd., therefore in 150 it is 36 yd.

Contents of Veins.

A vein of ore 1 in. thick, 6 ft. long, and 6 ft. high (= 1 sq. fathom) will measure 3 cub. ft.; 2 in. thick = 6 cub. ft. per fathom; and so on.

Weights of Ores 1 in. thick per sq. fathom (i.e. per 3 cubic ft.).

	lb.		lb.
Antimony, grey oxide	843	Iron, magnetic	1016
Barytes	750	„ pyrites	912
Cobalt, pyrites	937	„ specular	912
Copper, carbonate (malachite)	712	Lead, carbonate	1200
„ grey	900	„ sulphide (galena)	1406
„ native	1668	Nickel, arsenical	1406
„ pyrites	787	Silver, native	1875
„ red	1106	Tin, oxide	1256
„ vitreous	1050	„ pyrites	825
Gold, native	3281	Uranium, oxide (pitchblende)	1312
Iron, arsenical	1068	Zinc, red oxide	1012
„ hæmatite	750	„ sulphide (blende)	750

The above figures × 9 = weights per cub. yd.

Sampling and Charting.

The systematic measuring, sampling, and assaying of mineral veins, lodes, reefs, ledges or beds, as they are variously called, and the charting of the figures obtained on working plans of the mine, are necessary operations both in examining and reporting on properties, and in recording the results of operations at going concerns. In the former case, the inspecting engineer has to do the work himself; in the latter, the duty falls upon a member of the mine staff—the underground manager, the surveyor, the consulting engineer, or a special officer, according to the scale of operations. By constant and systematic sampling, measuring and charting of all development work, it is possible to gauge and adjust the output, both in quantity and quality for considerable periods in advance, and thus to provide for contingencies, conferring stability and regularity upon the enterprise.

In every drive, sink and rise on ore, the thickness and value of the ore-body should be ascertained and recorded at regular intervals; and where several parallel ore-bodies exist, or where "horses" of barren ground occur in the vein, these should be similarly measured, valued, and charted. The length of the intervals must depend on the degree of uniformity presented by the ore-body; where regularity in width and value is well marked, 10-ft. or even 20-ft. stages may suffice; but on veins showing much fluctuation, 5 ft. (or, as some prefer, the almost obsolete fathom—6 ft.) gives closer results. Indeed, there are some mines where measuring and sampling even at less intervals than 5 ft. fail to give the least approximation, and in such cases it is labour in vain; but they are exceptional. In addition to sampling all development work, daily samples should be taken from every stope, particularly to ensure that the whole width of the pay-stone is being removed, and with special attention to any subsidiary or separate stringers, leaders, droppers or branches.

Sampling cannot be conducted in a perfunctory manner; on the contrary, to be of any service at all, it must be reliable within narrow limits, and demands extreme care and precision. Every lode varies incessantly in hardness and in contents, and inasmuch as virtually all ores are softer or more brittle than the gangue, there will always be a tendency to break an unduly large proportion of the more valuable material into the sample. Apart from deliberate "salting" or fraudulent enrichment, to be specially guarded against by personal or other reliable supervision, and by dressing (and, if necessary, washing) down the face before breaking the sample, there is always the risk of involuntary exaggeration of the value of standing ground by neglecting to break the proper amount of hard resisting barren rock. Knowing the difficulty of securing a truly representative sample, many engineers are in the habit of discounting by 5 or 10 % the results of their assays, as a sort of compensation. Where the working exposes ore in one or both walls, as well as overhead or underfoot, the sample should be drawn from all portions, but particular care is necessary in sampling floors, because on them will always lie a certain amount of "fines" from mining operations, and these fines are sure to be above the average value.

In soft ground, a small prospectors' pick (a pole pick about 9 in. long, with a 12-in. handle) suffices; but mostly the ground is hard enough to require a moil or gad, and a 3- to 7-lb. hammer. Samples may vary in bulk according to circumstances, about 1 to 7 lb. being the usual weight; they should bear some direct proportion to the width of the ore-body (say 1 lb. per ft.), and to the size of barren lumps, such as the pebbles in a conglomerate. They are placed separately in stout canvas bags, about 14 × 10 in., previously numbered in sequence with bold black stencilled figures, and with a strong tier attached at about 3 in. from the top. The number on the bag will be recorded against the spot at which the sample is taken. When emptying the bag, it must be turned inside-out, and thoroughly cleansed by brushing, to remove any loose free gold (which is often spongy, and very adherent to textile fabrics), or fine-grained pyritic matter. In a mine producing very high-grade ore, it is not safe to use the same bags repeatedly and indiscriminately, even when every care is taken in emptying them. As a sort of tray for catching the sample while it is being chipped off, the most handy thing is a piece of rubber cloth (such as explosives are wrapped in), it having a non-retentive surface; failing that, a sou'-wester hat reversed is good. Neither a shovel nor a miners' pan is at all satisfactory, because it cannot accommodate itself to the inequalities of the face.

Under some circumstances, it is necessary to provide against tampering with samples, and each bag is sealed. Also, before assay, each sample is well washed in hot water to dissolve any gold chloride which may have been injected through the bags, and the filtered wash-water may be tested for gold. In many cases it is well worth while, after crushing the sample to pass 100 mesh, to make separate estimations of free amalgamable gold, percentage and value of concentrates (by panning), and contents of tailings, from these two processes.

The figures obtained by the sampler, plus those afforded by the assays when completed, are at once tabulated in a "sample book," which should embody the following information:—

Date.

Sample No.

Locality, e.g. drive, winze, etc., —— ft. north, east, etc., from —— shaft or survey datum.

Dimensions of vein (in ft. and decimals), and any remarks necessary, e.g. splitting, faulting, etc.

Assay No.
Value (in oz. or dwt. fine metal per ton, or in % in some cases).

From these data the "size-value" at each sample point is computed by multiplying ft. into oz., dwt., or %; thus 1 ft. × 12 dwt. = 12 ft.-dwt. (size-value); or 2·5 ft. × 50·25 oz. = 125·625 ft.-oz.; or 3·2 ft. × ·175% = 5·6 ft.-%. Then the mining value of any particular length or block of ore-body can be arrived at very quickly; the sum of the dimensions recorded for the distance sought to be valued, divided by the number of samples, gives the average dimensions; and the sum of the "size-values," divided by the sum of the dimensions, gives the average assay value. This assumes that the several dimensions and assay values recorded are correct for half the intervening space between them in each case. The actual tonnage and contents may be simply calculated when the weight per cub. ft. is known. If the ore-body is of such a size or character that it cannot be broken down free from country or other barren rock, the amount of extra ground so broken must be added to mining cost estimate; if this can be in part picked out afterwards, the proportion (weight) and cost must be considered; and if some unavoidably remains, the milling value must be proportionately lowered.

All these essential facts are worked out first in the sample book, and from there are charted on the working plans of the mine, or on a special tracing from them. It is well to use different-coloured inks or different numerals for dimensions and assays, so as to avoid confusion.

Where an abnormally high assay occurs in a series, which may easily be due to a chance particle of free gold getting into the sample, it is safest to substitute for it the average of the two samples next it on each side; but the risk of such exaggerated assays happening is reduced by first removing and estimating (from a comparatively large quantity) the coarse free metal.

It is obvious that the varying conditions of metalliferous mining will demand many modifications in the system of sampling and recording; also, of course, very much more information may be charted if desired, such as speed of driving, ratio per man per shift, cost, etc., etc., until the whole details of the undertaking are embraced.

Besides deductions for dykes and horses of barren ground, irrecoverable pillars must not be forgotten.

In calculations of weight from bulk, tables such as that on p. 146 are often used; but as a rule every ore-body is more or less complex, or possesses some peculiarity, which makes an actual trial measurement the only safe guide. The amount of moisture carried by the product must be carefully ascertained, as it may easily affect results by 10%.

While certain veins and masses possess sufficient constancy in size and value to give a fairly reliable basis for calculations of potential yield, there are many more in which such estimates of future returns are the merest guesswork. In the valuation of mines on behalf of purchasers, the only safeguard, in the author's opinion, is to secure a 6 months' working option on the property and to carefully note developments. In nine cases out of ten, the price paid, when this simple precaution is neglected, is ridiculously excessive.

No method of calculation will get over the inaccuracies liable to occur through (*a*) spacing samples too far apart; (*b*) sampling excessive or too small widths; (*c*) incorrect assaying; (*d*) failure to make allowance for freaks in structure of an ore-body; (*e*) mis-calculating waste-rock sorted out; consequently, the correctness and value of such an estimate must largely depend on the practical experience and judgment of the sampler, and the only conclusive proof of value is by actual treatment, controlled by careful assays of ore, pulp, and tailings, on accurately ascertained quantities of ore.

While special features characterise the Rand banket deposits, making it possible to introduce computing systems, relative to their extent and value, which come within the limits of "practical accuracy," that "practical accuracy" is more due to this fact, and to the compensation of cancelling errors over a long series of calculation, than to accuracy in the system of computation. Hence the results obtained in the mill are always lower than the "average" of the samples obtained in the mine, though the "average" taken is obtained after eliminating all high values. When values which are above the general average of the samples are brought down to the general average of the samples, even then the mill results do not bring out this diminished average result by 7% to 10%.

The usual calculation is thus made:—

100 tons	assaying 12 dwt. (corrected average) contain	1200 dwt.
15 ,,	waste rock assaying 1 dwt. contain	15 ,,
85 ,,	sorted ore contain	1185 ,,

One ton has a value after sorting of 13·9 dwt., and, on a basis of 90% extraction, a recovery value of 52*s.* 6*d.* Waste rock, sorted out after being in contact with banket, generally has an average value of about 1 dwt. fine gold per ton.

Instead of this recovery of 52*s.* 6*d.*, only about 42*s.*, on a basis of 15% sorting and 90% recovery, is saved in actual practice. The reason for this discrepancy is that no allowance has been made for the waste of actual mining. In measuring the width of a stope, it is usual to measure the actual face, the measurement being taken at right angles to the dip of the reef. The actual face between footwall and hanging-wall may in this way be 3 ft. If a measurement be made 3 or 4 ft. back from the face, between footwall and hanging, also at right angles to the dip of the reef, it will be found in the majority of cases (if the bed has remained uniform—not been cut out by a fault, and not pinched), that such a measurement will be greater by about 12 in. than the measurement at the face. The dilution caused by these 12 in. which have not been allowed for will account for the difference in nearly all cases. In other words, actual blasting on a 3 ft. lode, or attempting to carry a 3 ft. stope, will generally remove 6 in. more in the hanging-wall and 6 in. more in the footwall than the actual calculated width. (Way.)

The Trans. Inst. M.M. contain many recent papers on surveying, sampling, charting and valuing mines.

DRILLING.

THE term "drilling" is often used without discrimination both for that operation in which the object sought is a solid rock core and for that which aims only at making a hole (for insertion of explosive) and ignoring the material through which the hole is driven. Confusion is avoided by applying the word "boring" to the former operation (see p. 158), and confining the expression "drilling" to the latter process. The definition "churn-drilling" is sometimes employed, indicating that the churning or comminution of the rock is involved; but this is not sufficiently distinctive, because a certain amount of hole-sinking is done with rotary drills. Moreover, it would seem not improbable that the future will witness an extension of the principle, if suitable metal for the cutting edge can be secured, because the economy of making a hole say $1\frac{1}{2}$ in. diam. by cutting a narrow ring say $\frac{1}{4}$ in. wide or less, instead of pounding to powder the solid $1\frac{1}{2}$ in. of rock, is self-evident.

Meantime rotary drilling is almost confined to soft ground, such as is encountered at limited depths, and in coal, ironstone and similar formations. Here its superiority is manifested. But in hard ground, the percussive or churn drill is almost universal, and it is to it especially that this article is devoted. It may be hand-driven, or it may be actuated by pneumatic, electric, or other power.

Bits.—The drilling tool proper, however propelled, consists of a steel rod or "bit," shaped at one end for cutting, and at the other end it is either left blunt for receiving hammer blows, or is taper-rounded for insertion in the chuck of the machine.

The width of the bit varies, according to requirements, from 1 to $2\frac{1}{2}$ in. The stock is octagonal in section, and is made in lengths varying from 20 to 42 in. The shorter the stock, the more effectively does it transmit the blow, and therefore it is made as short as possible: for this reason several lengths are employed in drilling a blast-hole, the shortest being used at the commencement of the hole, a longer one to continue the depth, and a still longer one, sometimes, to complete it. To ensure the longer drills working freely in the hole, the width of the bit should be very slightly reduced in each length. The diameter of the stock is less than the width of the bit; this difference may be greater in coal drills than in rock or "stone" drills; a common difference in the latter is $\frac{3}{8}$ in. for the smaller sizes, and $\frac{1}{2}$ to $\frac{3}{4}$ in. for the longer. The following proportions may be taken as the average adopted:—

Width of the bit.	Diameter of the Stock.	Width of the Bit.	Diameter of the Stock.
1 in.	$\frac{5}{8}$ in.	$1\frac{3}{4}$ in.	$1\frac{1}{8}$ in.
$1\frac{1}{8}$ „	$\frac{3}{4}$ „	2 „	$1\frac{3}{8}$ „
$1\frac{1}{4}$ „	$\frac{7}{8}$ „	$2\frac{1}{4}$ „	$1\frac{1}{2}$ „
$1\frac{1}{2}$ „	1 „	$2\frac{1}{2}$ „	$1\frac{5}{8}$ „

The simplest and most common form of cutting edge is the plain chisel bit; this may be made quite straight or slightly curved; the straight edge cuts its way somewhat more freely than the curved; but it is weaker at the corners than the curved, a circumstance which renders it less suitable for very hard rock; it is also slightly more difficult to forge. Other forms resemble the letters Z and X, the latter particularly, called the "cross" or "star" bit, being much in favour for the first stages of a hole. Thus, in a 6-ft. hole, the first 12 in. may be drilled by a 3-in. star, the next 15 in. by a $2\frac{1}{4}$-in. star, the following 18 in. by a $1\frac{5}{8}$-in. chisel, and the final 27 in. by a $1\frac{3}{8}$-in. chisel. In such a case, the chisel bits would be made of $1\frac{1}{8}$-in. round or octagonal steel; the star bits of cruciform section steel about $1\frac{1}{8}$ in. diam., with a short round shank turned or forged on to fit the machine chuck. Star bits may be sharpened as easily as chisels by using proper swages.

Another pattern in great favour among the copper miners of Montana and Idaho somewhat resembles $\overset{\vee}{\wedge}$.

A number of experimental designs have been tried and discarded.

In hand-drilling, the wear of steel caused by the blows of the hammer is just about as great as that due to cutting the rock, so that the economy of metal in machine-drilling is very marked.

A set of single coal-blasting gear will include a drill, 22 in. long, with cutting edge straight and $1\frac{1}{2}$ in. wide, and weight $2\frac{1}{2}$ lb.; another drill, 42 in. long, with straight cutting edge $1\frac{7}{16}$ in. wide, weight 4 lb. 10 oz.; the hammer weighs 2 lb. 14 oz.; length of head $4\frac{1}{2}$ in., and that of handle $7\frac{3}{4}$ in. A single-hand stone set includes shorter drill, 22 in. long, cutting edge strongly curved, and $1\frac{1}{2}$ in. wide, and weight 3 lb. 10 oz.; longer drill, 36 in. long, cutting edge $1\frac{7}{16}$ in. wide, and curved as in the shorter drill, and weight 6 lb. 5 oz.; hammer weighs 3 lb. 6 oz.; length of head 5 in., and that of handle 10 in. A double-hand stone set comprises first or shortest drill, 18 in. long, $1\frac{3}{4}$ in. wide on the cutting edge, and weighs $4\frac{1}{4}$ lb.; second drill, 27 in. long, $1\frac{11}{16}$ in. wide on the cutting edge, and weighs 6 lb.; third, or longest drill, 40 in. long, $1\frac{5}{8}$ in. wide on the cutting edge, and weighs $9\frac{1}{4}$ lb.; the cutting edges of all these drills are strongly curved Sledge weighs about 5 lb.

The approximate average consumption of steel at Lucknow, N.S.W., in 5 years, under the author's management, was $1\frac{1}{2}$ lb. per ft. of sinking and driving, and 1 lb. per ton stoped, with machine drills.

Sharpening.—In quite easy ground, one bit will drill, without sharpening, from a maximum of 4 ft. to less than 1 ft., and a fair ordinary average would be 1½ ft.; in harder ground, it would not do more than 8–9 in.; and in very hard ground, 2–3 in. would be good work.

There are many improvements on the common open forge, such as the Blakney furnace, the Bradley forge (for coke), the Ajax forge (for crude oil), and the electric furnace (which may be used below ground).

To temper a bit correctly, a tank or trough should be used, constructed on the following lines. The cold-water inlet must be in the bottom, and the hot-water outflow near the top at one end, the supply being continuous and regular. A perforated iron plate is fastened in the tank in such position that there is always about ¾ in. of water flowing over it; and a rack is built around the tank—big nails driven into a board at about 3 in. apart—to hold the drills upright while cooling. Each drill as finished from the fire is stood with the cutting edge in the water until cold. Tests with this system gave 15% more depth per minute. (T. H. Proske.)

Power-driven drill-sharpeners have been widely introduced on the Rand, but not always retained after trial, as some involve cutting off and wasting about 2 in. of steel at each heating; they are designed both for chisel and for star bits.

The Kimber machine has recently been tried at the Village Main Reef; two of them are officially said to have saved 200*l.* a month over hand-sharpening.

The Ajax machine is popular in America. At the Centre Star, Rossland, 1 machine, with 3 men, sharpens 500–550 bits per 10 hr., and "saves 40% in the labour cost." Its capacity is about 1200 bits in 24 hr., the time for each bit ranging from ½ to 1 minute. Bits sharpened in this way are more regular and better than when hand-sharpened, and will put down more holes. At the United Verde and the Homestake mines, one Ajax machine in each case now does the work formerly requiring 12 men, and the air consumed to sharpen 600 drills in 10 hr. is about ⅕ that needed to run one 3-in. drill underground. The United Verde mines sharpen 500 bits in 7 hr., and the Homestake mines 300 in 4 hr. The Homestake Co.'s sharpening costs by hand and machine respectively are thus given:—

Hand sharpening: 10 smiths at 14*s.* 7*d.*, 7*l.* 5*s.* 10*d.*; 10 helpers at 12*s.* 6*d.*, 6*l.* 5*s.*; 1200 lb. coal, 1*l.* 10*s.*; total, 15*l.* 0*s.* 10*d.*

Average per smith and helper, 120 drills per 10-hr. shift.

Machine sharpening: 4 smiths at 14*s.* 7*d.*, 2*l.* 18*s.* 4*d.*; 4 helpers at 12*s.* 6*d.*, 2*l.* 10*s.*; 720 lb. coke, 19*s.* 10*d.*; compressed air, 8*s.* 4*d.*; fire-brick for repairs, 10*d.*; total, 6*l.* 17*s.* 4*d.* Output 1000 bits, 2 10-hr. shifts.

Cost per 100 bits: hand, 25*s.* 1*d.*; machine, 11*s.* 9*d.*

The Ajax machine costs about 250*l.*; coke forge, 10*l.*; oil forge, 30*l.*

Power Drills.—The "hand-hammer" drill is a development of the familiar pneumatic riveting hammer that has long been in successful use in machine shop practice. From this was evolved the plug drill, which found extensive employment in drilling small shallow holes for plug and feather work in quarrying, block-holing of boulders, large rocks, etc. Now drills of this type are designed for regular mining work. They are of light weight (only 15–20 lb.), and of simple construction. The working parts are few and easily kept in order. These are the chief factors in the usefulness of the machine. Incidentally the "helper" is dispensed with. The air consumption is comparatively large (approximately 20 cub. ft. per min.), and the maximum depth of hole is only about 3 ft. In drilling holes of 2 ft. depth in a fairly hard quartz the speed is about 1 in. per min. Considering that no time is lost in changing position, hand-hammer drills are found to compare favourably with the work of a 2¼-in. machine drill on short holes. Their chief objection appears to be their inadaptability to all classes of rock, the latter presenting difficulties when either very dry or very wet and soft. These difficulties are connected with the hole in the bit, through which the air is exhausted. In dry rock, the dust blown out of the hole becomes such a nuisance as to preclude the use of the drill, in some cases.

On the other hand, in damp, soft ground the hole in the bit becomes clogged so tightly after a few minutes' work that further operation is impossible; the only way to clean the bit in this contingency is to send it to the shop, and have the dirt drilled out. This appears to be a more serious difficulty than the dust trouble, inasmuch as the latter can be minimised by the provision of a suitable dust catcher, and possibly may be overcome entirely by the application of some form of water feed. In other kinds of ground, these drills have given successful results. They would appear to be at their best in compact, fine-grained, moderately hard rock; and especially in dry rock of that character, if the dust difficulty can be obviated. If for nothing else, they will be valuable for block-holing and breaking boulders, cutting hitches, and squaring up. (E. & M. Jl.)

There is a good deal of prejudice among miners against these machines, on account of both the dust created, and the fatigue of holding them to the work. In back holes, they must be intolerable.

Two patterns (the "little wonder" and the "little Jap") have been much used at the Davis pyrites mine, Mass., especially for block-holing and hitch-cutting, and at one time the former kept the stopes going for 2 weeks by drilling 4-ft. holes. But both gave much trouble at times through the hammer sticking, and they refused to cut the hard wall rocks, besides working ill in wet ground. Generally, the greatest source of trouble, otherwise, was the bending or breaking of the hollow bits at the point where the steel shank or point is welded or brazed to the stay-bolt iron constituting the body of the drill. This difficulty was lessened through the employment of a solid bit of hexagonal steel, with a ⅝-in. hole drilled lengthwise through it. These bits will not bend, do not require tempering of shank or points, can be sharpened like the steel used in the large drills, and do not require careful handling. Their use is a decided improvement over the soldered or brazed bits. Although the dust from these drills is annoying, their use results in a great saving over hand drills for block-holing and trimming ore on foot and hanging walls. (J. J. Rutledge, E. & M. Jl.)

The standard types of air- or steam-driven rock drills may be broadly divided into two classes, according to valve gear, viz.: (*a*) the tappet type, and (*b*) the fluid moved valve type. There is a third arrangement, which may be termed the valveless type, in which the moving piston acts as its own valve.

Table of Drills. (A. H. Smith.)

Cylinder Diameter.	Stroke.	Feed.	Depth of Hole Drilled Easily.	Diameter of Hole.	Weight of Machine Unmounted.	Consumption of Air at 80 lb. Pressure.	Boiler Power.	Class of Work best suited to the Machine.
in.	in.	in.	ft.	in.	lb.	cub. ft.	h.p.	
2	4½	12	4	¾ to 1½	88	35	5	Plug and feather, small shallow holes, light work.
2½	5	20	6	1¼ to 2	128	47	8	Ditto small tunnels, slate, narrow veins.
3	6½	27	12	1¼ to 3	190	81	8	Mining, tunnels, shafts.
3¼	7	27	16	1¼ to 3	238	92	9	Standard size for general hard mining work.
3½	7½	28	20	1¾ to 3	280	105	10½	Large open quarries, big tunnels and railway cutting.
3½×3¾	7½	30	30 to 40	2 to 4½	300	120	12	Special quarrying drill for deep holes, with large diameter front cylinder for lifting the bit in deep hole boring.
3⅝	7½	30	20 to 25	1¾ to 4	360	115	12	Heavy tunnelling, heavy railway work, etc.
4¼	8	30	28	2 to 5	360	160	15	Heavy quarry work in hard granite, etc.
5	8½	36	30 to 40	3 to 6	720	195	15	Submarine work.

Some machines will do fair work in soft rock at 45 lb.; but in most mines, 65–90 lb. are chiefly used, the tendency being to increase these pressures in hard ground.

Three types of drill recently introduced, which are a marked departure from usual practice, may here be mentioned—one is the "hand-power" machine, another is the "hammer-blow," and a third is the "rotary."

In so-called "hand-power" drills (Jackson type), the blow is given by the recoil of a compressed spring. The interior mechanism consists of a ram (with collars), cam-axle, cams, and power spring, together with a rifle-bar and ratchet controlled by pawls and springs to effect rotation. It is operated by means of a hand crank and fly-wheels. The drill delivers $3\frac{1}{2}$ strokes for each rev. of the crank, or 175–200 per min. at ordinary speed, each blow being about 50 lb., according to the strength of the spring. Length of stroke and tension of spring may be varied. The machine weighs about 140 lb.

In the "hammer-blow" drill (Leyner type), the borer bit is entirely disconnected from the piston, the steel being struck by the ram at each stroke. Provision for a water jet is made, a needle-valve at the ratchet end of the machine admitting water to a $\frac{1}{4}$-in. steel tube, which conveys it into the shank of the hollow borer (hole $\frac{5}{16}$–$\frac{3}{8}$ in. diam.), allowing it to be discharged at the cutting point. The 3-in. cylinder drill has a stroke of 3 in., and 24 in. feed. It is only adapted for use by compressed air, weighs 155 lb. unmounted, and consumes about 75 cub. ft. of free air per min. at 80 lb. It is a modern application of a very old idea, but is not attractive from a mechanical efficiency point of view. The water jet for clearing the cutting edge of the bit from accumulated sludge is, however, excellent. Among the disadvantages connected with the use of the water Leyner are the complicated nature of the machine, the necessity for the employment of a competent skilled runner, the greater skill necessary to sharpen the drills, the necessity for a uniformly high air pressure in order to obtain the best results, and the frequent breakage of chucks.

The rotary drill (Brandt type) has done good work in a special case, but the arrangement in its present form is unsuitable for ordinary conditions of mining. The costly installation (including a filter to free the feed water from grit), high pressure, amount of water used and necessity for pumping it out again, weight of machine and mountings, and difficulty of keeping it up to the working faces and protecting it, are all against it.

None of these innovations appears at all likely to become a serious rival to the two main types of drill actuated respectively by compressed air and by electricity.

But another pattern of water drill (the Gordon), lately extensively tried at the Robinson mine, Rand, seems promising. The whole outfit weighs 115 lb. (drill, 55 lb.; column and bar, 60 lb.).

There are no nuts or screws, wedges being used to attach the drill to the arm. It is claimed the drill can be easily worked by Kaffirs. The results at the Robinson mine show that 3 drills, employing 4 Kaffirs, are doing the work of 25 Kaffirs, and doing it better. The experiment has been carried out on a reef under 24 in. wide, the stopes being kept at 30 in., instead of the 48 in. necessary with larger machines in the same stope; this means that the grade of the ore is considerably increased, as only 6 in. of waste is mined and hauled, as against 24 in. with the ordinary drill. These machines require only 25 % of the air required to run a 3¼-in. drill, and the average footage per shift is 14·12 ft., reckoned on 3968 ft. drilled. (M. Jl., Jan. '07.)

Air-driven Drills.

Plant.—An air-driven drill equipment, with its approximate cost, may be as follows:—

Drills.

Size	2½ in.	2¾ in.	3 in.	3¼ in.	3½ in.	4 in. diam. of cylinder
Price	30*l.*	35*l.*	40*l.*	40*l.*	45*l.*	50*l.*

Accessories.

Stretcher bar and steel clamp, 8*l.* to 16*l.*
Tripod and weights, 13*l.* to 15*l.*
50-ft. length of hose with brass fittings complete, 7*l.* to 8*l.*
Finished steel tools, best quality, in assorted lengths and sizes, per cwt., 4*l.* 10*s.*
1 set of swages for sharpening tools, 1*l.* 10*s.*
Tunnelling carriage, complete, for 4 drills, 100*l.*
" " " 2 " 80*l.*

2½-in. and 3-in. diameter drills are best suited for ordinary quarry work where tripods are employed.

3¼-in. and 3½-in. diameter drills are commonly used for mine purposes.

4-in. diameter drills for heavy tunnelling work, to bore holes of large diameter.

A complete mining plant (e.g. 6 drills) would consist of the following items, viz.:—

6 3½-in. drills, complete.
6 stretcher bars with steel clamps.
6 50-ft. lengths of covered hose with all necessary brass connections.
6 sets of best finished steel tools, in assorted lengths and sizes.
1 set of swages for sharpening tools.
1 complement of spare wearing parts.

1 18-in. direct-acting air-compressor, with steam cylinder complete.
1 air-receiver, fitted with safety valve, blow-off cock, etc.
1 30-h.p. nom. boiler, complete with donkey pump.
Connections between boiler and steam cylinders and between air cylinders and receiver.
Wrought-iron air piping for conveying the compressed air to the machines.

A complete plant as above, for 6 drills, 3 in. or 3½ in., would cost approximately 1300*l.* to 1400*l.*

A complete plant as above for 4 drills, 3 in. or 3½ in., would cost approximately 800*l.* to 900*l.*

A small plant of say 2 3-in. drills, with 10-in. air compressor and 8 to 10 h.p. boiler, would cost about 500*l.* to 600*l.*

Air.—The amount of air at receiver pressure required per drill per minute when working at normal speed is about:—

Size of drill	2½ in.	3 in.	3¼ in.	3½ in.	4 in.
Cubic ft.	8·5	12	14	16·5	21·5

For the purpose of calculating the size of compressor, it is, however, well to assume a somewhat larger amount of air per drill than given above, as often, owing to loss due to clearance, heat, and leakage, the actual amount of air discharged from an ordinary compressor with poppet valves is 25 per cent. less than the theoretical amount.

Safe figures are:—

Size of drill	2½ in.	3 in.	3¼ in.	3½ in.	4 in.
Cubic ft.	10	15	17	20	25

With the best compressors, the loss of efficiency ought not to be more than 10 per cent. It is false economy to employ too small a compressor. Where a number of drills have to be driven, an allowance might be made, which could not be done in the case of a compressor to drive only two machines.

An effective air pressure of 50 to 75 lb. per sq. in. is usually employed for working drills. The lower pressures are used for drills employed on soft rock, and for machines which do not work expansively, using the full air pressure throughout the stroke. There is no economy in working with air at too low a pressure. The only limit to the pressure is the point when the tool gets too quickly blunted; the cost of the men to run the drill is a considerably greater item than that of compressing the air.

Approximately the volume of compressed air is equal to the original volume divided by the number of atmospheres (absolute) to which it has been compressed.

Ex. 1.—Required the size of compressor to supply air at 60 lb. effective pressure to work two 3-in. drills.

Air required per minute:—

$15 \times 2 = 30$ cub. ft. at 60 lb.,
or, $30 \times 5 = 150$ „ of free air at atmospheric pressure would be required per minute to work both drills simultaneously.

Taking the piston speed at 300 ft. per minute:—

$$\text{Area of cylinder} \times 300 = 150$$
$$\text{,,} \quad \text{,,} \quad = \frac{150}{300} = 72 \text{ sq. in.,}$$

which represents a cylinder diameter of $9\frac{5}{8}$ in., to which the nearest sized compressor would be 10 in. diameter.

A smaller plant than a 10-in. compressor and two drills is very seldom employed, and would not be economical.

Ex. 2.—Required the size of compressor to supply air at 60 lb. effective pressure to work six $3\frac{1}{2}$-in. drills.

Air required per minute:—

$20 \times 6 = 120$ cub. ft. at 60 lb. pressure,
or, $120 \times 5 = 600$ „ of free air at atmospheric pressure.

Assuming a piston speed of 350 ft. per minute:—

$$\text{Area of cylinder} \times 350 = 600$$
$$\text{,,} \quad \text{,,} \quad = \frac{600}{350} \text{ sq. ft.} = 247 \text{ sq. in.,}$$

which represents a cylinder of $17\frac{3}{4}$ in. diameter, the nearest sized compressor to which would be an 18-in. It would be possible to drive the 6 drills with a 16-in. compressor running a little faster, and taking into consideration that seldom more than five drills would be working at one time; on the other hand it would be more economical, and better in the long run, to employ an 18-in. compressor.

As the density of the atmosphere decreases with the height above the sea-level, allowance must be made for this when calculating a compressor plant to work at high altitude. Taking the efficiency at the sea-level to be 100, the decrease in efficiency is approximately 3 per cent. for every 1000 ft. above the sea-level.

In air mains it is fair to assume a velocity of about 25 to 30 ft. per second, so that for an 18-in. compressor delivering 120 cub. ft. per minute, the air main should be $\frac{120}{60 \times 30} \times 144 = 9 \cdot 6$ sq. in., or say $3\frac{1}{2}$ in. diam.

Machines.—In all the makes of machine drill now in established use, the work is done by compressed air, and the air is distributed by a valve either (*a*) moved by air pressure, the class being known as "pressure-valve" machines, or (*b*) thrown by some positive movement directly transferred from the piston, and termed "tappet-valve" machines. The former are preferred as being less liable to breakage and less likely to prematurely admit a reverse air current and thus "cushion" the blow. The forward feed motion is effected by a screw, and in a drill of about 3 in. diam., its length is about 24 in. The rotation of the bit is caused by the movement of a rifle-nut, forming part of the rear end of the piston, along a rifle-bar which extends into the piston through the rifle-nut; this rifle-bar is held by a ratchet and pawl, so that it rotates the drill during the backward stroke, and is itself made to rotate during the forward stroke; thus the drill always turns in one direction, and only on the back stroke.

According to expert opinion derived from observations with drills in workings of restricted size, a smaller and lighter type of machine using air up to 100 lb. per sq. in., instead of the maximum 70 to 80 lb. (often 40 to 50 lb.) now accepted, and with more handy holding contrivances, must be sought. Speed is much more important than weight of striking parts, because the effectiveness of the blow varies as the square of its velocity but only directly as its mass; and in slower movements there is increased risk of cushioning. The waste of air is enormous, but would be much reduced if it could be used expansively. Re-heating at the drill does not seem practicable, unless possibly by electric means, such as passing a current through a wire coil of very low conductivity in a special receiver close to the drilling machine, as suggested by Prof. Louis.

In operation, the drill is rigged on a vertical column, on a stretcher or horizontal bar, or on a tripod, according to the work. Columns are used in ordinary headings or drives, and in sinking; they are made of stout iron tube about 4½ in. diam. and 6 ft. long, or according to the height of the heading, and have at one end a cap-piece of larger diameter, which is pressed against a block of wood on the roof, by screw-jacks beneath. On this column is an adjustable arm for carrying the machine. Stretcher bars sometimes accommodate two machines at a time in large drives. Tripods are only resorted to when columns or bars cannot be fixed.

There are many well-known makers of machine drills both in this country and in America. Each claims some superiority for his own article, but in practical long-continued use there is no marked advantage in any one over the others. In all, the excessive weight, 245 to 290 lb., is a great drawback to their more extended application, though various attempts have been made to produce a smaller stoping machine of 150 to 175 lb. Even this, however, is too clumsy for single-handed manipulation. As already mentioned, speed is a matter of prime moment, and in this respect the Climax drill made in Cornwall is well to the front. The makers of this have also recently introduced a one-man stoping drill weighing all-told less than 100 lb., which, with air at really good pressure, to atone for the small cylinder (2 in. diam.), does excellent work and supplies an urgent need.

In machines of the established type, much of the energy is expended at each stroke in forcing the cuttings from the hole, and in doing useless work upon that portion of the cuttings which is not immediately expelled. The Leyner drill aims at remedying this by pneumatically forcing a jet of water to the end of the hole through a channel in the drill, as above mentioned.

The Hardy Patent Pick Co. turns out a stoping drill weighing only 112 lb., striking 600 blows per min., and drilling holes 9 ft. deep, at a cost of 25*l.*

Electrically-driven Drills.

The actuation of drills by electricity has long been under experimental trial, and a large measure of success has been attained, especially in those of the rotary class—twist-drills capable of doing satisfactory duty in easy ground, such as coal measures, ironstone beds and salt deposits. A variety of these machines is in use. One (General Electric Co.) drilling in coal makes 300 rev. a minute, consumes 2 h.p., and advances 1 ft. in 30 to 40 seconds; another (Jeffery) drills 6 ft. a minute, and is arranged to work at 220, 350, or 500 volts. A favourite machine (Steavenson's) in the Cleveland iron mines consists of a carriage supporting a standard capable of being turned in any direction, and carrying an arm, on one end of which is a small direct-current motor, whilst the other end holds a twist-drill driven by means of bent gearing, the connection between the drill and its driving shaft being made by a Hooke joint, so as to allow the drill perfect freedom of motion; it works at 220 volts, requires 20 amperes, or say 6 h.p. at the drill, makes 100 rev. per minute, and drills in ironstone at an average rate of 3 to 4 ft. per minute; in a working shift of 10 hours, each drill will put in 100 holes, averaging 4 to 5 ft. deep, in about 8 different working places. A Siemens-Halske drill, worked from a separate motor by means of a flexible shaft, uses 1 to $1\frac{1}{2}$ h.p., makes 200 rev. a minute, and occupies 1 to 2 min. per ft., at the Stassfurt potash-salt mines.

Electrically-driven percussive drills gain a reciprocating movement from solenoids, from a coiled spring, or from some form of crank.

The Bladray, which has been given several trials on the Rand, weighs 225 lb., though the frame is made of aluminium-bronze (90% Al, 10% Cu), which must render it costly; it strikes 700 blows of 3 in. and 400 lb. per min., and uses 2 i.h.p. Run on sandstone, it cut 7 ft. in 13 min., making 720 2-in. × 300 lb. blows, and required 70 volts at 19 amperes, or $1\frac{3}{4}$ elect. h.p.; no trouble was encountered in clearing sludge. In the hardest "blue" rock at depth, it attained 3 in. per min. The Bladray stoping drill of 130 lb. is claimed to make 900 blows of 120 to 200 lb. per min., using $1\frac{1}{2}$ i.h.p.

The Gardner, a Colorado drill, is receiving wide application in the United States, and is well spoken of. Its motor (working at 110 volts) is very thoroughly protected against wet and dust, and every part of the drill seems to have been designed on the simplest and strongest lines, so that it may withstand harsh usage, and wearing parts may be easily renewed. It is very handy, and quite within the compass of one man. The bit-holder, while effective, permits rapid exchange of drills. In speed and power, it closely resembles the Bladray. Drill and motor complete cost about 125*l*. In a 5 ft. by 8 ft. tunnel, in medium gneiss, at the Edgar mine, Idaho Springs, it averages 4 ft. per 8-hour shift; and the cost has been reduced to 15*s*. 3*d*. per lin. ft., as against 33*s*. 4*d*. by hand, while the speed has been tripled.

A German electrically-driven percussive drill, described by E. Dane (Tr. I.M.M. Feb. '02), used in Celebes, weighs 240 lb., delivers strokes up to 400 per min., and bores holes 16 in. deep, without changing bits; 6 drills were driven by a 12-h.p. direct-current dynamo at 220 volts, actuated by petroleum engine, and the installation, complete with ½ ton steel bits, cost in Europe 1460*l*., plus 200*l*. for petroleum engine. After 18 months' use, it was still good for another 8 to 12 months. Average power used was 4 amperes at 220 volts, or about 1⅛ b.h.p. per drill. Petroleum consumed, ¾ to 1 pint per b.h.p. per hour. Work done, 4 ft. per h.p., i.e. on basis of 35 ft. in 7 hours by 1¼ h.p.; as against 2·5 ft. per h.p. by air drills on usual rate of 120 ft. in 5 hr. by 10 h.p. at steam engine. Over 2000 ft. of driving, sinking, and rising was done by native attendants only. It often ran 2 to 3 weeks in ordinary ground without repair. The cost of driving a main 7 × 6 ft. tunnel in hard diorite with 2 machines side by side was, per ft., drillers, 4*s*. 7*d*.; shovelling and trucking, 4*s*. 7*d*.; petroleum power, 6*s*. 9*d*.; smith and coal, 1*s*. 3*d*.; steel, 2*s*.; mechanics' time repairing drills, 12*s*. 4*d*.; spares and breakages, 6*s*.; total, 37*s*. 6*d*.

The Union drill (a solenoid machine) used at Gangesberg, Sweden, 1899, drilled 578 holes, aggregate 4969 ft., average 8·59 ft., mean rate 22·47 ft. per 8-hr. shift, or 2·56 ft. per hr. working; as against air drill—1094 holes, aggregate 4243 ft., average 3·88 ft., mean rate 19·94 ft. per 8-hr. shift, or 2·49 ft. per hr. working. Another, at Bindt, Austria, 1895, advanced drive 2009 ft., bored 17,795 holes, totalling 42,705 ft., average 2·4 ft., in 13,301 hr. 54 min. = ·64 in. per min., costing per ft. of drive—drillers, 3*s*. 1*d*.; blasters, 1*s*.; carmen, 1*s*. 10*d*.; mechanics, 1*s*. 9*d*.; smithy, 1*s*. 3*d*.; repairs to machines, 7*d*.; steel, 2*d*.; total, 9*s*. 8*d*. The power consumed is from 4½ h.p. per drill with 2 to 4 drills down to 3 h.p. with installations of more than 6. At Altenwald, Prussia, the cost of repairs for spring drills was 1*s*. 6*d*. per ft. driven, as against 5*d*. for solenoids. (Dr. Simon, Tr. I.M.M., '02.)

Under certain conditions, electric drills may work more speedily and with greater economy of power than air drills; but facts that bear strongly against the electric type are their unhandiness (owing to excessive weight), and their increased liability to breakage (owing to more delicate mechanism).

The so-called electric-air drill is simply an ordinary air drill equipped with a small compressor attached to an electric motor, the combined driving mechanism being mounted on a travelling carriage.

Drilling Operations.

Placing Holes.—In drilling holes for blasting, it is necessary to use great judgment in determining their number, depth, relative position, direction and order of firing. The main object to be kept in view is that each hole shall perform its proper share of the work. Holes drilled too deep or too far apart may "blow out" and break nothing; too shallow holes leave "shoulders" or "toes" that necessitate a supplementary one; holes drilled too close together mean a waste of labour and of explosive.

The principal factors are the hardness or "tightness" of the ground, and the presence or absence of bedding-planes or lines of fracture caused by earth movements. The tighter the ground, the less "load" can be placed on a hole. Where a natural cleavage line exists, the "cut" is almost always drilled towards it, the only exception, perhaps, being where the shot would "run," and endanger the security of other ground; failing a seam or joint to cut to, it must be made artificially by placing the first holes converging towards each other, so as to remove a wedge or cone of rock on blasting. In a face of limited size and approximately square shape, 3 holes set like an inverted tripod may suffice; in the "sink" of a 3-compartment shaft, two converging rows of 3 or more holes each may be necessary.

Next in importance to the character of the ground is the system of work. With machine drills, it is essential to economy that they shall be able to put down a maximum number of holes at each rigging, because the greater the proportion of the men's time occupied in mounting and dismounting the machine, the less the work done for the wages paid. In hand-drilling, therefore, a more thorough regard for accurate placing is possible. Regularity of operations, and absence of any hindrance to the work of drilling as each successive shift comes on, must be provided for, whether the labour be on contract or on daily wages; thus the sequence of drilling, firing, and clearing away the shot ground becomes a matter of some moment, so that the shovellers, who are always wages men, shall neither be partially idle, nor cause delay to the drill-hands.

The first set of holes are called the "cut" or "sink"; the next around them (when necessary) are "easers"; while the outside row comprise the top or "back" holes or "headers," the side holes or "skimmers," and the bottom holes or "lifters." Each group is generally fired simultaneously, or nearly so, and renders effective the blasting of the next lot in succession. (See BLASTING.)

Speed and Cost.—It is to be observed at the outset that the assumption that increased speed means decreased cost is not safe; sometimes the gross cost may be lessened by increased speed, because the proportion of fixed general charges is reduced, but as a rule, greater speed means enhanced net cost. Local conditions, governing cost of power on the one hand, and cost and quality of labour on the other, cut a large figure in the calculation.

The relative value of hand- and power-drilling depends almost entirely on the hardness of the rock.

In the soft ground of the Staffordshire iron mines, the Kimberley diamond mines, and very many other situations, a churn driller, or a man wielding a "jumper"—a long heavy double-ended drill without a hammer—can sink a hole while a machine is being rigged. It is computed that a pair of men can accomplish 8 to 12 down holes, 2½ ft. deep and 2 in. diam., in a day, at a cost of 6*d.* to 9*d.* per ft., when the driller receives 10*s.* a day and his mate 4*s.* In similar ground, at horizontal and rising ("upper") holes, a single-hand hammerman, turning his own drill, is the cheapest. In somewhat harder ground, a long-handled hammer is necessary, with a mate to turn the drill. This is the most usual arrangement, the men taking turns in holding and striking, though sometimes economy is sought by putting two hammers on the drill. It is commonly reckoned that a pair of men can drill 25 to 33 ft. of 1-in. hole per diem in limestone, and 4 to 8 ft. in granite; and that with double hammers, 10 ft. of 2-in. hole can be driven in medium rock.

In many cases, machine drills have been discarded after trial against hand labour. At Mesquital, Mexico, the conditions of soft ground and untrained labour placed the machine-drills out of the question; native miners in the stopes earn 1 dol. (2*s.*) a day, and labourers about 70c. (1*s.* 5*d.*); drives or headings are contracted at 3½ to 6 dol. (7 to 12*s.*) per ft., and shaft sinking at 14 dol. (28*s.*) and upwards; the average rate of driving is 3½ to 4 ft. a week (3 shifts per 24 hours). In Norway, power drilling is twice as costly as handwork, but three times as speedy,

On the Rand, machine-drill driving ranges from 50 to 150 ft. a month, and averages about 80 or 90 ft.; hand-drilling varies between 30 and 70 ft., and has a mean of about 40 ft. a month. In vertical shaft-sinking, the machine suffers more from lost time in raising and lowering, while the human labourer gains in having all down-holes; hence there is nothing to choose between the two systems in speed (about 60 ft. a month), while the cost is in favour of hand work. In incline shafts, the machine asserts its superior speed, and accomplishes 100 to 150 ft. a month. In wide stopes, too (3 ft. and upwards), it can compete with manual labour, but in narrow places it is handicapped. Though more costly all round, machine-drilling is forced on most Rand mines by the demands for rapid "development" to keep pace with the milling capacity.

South African costs are given by L. I. Seymour (March 1899) as follows: Hand stoping in medium ground up to 4 ft., by Kaffirs, 5*s.* to 6*s.* per short ton. Machine drills on 6-ft. stopes cost the same. Each drill costs 115*l.* to 125*l.* a month to run, including power, labour, supplies (not explosives), and depreciation. Sometimes a white man runs two drills, and breaks 960 t. a month. Some detailed costs of machine-drilling by the same authority (September 1897) are given on p. 213; and similar costs from the Ferreira Co.'s report (1899), on p. 214.

Some records from Indian work are as follows: (*a*) At the Ooregum mine, 2 Climax drills sank a shaft 727 ft. in 12 months, or 60 ft. a month; (*b*) 4 drills drove 455 ft. of railway tunnel in 1 month, or 13 ft. a day; (*c*) 1 drill at the Nundydroog mine advanced a heading 115 ft. a month; (*d*) at the Nine Reefs mine, the same drills drive 6 times as fast as native workmen on contract, and at about the same cost per ft.

At the St. Andreasberg silver mines in the Harz, rock-drills worked by compressed air have recently been introduced. The air compressor is placed underground, and is driven by a Girard turbine. The special point of interest in connection with this installation is the regulator, which consists of an air reservoir hewn in the solid rock. By the employment of power-drills, it is found that in driving levels there is a saving in cost of 58·2 per cent., whilst 4·04 times as much work is done as that accomplished by manual labour. The cost of stoping by hand was 16*s.* as compared with 8*s.* 6*d.* by machine per cub. metre of ore won. In shaft sinking, the cost with machine, inclusive of explosives, was 4*l.* per running metre, as compared with 7*l.* 10*s.* with manual labour.

In America, light power-drills for stoping seem to have grown in favour more than in S. Africa, though a really handy machine has been lacking. At Sundum, Alaska, 2 men working a "baby" Ingersoll (weighing 175 lb.) drilled 40 ft. a day (7½-hour shift) in hard seamy quartz 1 to 3 ft. wide, and only performed half as much by hand-drilling. Several Californian mines, notably the North Star and the Utica, having abundant water power, find machine-stoping much cheaper than hand, as indeed they obviously should, hydraulic power being very cheap and wages in California not less than 10 to 12*s.* a day. Colorado mines mostly use a 3 to 3½-in. cylinder machine for large work; and a "baby" machine with 2-in. cylinder, taking ⅞-in. starters and ¾-in. long drills (seldom over 6-ft. holes), handled entirely by one man, for stoping, rising, and prospecting cross-cuts. According to V. G. Hills (En. & Min. Jl., Sept. 1, 1900), the following work has been done in highly-indurated comparatively-seamless andesitic breccia (hardness 6 to 7, sp. gr. 2·4 to 2·7): Total length of holes per 8-hour shift, 35 to 45 ft., average 39 ft.; number of holes, 9 to 12; advance of

Running Costs, 3¼-in. Power Drill.

	Cost per drill per month.			% of total cost.	Cost per ft. of hole drilled.
	£	s.	d.		d.
Coal for power	15	0	0	12·53	2·92
Oil, waste and general stores ..	2	1	9	1·74	0·41
Repairs to drill, hose, &c.	1	10	0	1·25	0·29
Spare parts of drill, including hose	9	12	4	8·03	1·87
Repairs to compressor, boilers, &c.	0	15	0	0·63	0·15
Kaffir stokers (2) at 3*l.* 10*s.*, and 1 white stoker at 22*l.* 10*s.* ..	1	0	0	0·84	0·19
Engine-drivers (3), 28 shifts each per month at 18*s.* 4*d.* (less ½ as these men drive 2 other engines) = 77*l.* 2*s.* 10*d.*	1	5	8	1·07	0·25
Sharpening bits, and cost of steel	12	10	0	10·43	2·43
White men (2) operating the drill, 28 shifts at 1*l.* each	56	0	0	46·77	10·91
Kaffirs (4) at 3*l.* 10*s.* each	14	0	0	11·69	2·73
Interest at 7 % on half the equipment at 450*l.* per drill	1	6	3	1·10	0·25
Redemption at 12½ % on 450*l.* ..	4	13	9	3·92	0·92
	119	14	9	100·00	1*s.* 11·32*d.*

Work done, 20 to 24 ft. of hole per 12-hour shift, or, say, 22 ft. × 2 shifts × 28 days = 1232 ft. per month. Air at 60 lb.

Running Costs, 2¼-in. Power Drill.

	Cost for 2 drills per month.			Cost per ft. of hole drilled.
	£	s.	d.	d.
Coal for 2 drills	15	0	0	1·78
Oil, waste, sundry stores	3	0	0	0·36
Drill spares	12	0	0	1·42
Repairs to compressor, boiler, &c. ..	0	15	0	0·10
Stokers	1	0	0	0·12
Engine-drivers	1	5	8	0·15
Sharpening bits and steel	12	10	0	1·49
Drill-men	63	0	0	7·50
Kaffirs	35	0	0	4·17
Interest on equipment	1	6	3	0·15
Redemption	4	13	9	0·56
	149	10	8	1*s.* 5·80*d.*

Work done, 36 ft. of hole (finishing 1 in.) per 12-hour shift between the 2 machines, or, say, 36 ft. × 2 shifts × 28 days = 2016 ft. per month. Air at 60 lb. Labour, 1 white man each shift and 5 Kaffirs—2 to each drill and 1 carrying water. In another instance the figures were 1*s.* 2*d.* per ft. as against 1*s.* for hand-drilling.

Running Costs of Power Drills, Ferreira Mine.

	Driving, 7 × 5 ft.		Cross-cutting, 7 × 10 ft.		Rising 9 × 5 ft.		Stoping.				
	Cost per ft. driven.		Cost per ft. cut.		Cost per ft. risen.		Cost per cub. fathom (216 cub. ft.).			Cost per short ton.	
	s.	*d.*	*s.*	*d.*	*s.*	*d.*	£	*s.*	*d.*	*s.*	*d.*
Labour, white	14	1·66	19	10·50	16	2·86	1	10	11·66	1	8·65
„ native	3	1·80	4	5·14	3	2·89	0	8	0·01	0	5·33
„ „ keep	1	0·15	1	5·08	1	0·35	0	2	6·23	0	1·68
Compressing air	11	1·38	15	7·51	16	8·63	1	9	10·31	1	7·91
Extension and maintenance of air-pipe lines	0	10·99	1	3·44	1	4·52	0	2	5·49	0	1·64
Maintenance of rock-drills	3	4·57	4	9·03	5	1·02	0	9	0·98	0	6·05
Steel, sharpening, and transport ..	2	10·71	4	0·78	4	4·20	0	7	9·23	0	5·18
Totals	36	7·26	51	5·48	48	0·47	4	10	7·91	5	0·44

NOTE.—The stoping cost is 5·035*d.* per cub. ft., and 12 cub. ft. = 1 short ton.

cross-cut 4×7 ft, 4·3 to 4·7 ft. per double shift, based on monthly returns; rising 4×8 ft., 2 ft. per shift, and securing the ground pending proper timbering; shaft-sinking, 900 to 1000 ft. of $17 \cdot 1 \times 8 \cdot 2$ ft., 2 drills, 5 ft. every 2 days, machines working only 1½ shift per 48 hours, to allow of timbering and removing dirt. Here machine men receive 16*s.* 8*d.* a day, and other miners 12*s.* 6*d.*

Notwithstanding the tedious nature of single-hand drilling, several authorities strongly advocate it in all but the hardest rock and in shaft-sinking. Doubtless, where capable single-hand miners can be had, there is abundant justification for the preference. Firstly, the single-hand hole is much smaller, and the actual economy in a ¾-in. hole as against 1¼-in., so far as amount of rock pulverised is concerned, is about 3 to 1; though this is somewhat discounted by the reduced force of the blow delivered. Secondly, there is less opportunity for wasting time in conversation—no small item, when the work is done on daily wages. The saving in time and cost is often computed at 30 % in soft schistose rock, and at 20 % in easy sandstone. The best single-handed drillers are found in the ironstone mines, and in similar situations where the low value of the product necessitates cheap work. In most of the Colonial gold-mining centres, there is either a dearth of such men, or local prejudice and labour laws prevent the adoption of the system.

A point of much importance in connection with machine drilling is facility for connecting and disconnecting air-pipes and for mounting and dismounting drills; and it is a singular fact that manufacturers, as a rule, while paying much attention to the structural detail of machines are quite oblivious to this essential feature. As an example, the author has found it always necessary to discard the poorly-threaded and scantily-metalled fittings for attaching air-pipes, and to construct others which would stand hard knocks, and not be rendered useless by the first forcible contact with a piece of hard sharp rock. So, too, with the king-bolts, which are commonly supplied so light in substance and with such a fine thread that stripping is almost sure to occur sooner or later; a 1½-in. bolt with deeply-cut No. 4 thread and 2-in. nut is none too heavy, and this view is endorsed by such an experienced Californian as W. C. Ralston. This eminent authority has also employed an improvement on the U-bolt, which saves much time in changing drills. "The two nuts on the U-bolt are riveted, a key-seat is cut in its upper end, and a wedge is inserted which has a lug in the small end to prevent its falling out. This wedge is put in the U-bolt with the taper side towards the end of the chuck, so that, after it is driven home, the constant hitting of the drill keeps tightening it. A couple of blows with a hammer loosen it and release the drill."

Some costs published by the Centre Star Mining Co., Rossland, B.C., show as follows:—

Cost per ft.	Sinking Main Shaft.	Sinking Small Shafts.	Rising.	Driving.
	s. d.	s. d.	s. d.	s. d.
Compressed air	14 6	13 7	12 9½	5 7¼
Drill fittings	6 1	6 4½	7 7½	2 8½
Labour	102 11	83 10½	69 10½	30 2¼

Average hand-drilling speeds in the United States and Canada are quoted by Gillette ('05) as follows:—

(1) 1½ in. starting bit, down hole (vertical), 2 hammers, open quarry—granite, 8 in. per hr.; basalt, 13 in.; limestone, 18 in.

(2) 1¾ in. starting, 1¼ in. finishing, 6 ft. deep, 1 hammer (8 lb.), diorite and porphyry, 18 in. per hr.

(3) 1¼ in. starting, 1 hammer, dolomite, 14 in. per hr.

(4) 20 ft. deep, vertical, 2 hammers, porphyry, 7–9½ in. per hr.

(5) ⅞ in. starter, 1½ ft. deep, 1 hammer, sandstone, 9–11 in. per hr.

It was found by W. M. James ('03) that Kaffirs on piece work (Rand) drilled 4·02 ft. per shift as against 3 ft. only on wages.

Machine drilling averages, with 3⅛-in. machine, at 70 lb., 2¾-in. starting bit, 1½-in. finishing, range from 3 min. per ft. in soft but gritty rock, such as sandstone and limestone, to 7–8 min. for hard granite; while very soft rock causing much sludge may require 10 min. per ft. (except with a water jet); and some highly elastic and tough rocks having an asbestos-like structure are almost unborable, the time amounting to several hours per ft.

Some American averages quoted by Gillette ('05) on 10-hr. shifts, thus embracing time lost in changing bits, setting up, etc., on deep (10–20 ft.) vertical holes in quarrying, are about 4 ft. per hr. in hard trap, 5 ft. in ordinary granite, 4–7 ft. in mica schist, 7–8 ft. in limestone, and 9–11 ft. in sandstone.

It is generally assumed that a good rock drill will do the work of 6–10 men. It takes 5–12 h.p. to furnish a drill with compressed air, and, with the exception of what are known as "Baby" drills, it takes two men—a "machine man" and a "helper"—to operate a machine.

The average drill in shaft-sinking consumes 100–120 cub. ft. of free air per min., compressed to about 90 lb.; in drifting, 70–100 cub. ft.; and in stoping, 40–70 cub. ft. In general, while holes are drilled very frequently to a depth of 8 ft., they average about 4½ ft., and the size of the hole is such as will permit the use of sticks of powder 1–1½ in. diam. The average work of a rock drill for one shift is 30–40 ft. of holes.

To ascertain the minimum cost of machine drill upkeep under the best conditions, a trial was made on one of the Rand mines, in 1897, the machine being placed in the hands of competent miners, and subject to special supervision. The machine was specially prepared before being sent underground, the cylinder being very carefully bored out, to ensure a glassy-like surface inside, and to make it perfectly parallel. A special piston, made from high carbon steel, was made and ground into the cylinder, great care being taken to ensure the tightest possible working-fit; the piston was made dead hard, except the chuck-rod and chuck. A special valve of tool steel was formed and fitted to the valve chest, after it had been carefully bored out, the same precautions being taken to ensure a good tight fit, as in the case of the piston and cylinder. The remaining portions of the machine received very careful attention, and everything possible was done to ensure the machine being in perfect order. The machine ran 14 months, double shift, in the hands of the same men, with the exception of four occasions when it was in the shops for extensive overhaul of cradle and front head; on each occasion it was laid aside 24 hours. During the period no work was done on the piston proper. The machine was finally injured by blasting. The wear between the cylinder and piston amounted to ·16 in. The following were the costs:—

	£	s.	d.
Machine repairs at start	15	0	0
11 chuck bushes, at 12*s*.	6	12	0
9 chuck bolts, at 4*s*. 9*d*.	2	2	9
4 new piston valves, including reaming chest	4	15	0
6 rotating nuts, at 12*s*. 6*d*.	3	18	0
12 sets ratchet pawls, at 9*s*.	5	8	0
3 feed screws, at 21*s*.	3	3	0
3 feed nuts, at 21*s*.	3	3	0
6 sets front-head bushings and leathers, at 20*s*.	6	0	0
Labour	26	0	0
Allowance for mechanical power	17	10	0
Supervision, office charges, etc.	14	0	0
	£107	11	9

Equal to £7 13*s*. 8*d*. per month.

The approximate footage drilled was 13,104 ft.

Of great importance in the operation of rock drills is the question of maintenance. No motor is subjected to such hard usage, both in the nature of its work, and by the carelessness of the men who run it. A good man, taking pride in his work, will keep a machine in use for months, where a careless man will send it up for repairs weekly. The bill for maintenance would be halved, if men were charged for every spare part, increasing the price per ft. to cover reasonable wear and tear. The necessity for plenty of drills is admitted, but in most cases there would be steel enough and to spare, if the men looked after it. There is not a big mine on the Rand in which tons of steel are not buried. Drills are thrown down the stopes at random—may reach the level and may not—and the miner does not trouble much either way, because they cost him nothing. If every man had issued to him a set of bits when he started, and had to account for these when he left, and to pay for any required over the set, and for all drills sharpened, the steel bill would come down with a run. Instead of relying on a few men to "boss up" the drills underground, every man who used drills would become at once a boss, looking after his own, he would keep a sharp look out that his drills were in good order, well sharpened, and gathered into safe places as soon as used; that waste was reduced to a minimum, and that sharp bits were not sent back to the shop. But while we may effect economies in power, and maintenance, and stores, the great saving must be in the labour employed; not by reducing the wages of the rock-drill man, but by increasing his efficiency. It is the man behind the drill who counts, but the degree to which he counts may be modified by adopting a different system of stoping. Men fail to break ground efficiently, not because they are unwilling, but because they do not know how. It is easy to train a man to run a machine and bore holes; it is difficult to train him to break the maximum tonnage, with its minimum quantity of explosives. The latter art makes the miner, and it requires long experience and good judgment of ground. The demand for such men might be partly met by dividing the work of rock-drilling into two classes: the first to lay out and blast the holes, and the second to bore them. The man behind the drill would thus be responsible solely for getting down the holes pointed out to him, as many as he could per shift, and he would be paid at so much per ft., or per fathom, and, as he would not handle explosives, his whole time would be devoted to drilling. He need not be a skilled miner to do this, but he would have to know all about operating a rock-drill. The men of the other class would be experienced all-round miners, selected on account of their ability to break ground, and to economise explosives. Each would be in charge of a certain section of the mine, and his duty would be to visit the stopes under him twice daily, to study the ground and pitch the holes, so as to get the greatest possible burden. In the afternoon, the same man would charge and blast such holes in his section as he deemed advantageous, aiming at economy of explosive. He would be on a regular salary, with a bonus for tonnage broken, to be shared with the rock-drill man. Such a system would not affect the wages of miners, except for the better, and would by degrees train indifferent machine men into first-class miners; but it probably would not be introduced without strenuous objections on the part of the miners. Yet it is certain that a system on these lines would very greatly increase rock-drill duty, and reduce costs in proportion. (Wager Bradford.)

The wisdom contained in the advice given in the foregoing paragraph is so obvious that one would have thought it unnecessary, especially addressed as it is to mine-managers on the so-called "foremost camp in the world." The author's practice everywhere is to pay an inclusive figure per ft. or per ton, and to charge *everything* which the miner takes out of the store—candles, explosives, steel, shovels, picks, axle grease, etc., etc. This is the only true principle to secure economy, and prevent reckless waste of supplies.

As to the respective merits of large and small drilling machines, some results obtained in Colorado (F. T. Williams, E. & M. Jl., May '06), on parallel work by a 3½-in. and a 2¼-in. machine, are most instructive. The ground was highly indurated andesitic breccia, having a hardness of 5·2–7·2. The large machine (*a*) drove a heading 7 ft. 6 in. by 5 ft. 6 in.; the small one (*b*), 4 ft. 6 in. by 7 ft. The respective figures were:—

	Labour per ft.		Explosives per ft.	Explosives per ft.		Total Cost per ft.		Total Cost per ton.		Ft. Driven per shift.
	s.	*d.*	lb.	*s.*	*d.*	*s.*	*d.*	*s.*	*d.*	ft.
(*a*)	14	4	15·40	8	7	36	7	11	6	2·76
(*b*)	10	8	9·27	5	4	26	8	9	3	2·44

The net conclusions arrived at were that the small machine saves 25 % in labour ft. per ft., and 50 % in total operating cost shift per shift; tramming is reduced by 20 %, explosives cost per ft. 37·7 %, and general expenses 20 %. But the small machine is 10–20 % slower. Total costs per ft. run of heading were 27 % in favour of the small machine.

Machine-drilling at the Treadwell group, Alaska, gives the following averages per machine per 10 hr.—Holes, 32 ft.; powder, 14 lb.; dirt broken, 33½ t.; cost: labour, 31*s.* 7*d.*; explosives, 9*s.* 5*d.*; drill sharpening, repairs, supplies, power, etc., 10*s.* 2*d.*; total, 51*s.* 2*d.*

Diamond-drilled Holes.—In the Minnesota iron-mines, diamond drills are used for putting down shot-holes ranging from 20 to 33 ft. and averaging 25 ft. deep, for stoping. The normal speed attained is 12 ft. in 10 hours. Cost of such drilling exceeds that of percussion-drilling; but the product broken per ft. of hole is much greater, rendering the actual expense per ton of ore won appreciably less.

Sometimes, in sinking a vertical shaft, recourse is had to diamond drills in a somewhat similar manner, the total number of holes required for the entire excavation being bored to the full depth before any blasting is done. Thus the boring is continuous until finished, and a very large proportion of the time wasted in intermittent drilling is saved. Examples from American colliery work are—35 holes in an area of about $25\frac{1}{2} \times 14$ ft. were bored by 3 drills in 6 weeks through 300 ft. of hard rock; and 25 holes sufficed for a shaft 16×14 ft. When boring has ceased, each hole is filled with water or sand to within 3 or 4 ft. of the top, plugged with clay at that depth, charged, and fired. After removal of broken ground, the operation is repeated as many times as are required for the full depth of the shaft.

The speed of sinking is very much increased in all cases where the ground is suitable for diamond drilling; and under, the most favourable circumstances, there is also a material reduction of cost.

See also STOPING PRACTICE.

BLASTING.

In blasting, rock requires ·25–1·5 lb. gunpowder per cub. yd., according to its degree of hardness, and position. In small blasts 2 cub. yd. have been rent and loosened, and in very large blasts 2 to 4 cub. yd. have been rent and loosened, by 1 lb. powder.

Tunnels and shafts require 1·5 to 2 lb. per cub. yd. of rock.

Gunpowder has an explosive force varying from 40,000 to 90,000 lb. per sq. in. That used for blasting is much inferior to that used for projectiles, the proportion being fully one-third less.

Nitro-glycerine is an unctuous liquid, which explodes by concussion, an extreme pressure (2000 lb. per sq. in.), or a temperature exceeding 600° F. if quickly applied to it; it will inflame, however, and burn gradually.

At a temperature below 46° F. it solidifies in crystals.

Its explosion is so instantaneous that in rock-blasting tamping is not necessary; its explosive power by weight is 5 to 6 times that of gunpowder.

No. 1 *Dynamite* is nitro-glycerine 75 parts, absorbed in 25 parts of a siliceous earth termed kieselguhr; it also explodes so instantaneously as to render tamping in blasting quite unnecessary.

It is insoluble in water, and may be used in wet holes; it congeals at 46° F., is rendered ineffective at 212° F., and has an explosive force by weight of 4 times that of gunpowder, and by bulk 6 times.

Gun-cotton is insoluble in water, and has an explosive force by weight of 2·75 to 3 times that of gunpowder, and by bulk 2·5 times. It may be detonated in a wet state with a small quantity of dry material.

Tonite, or as it is more commonly called, cotton powder, is nitrated gun-cotton with a mixture of baryta, etc., and is manufactured in the form of cartridges. It contains no nitro-glycerine, is considered equal in strength to No. 1 dynamite, has the advantage of not being affected by climatic changes, and is much safer to handle than dynamite. It is used extensively for subaqueous blasting.

Cellulose Dynamite is when gun-cotton is used as the absorbent for nitro-glycerine; it will explode frozen dynamite, and is more sensitive to percussion than it.

To Compute Charge of Gunpowder for Rock Blasting.

Rule.—Divide cube of line of least resistance by 25 as for limestone, to 32 for granite, and quotient will give charge of powder in lb. Or,

$$L^3 \div 32 = \text{lb.}$$

Example.—When line of least resistance is 6 ft., what is charge required?

$$6^3 \div 32 = 6{\cdot}75 \text{ lb.}$$

Line of least resistance should not exceed half depth of hole.

Tamping.—Dried clay is the most effective of all materials for tamping; broken brick the next, and loose sand the least.

Charges of Powder.

Usual practice of charging to one-third depth of hole is erroneous, inasmuch as volume of charge increases as square of diameter of hole. Hence holes of 1·5 and 2 in., although of equal depths, would require charges in proportion of 2·25 and 4.

Line of Least Resistance.	Powder.		Line of Least Resistance.	Powder.	
ft.	lb.	oz.	ft.	lb.	oz.
1	0	0·75	5	3	14·5
2	0	4	6	6	12
3	0	13·5	7	10	11·5
4	2	0	8	16	0

Weight of Explosive Materials in Holes of Different Diameters per inch of Length.

Diameter.	Powder or Guncotton.	Dynamite.	Diameter.	Powder or Guncotton.	Dynamite.
in.	oz.	oz.	in.	oz.	oz.
1	·419	·67	2·25	2·12	3·392
1·25	·654	1·046	2·5	2·618	4·189
1·5	·942	1·507	2·75	3·166	5·066
1·75	1·283	2·053	3	3·769	6·03
2	1·675	2·68			

Effects.

Gunpowder, from its gradual combustion, rends and projects rather than shatters.

A hole 5·5 in. diameter and 19 ft. 7 in. deep, filled to 8 ft. 10 in. with 75 lb. powder, has removed and rent 1200 cub. yd., equal to 2400 tons. The labour expended was that of 3 men for 14 days.

Temperature of gases of explosion, 4000° F.

Gun-cotton, from the rapidity of its combustion, shatters.

Dynamite, from the greater rapidity of its combustion over gun-cotton, is more shattering in its explosion.

In small blasts, 1 lb. powder will loosen about $4\frac{1}{2}$ tons. In large blasts, 1 lb. powder will loosen about $2\frac{3}{4}$ tons.

In small charges of dynamite, weak caps seem to develop the full strength of the explosive; but in large charges, stronger caps are necessary. Gun-cotton and gelatine-dynamite require much stronger caps than those ordinarily used for dynamite. Gun-cotton seems to produce a more rending explosion than dynamite. The statement, made on good authority, that black powder fired with a detonator is much more powerful than when fired by fuse alone, has been disproved.

Precautions and Methods of Using.

Testing.—There are certain simple tests by which it may be known whether explosives of the dynamite class are in good or bad condition. A perfectly good stick will not have a greasy feel, nor will any of its nitro-glycerine have oozed out and stained the paper wrapper. If in doubt, place the stick on clean dry brown paper in a temperature of 60–80° F. for 12 hr.: the paper should remain quite free from discoloration. Freezing and thawing greatly increase the tendency of the nitro-glycerine to separate out, and explosive which has thus suffered should be specially examined. Inferior qualities will weep in continued high temperatures also. Damp will cause an efflorescence of nitrates, due to leaching out of these soluble salts, and such dynamite is unreliable and dangerous. Worst of all is decomposition of the nitro-glycerine compounds, producing greenish spots on the covering paper: this is dangerous in the extreme.

Dynamite freezes at 42–50° F., and will freeze hard when water or moist earth does not freeze at all. Sometimes it will not freeze until the temperature is down to about 35°, owing to the protection of the boxes, and to the fact that the absorbents used for holding the nitro-glycerine are poor heat conductors. For the same reason, it sometimes does not thaw readily at a temperature of 50–60° F. When dynamite is completely frozen, the cartridges are hard. In this condition, it may refuse to explode at all, even when a strong cap is used; and it surely will not develop anything like full strength. It must be carefully and thoroughly thawed, to make sure that it will explode and do good work. At temperatures considerably above the freezing point, dynamite will chill, and must be slightly warmed in order to do satisfactory work; and the blaster should make sure that it does not again become chilled or frozen while being carried to the drill holes and loaded.

Dynamite can often be burned in open air without explosion; but practically all the accidents which have happened in thawing can be traced to a mistaken idea that dynamite can be safely burned under all conditions, and that any exposure to heat, short of actually setting fire to the cartridges, is safe. It is true that a dynamite cartridge can be ignited with a match, and completely burned without explosion, "most always," and this is because only the part of the cartridge that is burning becomes hot; the gases pass away freely, and the balance of the cartridge remains cool. If this same cartridge were laid against a hot steam pipe or smoke stack, the chances are that it would explode before it could take fire. It is absolutely certain that, if allowed to get really hot, it would explode from the slightest jar, or from the blow of a tamping rod. Cartridges heated in sand have exploded from simply rolling them about in the sand. Cartridges set on end around a fire have exploded from falling over. A bunch of cartridges, laid on top of a brick-covered boiler to thaw, took fire and exploded when struck by the end of a board in the effort to knock them off the boiler. All these accidents were the result of using a heat which could not be controlled, and the cartridges reached a very high temperature before the man in charge became aware of it. The remedy is to use a source of heat which cannot possibly rise above 100° F., even when neglected. Dynamite begins to undergo a change at 158° F., and, from above that temperature, becomes more and more sensitive to shock, until, at a temperature of about 356° F., it will explode simply from the heat.

Another mistaken idea is that dynamite is waterproof, and therefore any amount of soaking or steaming will not injure it. Many blasters will deliberately soak dynamite in hot water for ½ hr., or thaw with live steam, under the supposition that such treatment does not injure it. Dynamite soaked in hot water for 15 min. and then allowed to steam for ½ hr. has taken up over 10% water, an amount that would possibly make it non-explosive. Certainly it would reduce the explosive force by half. Dynamite left for a day in a thaw-house filled with escaping steam has taken up 18% moisture. It is not known exactly how much water is required to render dynamite non-explosive, but 5% greatly reduces the explosive force, and cartridges containing 12% have been rendered non-explosive.

Thawing.—For thawing quantities of 10–15 lb. of dynamite at a time, the well-known "Rundle" pattern of thawer is most suitable. The cartridges are inserted in tubes, and the thawer is filled with hot water (not above 170° F.), which completely surrounds each cartridge. The cover assists in retaining the heat, and the dynamite may be carried to the work in the thawer, and kept in good condition for hours. On no account should the thawer be placed over a fire, nor should any attempt be made to heat water in the thawer while there is dynamite in it. It is better not to heat water in it at all.

A good thawer, with a capacity for a case of dynamite at a time, can be made by burying a box in a pit filled with green manure, which should be rammed hard, and must be occasionally renewed. An iron pipe is used for a ventilator, as well as a handle for lifting the cover. Such an arrangement will thaw dynamite in the coldest weather. Dynamite should not be stored in it longer than necessary for thawing, because the dampness from the ground and manure would in time injure it.

A properly-constructed house for thawing dynamite, with a capacity of 500 lb., is shown in Fig. 53. The thawing house proper is 4 ft. 7 in. by 8 ft. in plan; height at front inside, 5 ft. 6 in.; height at back inside, 4 ft. 6 in. It is built of studding, sheathed inside and out with 1-in. plank, the outside being covered with corrugated iron. On the front, are four doors opening outwards. Inside these doors are 5 tiers of drawers, each tier being 4 drawers wide, in all 20 drawers, each drawer holding 25 lb. of dynamite, a total capacity of 500 lb. The bottoms of the drawers are perforated, to allow hot air to circulate.

Behind the drawers is a baffle, extending down from the ceiling to within 1 ft. of the floor. This divides the thawing house into two rooms or chambers—the front room containing the 20 drawers; the back room, a hot-water radiator. The radiator has a sheet-iron curb or baffle around it, to throw the hot air up inside the baffle, whence it goes down, outside under the centre baffle, and thence through the drawers containing the dynamite. There is a small drawer in the back of the house, to allow a man to enter if necessary to repair radiator or pipes, but for no other reason. It is impossible for the powder-man to get into the room where the thawing is done, thus avoiding the too common practice of "priming" in a thawing room, because of its comfortable temperature.

The house can be set on a brick floor, doing away with all foundations and floors except sills. The thawing drawers can be run into sleeves of wood just large enough to receive them, and thus be carried to the blasting face; or the thawed cartridges can be taken from the drawer and placed in a tight box, the thawed cartridges to be covered with sawdust, a woollen cloth, or other covering to prevent re-freezing.

The temperature in dynamite thawing houses heated by hot water can be satisfactorily controlled by a "regulator," such as Power's, Tagliabue's, or Hohmann & Mauer's. The maximum

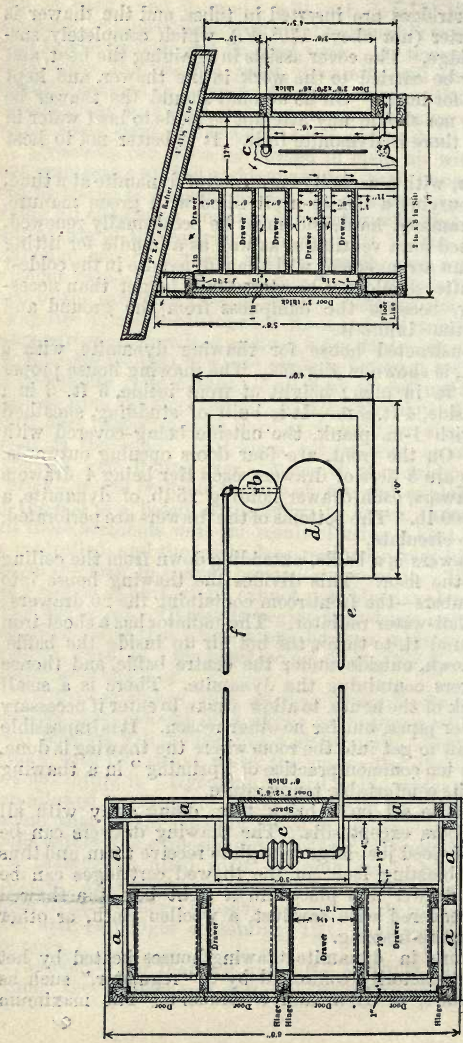


FIG. 53.—DYNAMITE THAWING HOUSE.

should not exceed 80° F. A thermostat should be placed in the thawing house, above the drawers, and on the side toward the doors. A small hydraulic pump is set in one corner of the building, behind the drawers, with a compressed air tank above it. This should be operated from the same water system as is used with the water-heater. Compressed air at 15 lb. is led from the air tank through the thermostat, in armoured lead tubing, and through a diaphragm valve replacing the elbow on the hot water supply line to the radiator, the opening and closing of this valve maintaining the temperature desired. Or, the air may be carried back to a diaphragm in the boiler house, where it would operate a damper in the draft door of the boiler, as well as a check draft in the smoke connection, and increase or decrease the temperature of the circulating water by the operation of these dampers.

But instead of this complicated and expensive arrangement, a small building heated by steam or hot water pipes, or radiators, may be constructed 50 ft. or more away from any other building or mine workings. The pipes or radiators should be placed in one end of the room, and encased in such a manner that it will not be possible for anyone to put dynamite where it can touch these pipes, or where any drop of nitro-glycerine, which might possibly exude from the cartridges, could come in contact with the pipes or radiators.

A good thawer can be provided by building a small house, in one end of which are coils of pipe, through which exhaust steam from the engine passes. Because of the fact that the steam is not confined, it is impossible to create dangerously high temperatures in such a room. Along the sides away from the heater, shelves are placed, and here the cartridges are laid on their sides until thoroughly thawed.

The drawers should measure $22\frac{1}{2} \times 16\frac{1}{2} \times 5\frac{1}{8}$ in. deep inside, with 1 in. slat bottoms at $\frac{3}{4}$ in. spaces, or 1 in. auger-holes. Spaces marked *a* may be left as air spaces or packed with sawdust, powdered charcoal, or ground talc. The heater-house should be at least 10 ft. from the thawing room, and the bottom of the expansion tank *b* should be above the top of the radiator *c*, with 1 in. drain-pipe *f* leading back from *c* to bottom of heater *d*. The hot-water pipe *e* issues from top of heater *d*. A ventilator at each end of the front of building, and at each end under the back-door sill, must be protected inside and out with $\frac{1}{2}$-in. wire netting.

Steam may be used instead of hot water, but in that case it should never exceed 10 lb. per sq. in., and there should be no valve at the outlet, or other possibility of preventing free issue.

Tools should never be permitted in a building where dynamite is being thawed, for it is almost impossible to prevent small particles of explosive from being scattered about, and a blow would explode them under certain conditions. Primers should never be prepared in a thaw house; even the most careful man will sometimes leave a cap where it may cause trouble. A thaw house should be so arranged that the floor can be kept clean and occasionally scrubbed with hot lye. The slats or shelves on which cartridges are laid should be removable, and subjected to the same treatment. The wood will, in course of time, take up some nitro-glycerine, which may be much more dangerous than the dynamite.

For carrying the dynamite into mines and keeping it thawed, a double-walled tin vessel, the interspace filled with non-conducting material, and having a non-conducting cover, the inside of the can kept brightly polished, is satisfactory for small quantities. Where the consumption is large, it is desirable to have special small thawing rooms, properly safeguarded, underground. Such an arrangement was fully described (and illustrated) by W. Kelly, in E. & M. Jl., Aug. 19, '05.

Application.—For the proper application of any explosive, to secure maximum results with minimum consumption, certain precautions must be observed, among which the following are most important:—

Select the right fuse for the kind of work, and proper caps for the kind of powder, and see that both are thoroughly dry.

Powder must not get shaken out from end of fuse, nor sawdust or other obstruction get in between fuse and cap composition. Cutting fuse slanting not only allows a little of the powder to shake off, but often makes an obstruction to the fire because the slender end may fold under. Also a sharp-pointed piece of fuse is not a desirable thing to thrust into any cap.

Cut the fuse straight across, not slanting, and push it into the cap ½ in. or more, all the way down to the powder. If the fuse be ragged at the end or too large to enter the cap easily, never peel off any of the tape or yarn, but swage the end of the fuse to the proper size. This may be easily and quickly done by twisting and squeezing the large part with the crimper, if it be a broad one. Having inserted the fuse, squeeze the shell tightly to it with a broad crimper placed around the shell so that one side just overlaps on to the fuse. This will make a compression about ¼ in. wide around the extreme upper end of the shell.

The holes should be carefully charged, squeezing each cartridge separately with a wooden rammer, so as to fill the hole completely to the desired height.

Having crimped the cap securely to the fuse, insert all the cap —but none of the fuse—into a stick of powder, and tie together; then put this priming stick upon the rest of the powder in the hole, and do not ram it until some clay or other tamping has been put in. Use tamping without any sharp grit in it, so as not to damage the fuse.

Wherever a whole blast may be fired at once, and for all work in very wet places, electrical fuses will be found of advantage.

The selection of the type of explosive is a matter requiring much study. For example, it has been found with Cleveland ironstone that the high explosives are quite unsuited, pulverising the ground without bringing it down, and better work has been done by compressed pellets of black powder weighing about 2 oz. each. As a general rule there is a great tendency to use most unnecessarily strong explosives, and to use them most extravagantly. Often the employment of blasting gelatine is quite wrong in principle, because, while it allows a maximum burden on the hole, it also creates the maximum proportion of fines (through its shattering effect), and these fines cannot possibly be sorted out from the milling dirt.

The practice of selling to the contractor or miner every lb. of explosive, ft. of fuse, and box of caps he uses, and paying an extra price to him per ft. or per ton, is eminently sound, and, in the case of many types of native labour, is the only course to be thought of as a preventive of wholesale stealing.

Sometimes, instead of allowing holes of various depths, burdens and charges according to the ground, it will be found preferable to adopt a standard charge of say $\frac{1}{5}$ lb. dynamite per hole; but usually an improvement on this system is to place in the hands of selected men the duties of setting, loading and firing the holes, leaving the miners only to do the actual drilling according to orders.

"Chambering," and the use of more than one kind of explosive in the same hole, has great advantages in some cases, and will be further described under Mining Methods.

Cost.

The range of cost for blasting in mining work is as wide as the variation in the principal factors—tightness of ground, extent of face, and price of explosive. Where black powder can be used, as in easy ground, and in many open-cut workings, very large masses can be moved with very small expenditure, and the proportionate cost for the operation sinks to insignificance. In other cases, toughness of rock, limited size of ore-body, and selling price of high explosives may combine to render the figure the greatest in the total cost of winning the ore.

In most metalliferous mining, the situation demands a high explosive of the dynamite class, containing say 65–100% nitro-glycerine. The ordinary brands are: No. 2 dynamite, 65%; No. 1, 75%; gelignite, 80%; gelatine-dynamite, 85%; blasting gelatine, 100%.

Very commonly the cost per short ton of ore extracted is 6*d*. to 1*s*., under normal conditions.

On the Rand, it varies mostly between 1*s*. and 3*s*. a ton. Actual figures at the Ferreira mine (1899) are as follows:—

	ft.		*s*. *d*.		*d*.
Per ft. driven	7 × 5	Dynamite,	20 11·60	Caps and Fuse,	6·00
" "	7 × 10	"	29 5·70	" "	8·40
" risen	9 × 5	"	14 11·09	" "	4·72
Per short ton stoped		"	1 6·42	" "	0·42

At another well-managed mine in 1896 they were :—

	s.	d.
Per cub. ft. blasted: dynamite, caps and fuse	0	4·4
Per short ton of rock removed: dynamite, caps and fuse..	4	9·5
Per lineal ft. of excavation: ,, ,, ,, ..	28	6·5

About 85 % of all the explosives used on the Rand is of the higher grade, about 60 % being blasting gelatine. Prices at depot, before the war, were : Gelignite, 85*s.* per case of 50 lb. ; gelatine-dynamite, 98*s.* 6*d.* ; blasting gelatine, 107*s.* 6*d.*

At Lucknow, N.S.W., Australia, the author paid about 1*s.* 5*d.* a lb. for gelignite delivered on the mine, and the working costs for explosives, including caps and fuse, were :—

1897 : 1*s.* 10·6*d.* per short ton stoped, plus 3*s.* 8·5*d.* per ft. of drives, etc.
1898 : 2*s.* 2·2*d.* ,, ,, ,, 3*s.* 3·6*d.* ,, ,,

The workings average 6 × 5 ft.

At the Centre Star Mine, Rossland, B.C., the costs for explosives (1900) are given at 18*s.* 3½*d.* per linear ft. for sinking main shaft, 21*s.* 7½*d.* for small shafts, 16*s.* 11*d.* for rising, and 10*s.* 2½*d.* for driving.

In Cleveland ironstone, 2 oz. pellets of compressed black powder surpass dynamite of any strength, and blasting costs only about 1½*d.* per long ton, each hole breaking some 2 tons.

Electric Firing.

In certain circumstances, the firing of loaded holes by electric current presents great advantages, more particularly in the matter of safety for the men and in securing the cumulative effect of a series of simultaneous shots. Especially in sinking shafts and winzes, the risks run by the "fireman" when using ordinary fuse are very great ; any hitch in the arrangements for his ascent after igniting the fuse, or any defect or miscalculation in the length of fuse, must almost certainly have fatal results. In drives and stopes there is not the same danger, because escape is so much more easy ; but even here two sources of accident are present, one being that a shot may miss-fire through faulty fuse, leaving a loaded hole in standing ground which the next shift of men may drill into ; and the other that it may be cut out by a previous shot and lie unsuspected in broken ground which is subsequently spalled or machine-crushed. Men are supposed to count the holes loaded and the number which explode, so as to obviate these risks, but they cannot always be relied on. Hence the value of a firing system which is certain. In quarry work too, where the market value of the product is so low that the utmost economy is of paramount importance, means of igniting a large number of shots at the same moment will effect considerable saving both in the number of holes and in the charge for each, when an extensive open face is available. The effect of simultaneous firing is considered to be nearly 1½ times that of single shots.

In apparatus for electric firing, that portion of the conductor, which is inserted in the ignited substance itself, must be heated thus the specific heat of this portion of the conductor must first be taken into consideration. As a rule, ignition occurs when the pro duct of the square of the current's intensity and the resistance of the point of ignition attain a certain value, there being two ways in which this may be brought about. Either a slight resistance must be given to this point, in which case a considerable intensity of current must be employed, as in incandescent ignition; or the resistance must be considerable, when the intensity of current may be greatly reduced, as in spark ignition.

It is important that the resistance of the point where ignition occurs be properly proportioned to the tension and quantity of the current; the igniter may possess too high or too low resistance for a given current. If the resistance be too high, no current will pass; if too slight, the current will pass through the ignition-point without raising its temperature to the required degree. For electric ignition underground, the current conditions of the igniting apparatus are of the highest importance.

In incandescent ignition, a small platinum wire is brought to incandescence inside the ignition-substance, the resistance of such wire being only about 1 ohm; and the quantity of current required to effect this is about ½ ampere. With a copper wire of 1 mm. diam., firing at a distance of 150 ft, the resistance will be about 2 ohms; with an iron one of the same diameter, it will be increased to 14½ ohms. Thus the resistance of the conductor as compared with that in the igniter is very great. To prevent the source of current from being inconveniently large with respect merely to the high resistance of the conductor, expensive copper wires are used in preference to iron for incandescent ignition, and copper cannot be dispensed with for the igniting wires themselves. On the other hand, the danger of loss of current, through insufficient insulation in damp situations, is not material; an advantage is that each igniter may be tested before being connected up, so that miss-fires are scarcely to be feared.

In spark ignition, the resistance of the igniter must be estimated at several million ohms as a minimum, and it makes no difference whether the resistance of the conductor be 2 or 15 ohms, therefore iron is just as good as copper wire. But perfect insulation of the conductor becomes a matter of urgency. Spark igniters are cheaper than incandescent ones. Their great disadvantage is that the frictional firing-machines employed with them are very ill-adapted to resist damp.

While simultaneous firing is highly beneficial in many cases, it is also often objectionable. In all confined work, a proper sequence in the shots is necessary, each hole or group of holes being planned to facilitate and augment the useful effects of the next (see p. 210). There is no "get-away," as in open-cut operations, and most of the advantages of group-firing are nullified. True, the interval need only be a second or so in duration, but this the ordinary igniter cannot give. Just as, in machine-drilling, economy is largely dependent on the ability to utilise a single mounting of the drill for making a number of holes, so, in shot-firing, the greatest advantage would be in reducing the time-losses of men waiting for fumes to clear away, by concentrating the firing—enabling a long series of holes to be fired in rapid sequence (not of necessity simultaneously) without repeated visits to the face for charging or connecting.

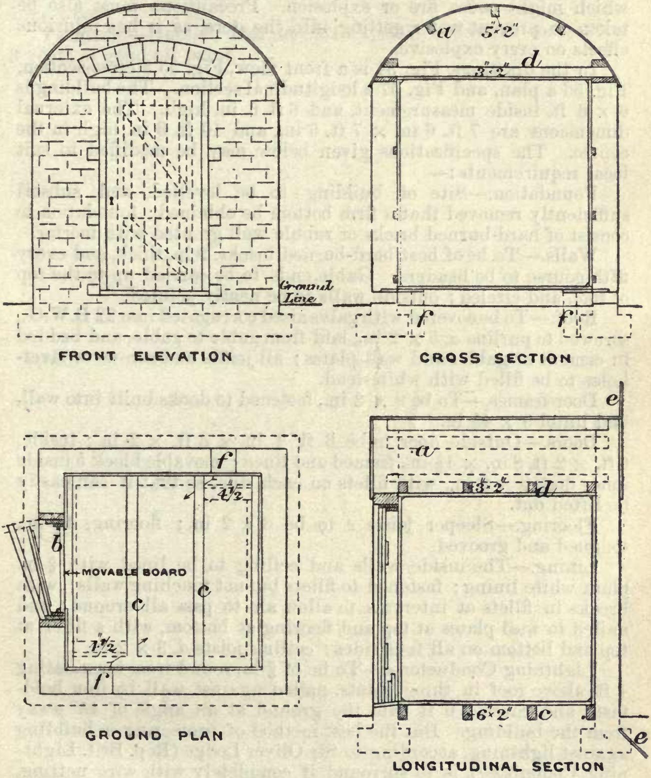
Within the last year or two, an igniter has been introduced in Westphalia which seems to successfully accomplish this very desirable object. It is a compromise between the incandescent and the spark igniter (locally called *Spaltglühzünder* or "split-incandescent-igniter"), with all the advantages of both systems and the disadvantages of neither. Each separate detonator can be tested by a galvanometer before use. They have been made with as low a resistance as 100 ohms, and act perfectly with naked iron wires, even in wet workings. By selection of igniters, it is possible to fire a series of shots (5 to 50) either simultaneously or singly, at intervals ranging from 2 to 5 seconds. Ordinary electro-magnetic firing machines of recognised patterns may be employed, but a somewhat superior article is turned out by the makers of the special igniters, N. Schmitt & Co., Küppersteg.

Condensing Fumes.—The condensation of fumes and purification of the air after blasting, by which the men's return to the face is much hastened, may be attained by immediately spraying water under great pressure (say 200 lb. per sq. in.) in the face; this is cheaper and much more effective than a jet of compressed air, which, moreover, is not available when manual or electric drills are used.

Explosives Store. (R. Hunter, "Colliery Guardian.")

The strict regulations in force in Great Britain compel great care in storing explosives in the neighbourhood of mines. Special buildings are provided, and much care is taken for their construction. A very good plan for such a structure, capable of holding 1000 to 4000 lb. of the material, is shown in Figs. 54 to 57.

The building must be substantially constructed of brick, stone, or concrete, or excavated in solid rock, earth, or mine refuse not liable to ignition. It is not permissible to have bare brick or stone walls inside; and a coating of cement on the inside walls is not much better, as, unless great care is exercised, the cement is apt to scale off and become a source of danger; though if the cement is painted, it may keep in fairly good condition. The best plan of all is to have the building lined, floored and ceiled with wood, all fastened with copper, brass or zinc nails, or with iron nails counter-sunk and puttied, and to apply two or more coats of paint or varnish to walls and ceiling. Double doors are necessary; both should open outwards, be fitted with strong locks, and with hinges inaccessible from the outside.



FIGS. 54–57.—EXPLOSIVES STORE.

It is essential that the building be ventilated, as a store is generally in such a situation that it is exposed to sun and rain, and unless free ingress and egress of air be provided for, many explosives will deteriorate in the moist atmosphere which such influences generate and maintain. The woodwork of the store is also likely to rot and require frequent renewal. Ventilators should be so constructed that access cannot be obtained by them to the contents of the store, and that malicious persons cannot insert combustibles which might cause fire or explosion. Precautions must also be taken to prevent water getting into the store, as it has injurious effects on every explosive.

In the drawings, Fig. 54 is a front view, Fig. 55 a cross-section, Fig. 56 a plan, and Fig. 57 a longitudinal section. The building is 6 × 6 ft. inside measurement, and 6 ft. 6 in. high. The external dimensions are 7 ft. 6 in. × 7 ft. 6 in., and 10 ft. 3 in. high in the centre. The specifications given below may be modified to suit local requirements:—

Foundation.—Site of building to be levelled, and subsoil sufficiently removed that a firm bottom be obtained; foundation to consist of hard-burned bricks or rubble well grouted with mortar.

Walls.—To be of best hard-burned bricks, 9 in. thick, and every fifth course to be headers. Gable ends to be carried up to the top of roof, and circled; outside walls to be neatly pointed.

Roof.—To be covered with galvanised corrugated iron 22 B.W.G., screwed to purlins *a*, 5 × 2 in., laid from gable to gable, and bedded in cement at gables and wall-plates; all joints and screw- or rivet-holes to be filled with white-lead.

Door-frames.—To be 9 × 2 in., fastened to dooks built into wall, and lintel 9 × 4½ in.

Doors.—Outside door to be 6 ft. 1 in. × 3 ft. × 2 in.; inside, 6 ft. × 2 ft. 8 in. × 1½ in., framed and lined; movable block *b* inside inner door 9 × 2 in., with fillets on each side, so that it can easily be lifted out.

Flooring.—Sleeper joists *c* to be 6 × 2 in.; flooring, 1⅛-in., tongued and grooved.

Lining.—The inside walls and ceiling to be lined with ⅝-in. plain white lining; fastened to fillets but not touching walls; with breaks in fillets at intervals to allow air to pass all around; and nailed to wall plates at top and flooring at bottom, with a fillet at top and bottom on all four sides; ceiling joists *d*, 3 × 2 in.

Lightning Conductor *e*.—To be of ¾-in. round iron, terminating 4 ft. above roof in three points, nailed against wall by iron holdfasts, and carried 6 ft. into the ground at an angle of 45° away from the building. But the best method of protecting a building against lightning, according to Sir Oliver Lodge (Rep. Brit. Lightning Committee), is to surround it completely with wire netting, suitably grounded. In the case of a powder magazine, this is quite practicable.

Ventilators.—One cast-iron air-box *f*, 9 × 4½ in., to be inserted on each side, with a dwarf wall 18 in. long opposite each, and a slate or slab same length on top just under flooring.

Locks.—All locks, hinges and nails, where exposed, to be of brass or copper, or counter-sunk and puttied.

Woodwork.—Outer and inner doors with their frames to be of best red pine; remainder of woodwork of white pine. All to be thoroughly well-seasoned and perfectly free from knots, shakes or blue-wood.

Painting.—Outside door and all exposed woodwork to receive 3 coats of good oil paint.

SHAFT-SINKING.

Style and Location.—Three recognised styles of shaft are in use, choice depending in part upon location in reference to the vein or bed to be worked. In metalliferous mining, operations are usually commenced on a prospecting scale, and the shaft is sunk on the vein, following its underlay, and proving it foot by foot; this would seem to be the best practice, as dead-work is thereby minimised. It very rarely happens that a prospecting underlay shaft can afterwards be converted into a working shaft which will be as satisfactory and economical in the end as a new shaft; so that when the prospects justify thorough systematic development, a decision has to be made whether the working shaft shall be of the vertical, the underlay, or the composite kind, involving its location in the vein, or in the country, or partially in both.

Various factors enter into the question, the most important being:—

a. The degree and regularity of the dip of the vein, and at what depth it will pass out of the property.

b. The hardness and standing qualities of the ground forming the vein and the country respectively.

c. The money and time at disposal for development work.

For proper judgment of *a* and *b*, considerable diamond-bore prospecting (see p. 167) is of the utmost value; but most undertakings are commenced with so much impatience and such a small margin of working capital that this is rendered impossible, and the only data available are such as may be gathered from the preliminary workings.

Of vertical shafts it may be said that, as a rule, they are more costly per foot sunk, because the ground traversed by them is almost always harder to drill in and tighter to shoot than a vein; but, on the other hand, they are generally more satisfactory in the matter of permanency, cost far less in upkeep, and sometimes have the advantage of exposing unsuspected parallel ore-bodies, whilst in purely deep claims they are virtually compulsory. Many instances might be cited where a vertical shaft in the country has had to be ultimately resorted to, because of repeated injury to the underlay shaft caused by slipping and crushing movements in the vein.

In coal and deep alluvial mining there is scarcely any instance of a departure from vertical shafts. They are the most direct means of access to the beds in the great majority of cases, give better hoisting facilities for immense outputs, and can be made more secure in wet and bad ground.

The underlay shaft is well adapted when the angle is reasonably constant, and its claims increase as the dip is more flat, because the length of cross-cuts is so much augmented in a vertical shaft on a flat vein. Thus, on the Rand, all the early vertical shafts have been replaced by underlays. But an incline shaft must either be sunk on a uniform grade, or the changes of grade must be made on very easy curves. The Rand banket beds favour the incline system, their angle ranging pretty gradually from about 80° near surface to 25° at several hundred feet, and, at a vertical depth of 800 ft., the dip scarcely anywhere exceeds 32° along the whole series. Thus, while the vertical shaft was justified at first, the underlay has very properly taken its place.

The easy sinking on the vein which is an argument in favour of the underlay shaft is not always an unmixed blessing. It is possible for this advantage to be practically nullified by the cost of timbering and maintenance. The author had an experience of this in an Australian mine. While sinking in the serpentine could be done at about one-third the cost per foot and at three times the speed, the total expense of sinking in the diorite was less in the end, owing to the close timbering and constant repair necessitated by the insecurity of the ground in the former case. Much the same thing occurs on the Mother Lode of California. Owing to soft swelling ground near the vein, the timbering requires close attention and frequent repairs. The dip of the lode is somewhat irregular, varying occasionally from 45° to 70° in a single section, and in trying to follow it, even approximately, there occur bends or marked changes in the inclination of the shaft. The ultimate results are great expense of repairs, and material reduction of the safe speed of hoisting; consequently, when depths of 1500 to 2000 ft. are reached, a new vertical shaft becomes a necessity.

Composite shafts, or a combination of vertical and underlay, are objectionable as a rule, because they always incur heavier wear and tear on the hoisting gear, and therefore show higher running costs. Yet there are situations where they commend themselves, as for instance in deep-level schemes on veins suitable for underlay sinking, where the conditions (such as non-ownership of land) prevent following the vein from surface; there is little doubt, however, that in almost all cases it would pay better in the long run to sink vertically near the dip boundary, breaking entirely by overhead work, and passing dirt by gravitation to the shaft bottom. Abrupt transition from vertical to incline is to be avoided in any event, a gradual curve being interposed. At a New South Wales gold mine, under the author's management, a composite shaft in use was so extravagant in fuel and ropes that he condemned it, and carried the underlay through to surface, a distance of about 300 ft. of driving, partly through loose and shifting ground. Such was the economy in running thereby gained that, while previously two boilers were kept under full steam, after the change the second boiler was not required, although two or three times as much hoisting was done; and the saving in fuel and stokers' wages paid for the work inside 6 months.

As to length of ground adapted for service by one shaft, no hard and fast rule will hold; but approximately 1500 to 2000 ft. can be economically served by a single working shaft, especially if the original prospecting shaft has been kept open for ventilating purposes. On the large Rand properties, 4000 to 6000 ft. long, several shafts are necessary, to afford accommodation to the enormous output, to minimise underground haulage, and to provide adequate ventilation; distances range from 1000 to 1700 ft. apart.

In opening a mine with a new shaft, there are many things to consider in connection with raising, crushing and stamping the ore, the main objective being to have one lift only, and sufficiently high at the shaft to permit of the ore gravitating to the several points for treatment. Often the shaft is cramped in size, which constitutes a drawback for all time; or else, in low ground, no attention is paid to raising the collar to a suitable height to meet contingencies for handling or tramming the ore. When temporary structures are being erected at a new shaft, it is conducive to great saving in the future if foresight be shown with regard to the permanent plant, and the main foundations for head-gear, etc. Temporary gear can be fixed on the same site, and the permanent work be built around the temporary, without having to suspend operations in the shaft, or do unnecessary and costly excavating.

With essentially deep level propositions, as on the Rand, the first condition is a shaft or shafts capable of exploiting and ventilating the whole mining area. One may assume two shafts (it is practicable to join them expeditiously when 3000 ft. apart) of a maximum depth of 5500 ft., and large permanent inclines extended on the average dip of the reef, but not following its faults and sinuosities. In connection with these main inclines it will also be necessary to have three subsidiary inclines, which need only be equipped with light hoisting machinery to command a depth of 500–1000 ft., and, at each 1000 ft. the engines can be lowered. Tramming to the main incline shafts from the subsidiaries can also be done mechanically. The subsidiary inclines, being flexible, could be placed to the best advantage in accordance with the mining conditions met with, and made to conform to faults and disturbances which could not be calculated upon in starting work from the surface. A large lateral area, connected with only two vertical shafts, is an insurance against the risk of encountering serious faults, dykes, disturbances, and barren zones, which, with limited lateral area, and shafts closer together, might spell ruin. The main shafts from surface might have 7 compartments : 1 for ladder-ways, air-pipes, electrical cables, etc., 2 hoist-ways for lowering men and material, 2 hoist-ways for winding ore and water, and 2 in reserve for improving ventilation, and in case it might be advisable to employ another winding engine. These reserve hoist-ways can at first be bratticed, and, in connection therewith an exhaust fan can be installed The artificial ventilation thus introduced will greatly aid the speedy connection of the two main shafts, and improve the ventilation of the whole development work, prior to the two shafts being connected and the establishment of ample winzes. When connection has been made, and thorough ventilation is established, a second hoisting engine can command these two extra compartments, if the mine requirements justify it; and, even if the compartments are not required for further hoisting, they will aid the natural ventilation, which is most important.

After reaching the reef, an independent equipment is adopted underground. The main incline shafts must have the same capacity for output as the vertical shafts; but, on an angle of 27°, double the load can be hauled with the same power as in the vertical shaft. The weight of the rope is also taken up by pulleys on the incline, so that in the distance on the incline (5350 ft.) the hoisting engines would only need to have power equivalent to hoisting 2429 ft. in the vertical with the same factor of safety. The arrangement of the main vertical shaft crosswise to the formation allows most advantageous planning of the underground work when independent hoists are used, but is disadvantageous if it is desired to continue sinking from the surface on the incline, as well as from the vertical shafts, which system can only be advocated for moderate depths.

Shape and Size.—All inclined shafts are of necessity rectangular. Vertical shafts for ore-winning are also almost invariably of this form, while coal-raising shafts are as universally circular. These latter are capable of withstanding much greater pressures, and can be sunk through and maintained in very wet and running strata, which conditions are not encountered in vein mining, though they often are in deep alluvial.

Dimensions of shafts are computed on the basis of the output of ore and waste, plus allowance for ladder-way, pump-column, air-pipes, electric wires, and so on. The accommodation for pipes and wires provided in the ladder-way is often so limited as to be very prejudicial to economic running, access to joints and fixings being so hampered that leaks and damages go unremedied and involve waste. Very large pipes and ample room are necessary when using Cornish pumps, even though the clack-boxes and H-pieces be arranged in chambers cut for the purpose.

Rectangular shafts vary in size from about $8 \times 3\frac{1}{2}$ ft. clear inside timber and lagging (divided into 2 hoisting compartments and a ladder-way) to 25×7 ft. (with 4 hoisting compartments). Vertical shafts on the Rand outcrop properties, reaching to about 1000 ft., are commonly 12 × 5 ft., giving 2 hoisting-ways, each 4 × 5 ft., and a ladder-way 3 × 5 ft.; those on the first row of deep levels, down to about 1500 ft., are 16 or 21 × 6 ft., making 2 or 3 hoisting-ways 4 or 4½ × 6 ft., and a ladder-way 6 × 6 ft.; and those on the second row of deep levels, 2000 ft. and more, are 26 × 6 ft., with 4 hoisting-ways 4½ × 6 ft. and a ladder-way 6½ × 6 ft. Sometimes, when a shaft cuts several veins, one or more compartments cease below the intersection.

On the Rand it has been customary to put down rectangular shafts with their longest axis parallel with the strike of the reef, so that if necessary any or all of the compartments may be carried on on the underlay of the reef in the same plane. Pettit considers it probable that when the very deep shafts are sunk, each property will only have one, owing to the immense outlay of capital requisite for sinking. The properties will be longer along their line of dip than along their strike. Ventilation will be effected by a connecting drive between two shafts of adjoining properties.

A typical shaft in the first row of "deeps" has three winding compartments (2 for rock, 1 for men and tools), and a pump and ladder-way. Two such shafts are easily capable of supplying a 200-stamp mill pulling 6 days a week, and allowing a margin for most emergencies and break-downs. In the second row of "deeps," in almost every case, a second man-compartment is added, the extra capacity being required to more readily handle the shifts, and to give more economic hauling by balanced winding in the auxiliary compartments. The third row of "deeps" will have 4 hauling ways of somewhat larger dimensions than the second row, with a pump and ladder-way. In the very deep shafts to be sunk in the future, it has been suggested to set out 7 compartments —1 pump and ladder-way, 8 ft. 6 in. × 6 ft. 6 in., and 6 hauling compartments (each 5 ft. × 6 ft. 6 in.), giving a shaft 42 ft. × 6 ft. 6 in. in the clear. It would seem better to enlarge the rock compartments from 4 × 6 ft. to 5 × 6 ft., or even larger, as in so narrow a width a great portion of the useful area is taken up by the guides and necessary clearance for the skips. Thus, it is usual in a 4 × 6 ft. compartment to put in runners 4 × 5 in., or better still, 4 × 8 in., reducing the original 4 ft. to 3 ft. 4 in., whereas in a 5 × 6 ft. compartment the guides would be 5 × 6 in., or 4 × 8 in., giving a width of 4 ft. 2 in. or 4 ft. 4 in. in the clear. Taking into consideration the skip sides and frame, and the necessary clearance, the available area in the usual form is 44%, and, in the proposed modification, 56% of the whole. (A. E. Pettit.)

Incline shafts do not materially differ from these several proportions, the mean size on the Rand being 16 × 5½ ft. inside, affording 2 hoisting-ways 4½ × 5½ ft., and a ladder-way 6 × 5½ ft.

Though unusual in metal mining, circular shafts are sometimes adopted, and there are notable instances on the Rand, where

L. Hamilton preferred them as being cheaper to sink and timber, stronger, and better for ventilation. Examples are shown. Fig. 58 is 900 ft. deep, 15 ft. diam., and 176 sq. ft. in excavation area; it has 4 hoisting-ways, *a b* and *c* each taking a 3-t. skip 4 × 3½ ft., and *d* a man-cage 5 × 4 ft.; and 4 (convertible into 8) pipe-ways for holding air-, steam- and water-conductors; the timbers are *e* 8 × 10 in., *f* 6 × 9 in., *g* 6 in. sq.; *c* is the sinking compartment. Fig. 59 is 400 ft. deep, 11 ft. diam., and 95 sq. ft. in area; it has 1 hoisting-way carrying a 2-truck cage *a*, held by 40 lb. rail guides on the 6 × 9-in. timber *b*, and a single cage *c* controlled by 3 guide-ropes *d*; the space *e* accommodates pump-mains, etc. A masonry curbing runs round the top, and is continued as far down as the insecurity of the ground demands; but once in solid rock, the shaft walls are self-supporting, and the only timbering needed is to make subdivisions and carry the cage-guides.

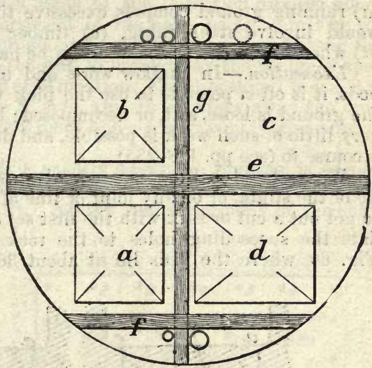


FIG. 58.—CIRCULAR SHAFT.

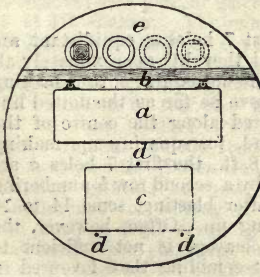


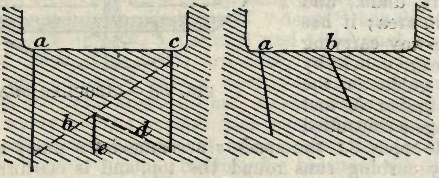
FIG. 59.—CIRCULAR SHAFT.

H. F. Marriott is of opinion that the adoption of circular shafts would obviate to a very great degree difficulties in ventilating the unconnected deeper level mines. They possess the vital advantage of not requiring a stick of timber, in some classes of ground where the strata approach the horizontal rather than the vertical. Such shafts would be used only for hauling rock, with skips running in rope guides. Great speeds could be attained with a minimum risk of accident, and the cost of the shaft and its upkeep would be less by the amount of the timber bill. Rectangular shafts would also be included in his scheme, for the purpose of transport of men and materials, and for pump, cable and ladder-ways.

Disregarding ventilating area, the working space in a circular shaft is 35–45% less than in a rectangular shaft giving the same hauling facilities. But circular shafts are distinctly superior when (*a*) running ground compels excessive timbering, (*b*) heavy water would involve steel lining, (*c*) timber is scarce or costly, and (*d*) when brickwork or concrete can be had at reasonable figures.

Excavation.—In shallow work and in traversing sedimentary beds, it is often possible to use the pick with advantage, or where the ground is loose, soft, or decomposed; but in really deep sinking, very little of such work is possible, and drilling for blasting is had recourse to (see pp. 198–235).

Where hand-drilling is used, advantage may be taken of the lie of the strata, or of any joint or line of fracture in igneous rock, to get out a cut or sink with the first set of holes, and to accommodate the succeeding holes to the rock which is left. Thus in Fig. 60, where the beds lie at about 36°, and the excavation is



FIGS. 60, 61.—DRILL-HOLES IN SINKING.

18½ ft. long and 8 ft. wide, the first 7 holes *a*, equidistant, are put down about 3 ft., and when fired they lift approximately all the ground between the surface and the dotted line *b*, thus making way for the next 7 holes *c*, which remove as far as the dotted line *d*; finally about 4 holes *e* are required along the centre of the bottom, and about 2 more at each end, for squaring up, making 22 in all. Or, as in Fig. 61, 24 × 8 ft., the first 7 holes *a* are drilled deep, and are supplemented by a second row *b* numbering only 3, these 10 forming the sink; after blasting, some 14 to 16 other holes are necessary for squaring up. Often, however, the help to be gained from a natural cleavage is not sufficient to merit its recognition, and a system resembling that favoured in machine-drilling is followed, only that the excavation is terraced or stepped, the sink being always in advance, and some drilling being done on each shift. In hand-drilling, the depth of hole mostly varies between 3 and 5 ft., the latter being about as much as can be achieved economically.

The area occupied by a pair of men in sinking with hand-drills is not less than 20 sq. ft., if the full value of their labour is to be reaped.

On the Rand, the general consensus of opinion trends to hand-sinking. Various methods have been tried, as in the Robinson Deep, where advantage was taken of the formation dipping about 36° S., the general arrangement of the holes being as in Fig. 62. A row of holes, 1 to 7, each about 3 ft. deep, were placed along the S. side of the shaft, these tending to break away the wedge-shaped piece of rock made by the stratification; then another 7 holes, 8 to 14, were placed along the N. side, and these broke away the bulk of the remaining rock, a few more holes and "pops" squaring up the round or "sink."

The following method is usually adopted. Sinking is carried on with 3 consecutive shifts of 8 hr. each, with 1 white man and 40–56 Kaffirs on each shift. At the commencement of the shift, the white man wedges a plank at each end of the shaft some 7 ft. from the bottom, and starts some of the Kaffirs drilling squaring holes at each end of the shaft, thus giving them the whole shift to drill. He then cleans up the dirt from the preceding shift with the remainder of the Kaffirs, taking care to remove (with pick, or gad and hammer) every piece of rock which is at all shaken. The sump is carried at right angles to the strike, and at about the centre of the shaft, thus forming benches, and giving larger drilling area. On the average, 20 holes are fired per shift, varying in depth from 4 ft. 6 in. to 5 ft., with the exception of the end holes referred to, which are usually 6–7 ft. All Kaffirs drill double-handed, giving one hole to two workmen. Toledo steel of $\frac{7}{8}$-in. octagonal section has given the best results. Fuses are left 8 ft. long, and 8–9 sticks ($1\frac{1}{2}$–2 lb.) blasting gelatine are put in each hole. In order to prevent "fitchering," or seizing of drill, plenty of clearance is given, the starting drills being $1\frac{1}{2}$ in. No inconvenience is caused by the fumes from blasting, one shift following the other immediately; and no artificial ventilation has been found necessary, except bratticing off the pump and ladder-way, thus giving a natural path for the descending and ascending air. In the very deep shafts, air is sometimes blown down, so that, when the last hole of the previous shift has been exploded, the next shift immediately descends and meets the ascending fumes as they go down; the time occupied in passing through this zone of bad air is so short that no inconvenience results. (A. E. Pettit.)

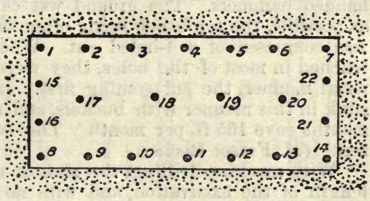


FIG. 62.—HOLES IN HAND-SINKING.

The subdivision of an 8-hr. shift in a deep-level shaft is:—

Average hours drilling per shift	.. ..	3·07
,, ,, winding rock	.. ..	3·26
,, ,, ,, men and tools	.. ..	1·17
,, ,, blasting	.. ..	0·50
		8·00
Average number of holes per shift	.. ..	20·31

(R. M. Catlin.)

The sinking of the Jupiter East shaft (28 × 8 ft.) was worked in 3 shifts of 8-hr. each, 40–50 "boys" being employed on each shift. Each shift, when going on, cleaned out the ground broken by the one preceding it, and then commenced drilling with double-handed hammers. The ground was carried in as many benches as possible, the top bench being often 20 ft. above the sump, which was composed of a 4-holed cut. When an average of 3 ft. was reached in most of the holes, they were charged and fired in the usual manner, the cut coming first. In one month, 211 ft. were sunk in this manner with buckets, and an average taken over ten months gave 165 ft. per month. The conditions were almost perfect. (H. Fraser Roche.)

With machine-drilling, the sink is made about central in the length of the excavation, and with no reference to the stratification; 2 to 4 stretcher-bars are rigged across the shaft at suitable intervals, each bar sufficing for the drilling of 4 holes on each side of it. Sometimes 2 machines are mounted on one bar. The central two rows of holes are inclined towards each other, so that they meet at the bottom, forming the sink proper; the next succeeding row on each side is also inclined, but not quite to the same degree, and extends the cut; while the final row at each end is vertical. The central compartment of the shaft is in this case the sinking compartment, which is a usual arrangement. Sometimes the sinking compartment is not central, and then the cut is made towards one end in conformity. A common depth for holes is 6 to 7 ft. The system of firing varies widely. The sink may be taken out first alone, followed by the next rows, and finally by the outsides, squaring up the bottom after each round; or the sink and followers may be fired together, and cleared away before charging the outsides. The aim is to do all that is possible with the drills when once they are rigged, so as to minimise the loss of time in mounting, dismounting, raising out of the shaft, and lowering back to the work. A contingent advantage of such a regular sequence of operations is that it permits a segregation of the labour, drillers being employed on one shift and shovellers on another. With machines, 6- and 7-ft. holes are counted upon, unless the ground is exceptionally bad for shooting. Electric firing is approved by some and objected to by others (see p. 230).

In sinking on the underlay, when the bedding is helpful, rapid progress can be made by hand. First a cut is made by almost vertical holes along the bottom of the face, and these prepare for successively less inclined series, till the back holes are parallel with the stratification. Machine-drills are rigged to make a central cut across the short width, and the terraces are afterwards drilled and shot towards the cut.

In a vertical shaft (32 × 9 ft.), experiments at the Village Deep point to 12 machines, a net gain of 1 hr. per shift being recorded by 12 against 8 machines. These are run by 6 whites and 24 natives, the arrangement of holes being as in Fig. 63. Holes 5, 13, 15, 22, 24, 32, may often be dispensed with, depending on the ground to be broken. In elevation, the sumps of holes 16, 17, 18,

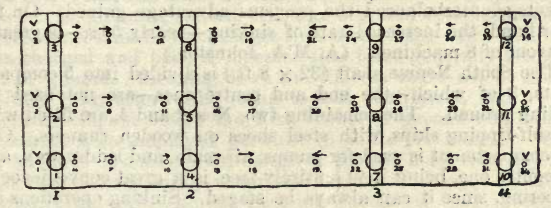


FIG. 63.—VERTICAL SHAFT (32 × 9 FT.), SHOWING POSITION OF BARS, MACHINES AND HOLES.

should practically reach sumps of holes 19, 20, 21, and the sump of a hole such as 14 should be within 3 ft. of the sump 17. If this is carried out, no trouble is experienced as regards blasting the timber. (H. Fraser Roche.)

At the Cinderella Deep 5-compartment shaft, 4 measure 5 × 6 ft. inside timbers, and the other 6 × 6½ ft. inside. The last is used for ventilation, and is boarded up to within 400 ft. of the bottom. No hindrance was caused by insufficient ventilation; the men were able to start work again 15 min. after blasting. Tests were made to determine the various speeds and costs by using 4, 6, and 8 machines. With 4 machines fixed on 2 bars, 3 whites were employed, one to superintend the natives shifting broken rock and drilling a few extra holes or easers. The method of working adopted was the "sump and bench," and 99 ft. per month were averaged. With 6 machines on 3 bars, 4 whites were employed, and a few extra "boys," but it was found that the footage was very little improved. About ¾ of the shaft was drilled over by machines before blasting, and any holes in addition, to help these, were put in by hand. Then 8 machines were tried on 4 bars, and 6 whites were in charge, the whole bottom being drilled over with the machines. This system was in vogue for only 2 weeks, but the speed attained was 138 ft. per mo. The costs with 4 machines (labour, explosives, stores, cleaning up, hauling, and hoisting) were 227*s.* 8*d.* per ft.; with 6, slightly more with no additional speed; with 8, 246*s.* 7*d.* With 4 machines, the time taken to blast the whole bottom and clean up was 57 hr.; with 8, 42 hr. An advantage gained by the 4 machines was in the quickness with which they could be set up, the bench forming the first point of attack. Also the holes invariably broke to the bottom, and it seemed that this method was safer, and did better work, the advantage of hand labour being apparent in squaring up. When working 8 machines, 4 buckets were required for hoisting, thus bringing into use 2 engines; these 4 buckets could not be kept going all the time, and the loss incurred through the temporary hanging up of one of the buckets counterbalanced the previous advantage gained. On the other hand, the increased rate of sinking—nearly 50%—is greatly in favour of 8 machines. (A. M'A. Johnston.)

The South Nourse shaft (32 × 8 ft.) is divided into 5 compartments, 3 of which—the end and centre ones—are not used for hauling ground. The remaining two, Nos. 2 and 4, are fitted with 2-t. self-tipping skips, with steel shoes on wooden runners. One end compartment is used for pumps, air-main, and ladder-way, and the centre one, being kept entirely free, is a great convenience in timbering, since it can always be staged. Sinking operations are conducted as follows: 6 small machines (2½ in. diam.) on 6 bars are in use, each tended by one white with native or Chinese helpers. Some 40–50 holes, 4–5 ft. deep, are drilled over the bottom of the shaft, their position, of course, depending on the state of the ground to be broken. These are then blasted, and cleaning begins as soon after as possible, the first skip of ground being hauled in 30–45 min. About 10–14 lashers are employed per shift, which varies in duration, but two shifts are usually sufficient for each round. The timber is similar to that ordinarily used in deep-level shafts, viz. wall-plates, dividers, end-plates, and studdles. The wall-plates are mortised to receive the studdles and dividers, each divider effectually blocking two studdles at each end. The whole sets are blocked in the ordinary way with the sides of the shaft. The use of skips necessitates the timber being kept within 30 ft. or so of the bottom of the shaft. The bottom wall-plates are cleated with sheet iron, to protect them against the blasting, but these are capable of rapid adjustment and removal. Two sets of wall-plates are lowered in quick succession beneath the skips, and the hanging is operated from the last blocked set, and the bottom of the shaft. The runners are kept as close as possible to the bottom set in position, and, below this, temporary runners in 6 ft. lengths are used, thus allowing the skips to reach the bottom without the shoes leaving the runners. Bearers are placed at suitable and regular intervals. The proximity of the timber to the bottom of the shaft is of great value in rapidly removing the smoke and gas after blasting, since ventilating pipes can be carried down to the bottom set and connected with a fan.

At the Village Deep western shaft, buckets are run in cross-heads with steel shoes on wooden runners. The tipping process with these buckets is far inferior to the skip method. The 3 centre compartments are used for hoisting, 1 for pump, air-main, and ladder-way, and the remaining one is kept clear. The sinking is carried on as follows: 4 bars are rigged—2 at 1 3 ft. from the end of the shaft, and the others 7 ft. from each of these; 3 3¼-in. machines were used on each bar, each side machine drilling 3 holes, the centre ones drilling 3 also, but sometimes one of these was omitted when the ground allowed. These were tended by 8 whites with native helpers. The cut holes are drilled 10 ft. deep, and the side 8–9 ft., all being started with star bits, and completed with a succession of chisels, the average time taken being 8 hr. All were then charged and blasted together, this blast generally breaking 4 ft. of ground. Cleaning down the timbers could not be started for fully 1 hr. afterwards, owing to bad ventilation, and 1½ hr. generally elapsed before the first bucket of ground was hauled. 18 lashers were employed per shift (3 shifts per round). When the 4 ft. was cleaned up, the sumps of the old holes from the first blast were cleaned out by compressed air, and all charged again and blasted, after which, similar cleaning operations were proceeded with. The average depth per round cleaned was 7·5 ft., 20 such rounds being drilled, blasted and cleaned up during the month. The average time per round was 37·4 hr., 4200 t. of ground were hauled, and ·96 lb. dynamite was used per ton of ground hauled. (H. Fraser Roche.)

In the Davis pyrites mine, Mass., a V-cut is used in the 18 × 9 ft. shaft, 4–5 holes being needed in a round; 60% dynamite is employed in the cut (Nos. 1 and 2 rounds), and 40% in Nos. 3, 4, as cleaning up is more rapid with lumps than with fines. Holes are 5–7 ft. deep. (Rutledge.)

At the Alaska Treadwell, it has been proved that 3⅝-in. machine drills do 30% more work than 3¼-in., and that 10-hr. shifts are better than 8-hr. The centre-cut system is used: the cut is first drilled, fired, and cleaned up; then the other holes are similarly served, except that the end holes are reserved and fired with the cut holes of the next round. (Kinzie.)

In sinking the incline shafts of the Lake Superior copper region (about 38°), generally the men are working under protection of a rock pentice, 6–7 ft. thick, by starting from each new level with a small sinking shaft 5½ × 9 ft. (instead of the full 17 × 9 ft.), in line with the man-way of the finished shaft above. Only 1 drill is used, even in the full-sized shaft, the holes, arranged to take advantage of the conditions, being spaced as in Fig. 64 at the ends, but, at or near the middle, they are drilled on both ends and slope towards the middle. (W. R. Crane.)

Removing Dirt.—When there are no vacant stopes into which the broken rock can be run for stowing, it must all be hoisted to the surface.

For shallow depths and small shafts, a kibble or bucket suffices, two being in use at a time, one filling while the other is travelling. A good kibble should be about 4 in. diam. larger in the centre than at the top and bottom, and the curve should be gradual from top to bottom. A convenient size is about 3 ft. high, 22 in. diam. at the bottom, 23 in. diam. at top, and 27 in. diam. in the centre. When the shaft gets down to any considerable depth, gates or crossheads should be used. These slide up and down on the runners, and as the kibble leaves the last set of timber, the gate encounters a block on the runner, which holds it until the kibble comes up again, the rope in the meantime passing through two holes in the top and bottom bars of the gate. For a gate to be successfully used, the poppet legs must stand at least 20 ft. above the landing stage where the kibble is to be tipped.

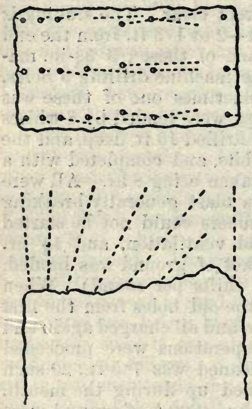


FIG. 64.—SINKING INCLINES.

In deep work, and in operations on a large scale, a self-dumping skip is imperative; but it entails the timbering being kept within about 15 ft. of the bottom of the shaft, and is useless without rapid winding. It may hold $1\frac{1}{2}$ to 3 t. as against the $\frac{1}{2}$ t. carried by a bucket or kibble; and may be equipped with safety-catches in the same manner as a cage (see Hauling and Hoisting).

When 3 buckets are in use (1 filling while the others are travelling), dirt can be removed more quickly than by any other method.

On the Rand, in a 6 × 4 ft. compartment, a round bucket of 17 cub. ft., or an oval one of 20 cub. ft., is used; in a 6 × 5 ft., one of 50 cub. ft. is customary.

At the Alaska Treadwell, the bucket is flat-bottomed, with the sides projecting 2 in., thus giving it a more secure base, and permitting it to remain upright on an uneven surface while being filled.

In the incline shafts of the Lake copper mines, the dirt is handled by a bucket operating on an aerial rope system, and raised by an air winch. The bucket has a capacity of about $\frac{1}{3}$ t. At the station, its contents are dumped into a skip standing on the main track. Owing to the low pitch of the shaft, skidding on timbers is rendered impracticable, as only a small part of the bucket could be filled, even for very slow hoisting. The arrangement is shown in detail under Hauling and Hoisting.

Lighting.—The question of light at the bottom of a shaft is always vexatious, owing to the water and blasting; but this has been solved at the Alaska Treadwell by using a group of 36-cp. electric lamps in the place of torches or candles. The wire is brought down the shaft to some convenient point, where a small reel is placed, with which the lamps can be lowered or raised at will. It has been found that the best wire to use is the ordinary lamp-cord, wrapped with heavy canvas, and given a good coat of P. and B. paint.

At Raub, Malaya, the author used 2 32-cp. lamps in a vertical 18 × 5 ft. shaft.

Unwatering.—Nearly every shaft makes more or less water. Very often this is due entirely to local sepage from the loose surface soil, and one of the first cares should be to shut it out, whether by clay pugging, iron caissons, or solid masonry, as the circumstances demand. When the volume is small, a bucket or an automatic baling-tub can effectively cope with it; as it increases, some form of pump becomes necessary. The constant repetition of lowering the pump into place and drawing it up out of the way of shooting involves in the aggregate a great waste of time and of money, and, where the flow is really considerable, the drawbacks to pumping by way of the shaft become exaggerated. A remedy for this which has been successfully adopted by G. C. McFarlane (En. & Min. Jl., Apr. 7, 1900) in sinking a coal shaft which traversed beds of water-bearing gravel and sandstone, giving a flow of 200 to 300 gal. a minute, is to bore two holes to the full depth of the intended shaft, one (*a*) from the bottom of the shaft, and the other (*b*) from surface outside the shaft, connecting them at bottom by enlarging each hole with a suitable tool (called a "reamer"), or by repeated charges of dynamite. Each hole is cased with say 8-in. pipe, that in *b* being contracted about an inch at some 6 in. above the bottom. The pump-barrel consists of a 16-ft. length of 7-in. gas-pipe with a leather seed-bag in the lower end, which is dropped into the bore *b* and rests on the contraction of the casing. The pump-bucket is a line of 6-in. gas-pipe, reaching from the bottom of the pump-barrel to about 8 ft. above the ground level, and fitted at the lower end with a steel valve-box and ball-valve, the outside diameter of the box being $\frac{1}{16}$ in. less than the inside diameter of the pump-barrel. The "bucket" carries at top a T for delivering the water, and a swivel for attachment to a rope which is made fast on a drum at the other end, and, by the action of a crank and pitman tightening and slackening this rope, the bucket is alternately raised and lowered. At 50 to 70 strokes a minute and 2 to 4-ft. stroke, 200 to 400 gal. a minute was raised; and rock fragments and gravel up to $1\frac{1}{2}$ in.

cube gave no trouble. But the power necessary was a 70-h.p. engine for a lift of 200 ft.

It would seem that in some cases equally good results could be got at less cost by boring a single hole alongside the shaft, casing it, equipping it with an air-tube and a water-tube, and forcing the water up by compressed air, without any pump or other mechanism, in the same way as the Bacon air-lift pump is applied in Germany and America. A 10 × 12-in. compressor at 90 rev. and 60 lb. steam will raise about 130 gal. a minute to a max. height of 120 ft., and in most metal-mining operations an air-compressor is part of the equipment (see Pumping).

When the volume of water is excessive, as in coal and salt mining sometimes, sinking cannot be carried on till the ground has been frozen by passing a freezing mixture down pipes let into bore-holes around the shaft, and maintaining the ground in a frozen condition until the water has been shut out by solid masonry.

Timber.—As it would be inconvenient to devote a separate paragraph to timber under each section of mine work, the subject is discussed here in a general way, not confining observation to such timber as is used in shaft-sinking only.

Resistance to Pressure.—It is an established principle in mining that timber presents far greater resistance to pressure in line with the grain than across the grain, and this is availed of wherever possible. Rupture of timber by the incidence of end-pressure may ensue from three causes—buckling, crushing, and shearing. A number of trials made by Prof. H. Louis in 1898, and by others, lead to the following conclusions:—

Buckling (embracing any form of break that commences by a bend, whether the action be slow or rapid) is the most usual form of breakage, even in the testing machine, whilst in actual practice it is even more common. Tests gave an average of 63% broken by buckling, 25% by crushing, and 12% by shearing. It is obvious that in ordinary mining practice, where the ends of posts are not carefully squared, and where the axis of the post is but rarely exactly parallel to the direction of pressure, an even greater proportion must yield by buckling. When a heavy wind-shake exists, there is great tendency to yield by buckling at right angles to that wind-shake, which then becomes a neutral plane; hence one large wind-shake has less effect in weakening than might at first have been supposed, especially when it coincides in direction with the greatest diameter. A post with several wind-shakes is, however, decidedly weakened, and is apt to yield by crushing.

Crushing can result only when pressure is applied to the ends of a stick which is perfectly straight, and the ends of which are at right angles to its axis, while the material is symmetrically disposed about the central (vertical) axis, so that each portion is subjected to equal stress, and there is no tendency to any deflection from the vertical.

Shearing is the least usual mode of yielding, and the condition under which it occurs seems to be chiefly that the fibre has been more or less injured, so that the leverage which is necessary to split by crushing cannot be exerted—owing to the lesser length of sound fibre available; and it appears to be more apt to occur in timber containing a large amount of moisture, either natural or artificial, which may act as a lubricant between the fibres, and assist them to slide one over the other.

Any considerable difference in the strength of opposite sides of a stick is highly injurious to its strength as a whole. Such differences may often be caused by large knots in the wood, and still more by the gouge-marks used as brands; these often penetrate to a depth of over ½ in., and have proved to be a cause of great weakness, a piece of timber so marked almost invariably failing through the gouge-cut.

In ordinary mining practice, it may be taken that the length of a timber is usually between 10 and 15 times its diam., 12 being an average proportion. In the tests, the ratios of length to minimum diam. were varied between the extremes of 3·1 and 20·4, without any effect upon the strength, calculated to the sq. in. of area of the top or small end. There appears to be no relation whatever between strength and ratio of length to diam. In practice it is generally observed that a larger proportion of long timbers break than of short ones, probably due to the fact that they are naturally set in places where pressure is greater than in narrow seams; and, as the strength of a post is that of its weakest part, a long one presents a greater probability of including some spot of especial weakness than does a short one, this probability of failure being dependent upon absolute length, and not upon ratio of length to diameter.

Experiment showed that a timber which has once been subjected to strain, even though no injury be discoverable, is almost 15% less strong than before use.

Slowly accruing stress is appreciably more destructive than rapid pressure.

The breaking weight per sq. in. of area is pretty constant for all sizes of same wood and similar seasoning.

Round timbers are stronger than square-sawn pieces, in which the grain of the wood has been cut and weakened by the saw. Used in the mine, round timbers are less easy to handle and to align properly, and it is impossible to satisfactorily reinforce sets framed from such timbers by the usual false sets or pieces. The bark should invariably be removed from round timbers, as it collects moisture and fungus, and thus hastens decay.

Age seems to affect strength, older sticks being stronger in proportion to their area than younger ones. Coarse grain or wide spaces showing between the annual rings is not proof positive that a timber is lacking in strength, unless the growth between the rings is of a spongy nature and easily picked out. When pine timbers have been exposed to the action of the weather for any considerable length of time, they become what is termed "season-checked." This does not impair the strength when the checks are parallel with the grain; where the checks cut across the grain, the timber is weakened. Pitch seams have much the same effect as season-checks. Tight knots do not matter in stuff 8 × 6 in. and upwards, but large loose or rotten knots are condemnatory. The size of tree producing the lumber is not important, so long as the grain runs lengthwise.

Seasoning and Pickling.—All mine timber should be seasoned by at least 6 months' exposure in stacks allowing free circulation of air. Painting the ends is a potent preventive of checking.

Numerous means have been tried to prevent timber from being attacked by cotton-mould fungus, such as letting water trickle constantly down it, steeping in salt brine, charring surface with fire, whitewashing, creosoting, etc.

At many mines where a mass of close timber is used, involving great risk from fire, it is coated with zinc chloride and white-washed. Very much more satisfactory results, however, are obtained by injecting the preservative, using a solution containing the equivalent of ½ lb. dry zinc chloride per cub. ft. of timber.

In wet mines, creosoting is most effective. The timber is placed in a wrought-iron cylinder, with one end spherical and the other consisting of a door turning on hinges. When closed, this is made air-tight by screws and clamps. The air is then exhausted from the cylinder, and also from the pores of the timber, by means of an air-pump, a vacuum of 9 to 12 lb. per sq. in. being formed. From an adjacent tank, by means of a donkey-pump, creosote is forced into the creosoting tank, and continued until the pressure equals 100 lb. per sq. in. Fir, pine, etc., absorb 10 to 12 lb. of creosote per cub. ft., oak and other hard wood, about 6 lb. A second seasoning should follow the pickling.

The application of an antiseptic slightly reduces the initial strength of the timber, but this is soon equalised by the arrest of decay. It is well to bear in mind that there are vast differences in the qualities of creosote.

Kinds.—There is generally a disposition to use locally-produced timber for mining purposes, where any is available, because of the heavy transportation costs to most camps; but this is not always wise.

In England, the consumption is principally coniferous woods of Scandinavian growth, such as white-wood, red-wood, Scotch fir, and larch. At a conservative estimate, their resistance to crushing is fully 3360 lb. per sq. in., Scotch fir being perhaps the weakest.

In the Harz, for permanent timbering of levels, oak is used wherever possible; it lasts for 10 to 15 years, whereas pine lasts only 4 or 5 years; so that, in the long run, oak is more economical. It is found, however, that where there is much water, pine is better. For temporary timbering of stopes, pine is almost always used.

It has been lately observed, however, that acacia, which grows readily on the shale-tips, spoil-heaps, and waste ground generally in Belgium, is as strong at 25 to 30 years as Scotch fir at 50, and it is undergoing trial in some of the State mines.

In the United States, the common "bull" pine and the well-known Oregon pine are very largely used. Some tests made at the Californian University afforded the following instructive comparisons:—

	Oregon. lb. per sq. in.	Bull. lb. per sq. in.
Modulus of strength at elastic limit	7,750	4,500
Modulus of strength at rupture ..	10,130	5,460
Modulus of elasticity	2,449,200	1,194,200
Tension tests	17,049	5,332
Crushing endwise	5,849	3,335
Crushing across grain, 3% of height	741	598
Crushing across grain, 15% of height	1,048	808
Longitudinal shear	574	327
Proportion of moisture .. .. %	25·03	23·28
Weight per cub. ft. .. .. .. lb.	40·71	31·00

The results prove that the additional cost of Oregon pine would be more than made up by the decreased size of timbers needed to stand a certain strain. In many mines, 14 × 14-in. sawn Oregon timbers are now used instead of round bull pine 22 and 24 in. diam.

South Africa is dependent on imported timber, and uses chiefly Oregon, though some of the denser and stronger Australian woods are coming into favour in the deep mines.

Australia has a great variety of good lumber from the gum-trees (eucalypts), several of which, such as iron-bark and yellow box, are equal to oak in strength and durability, but somewhat heavier.

The best timber trees of Tropical America (Guiana, Venezuela, etc.), are greenheart, wallaba, mora, and bullet-wood; most other kinds decay rapidly, though they make good fuel.

Substitutes.—In many collieries, steel framing has come into successful competition with timber, a great advantage being its non-inflammability; but it is only in the last year or two that metalliferous mines have adopted it. In many cases, the corrosive nature of the water would be prohibitive. At least one large Colorado gold-mine (the Portland, Cripple Creek) in 1898 had recourse to steel posts for its large 3 compartment shaft, a considerable reduction in cost, and increased longevity, being anticipated.

Timbering.—Every shaft in which machinery is to operate requires to be more or less timbered, to afford secure fastening for the various gear, to maintain the shaft of true shape against compression or movement, and to give protection from falling particles of loose material. The amount and style of timbering vary very widely. The simplest form consists of a mere lining of poles or planks forming only a shield to prevent matter shed from the walls falling down the shaft; it is applicable solely to small prospecting shafts.

Vertical Shafts.—In single-compartment shafts, if the ground is hard and sound, timbering may be almost dispensed with, as the cage or skip can be guided by anchored steel-wire ropes, and an occasional stretcher suffices for carrying air-, water-, and steam-pipes. In a rectangular vertical shaft of more than one compartment, there must be timber subdivisions; to hold these, frame-sets are needed at short intervals. If the ground be loose, lagging or lining-boards must be added.

Conditions which determine timber dimensions are the nature of the ground (whether firm or loose, and whether any crush, creep, or twist is anticipated), the strains likely to be created by the size and speed of the cages and skips, and the weight of pump-column, air-pipes, etc., to be borne. Whatever the dimensions chosen for the square-set timbers, they should always be supplemented by bearer-sets at every 40 to 100 ft. These bearer-sets, as their name implies, are designed to carry the weight of the frame-sets in series. They are of stouter timber, and they project lengthwise beyond the other sets, being let into hitches cut in the solid rock. Lighter bearer-sets for carrying ladders should be placed at every 20 ft. The frame-sets are inserted at intervals of 4 to 6 ft., and consist of wall-plates (embracing side-plates and end-plates) encircling the whole shaft: corner-posts or studdles connecting one frame to the next; and girts, struts, braces, or dividers for partitioning the shaft into compartments.

Frame-set timbers range generally from 6 × 6 in. to 12 × 12 in., few cases requiring the latter dimensions. Oregon and pitch-pine are mostly in favour, being easy to work and light in proportion to their strength; kauri and various eucalypts are general in Australia, and though more costly to cut and exceedingly heavy to handle, their durability far outweighs these considerations.

The ends of side-plates and end-plates are halved and lapped to form a joint, the side-plate always supporting the end-plate. Fig. 65 shows a usual style of cutting the lap; the timbers are 10 in. wide and 12 in. deep; the side-plate *a* has a piece 10 × 9 × 6 in. removed; the end-plate *b* is served in a corresponding manner, but in addition a slice 12 × 1 in. is taken off the inner side, leaving a shoulder *c* 1 in. wide, which engages the inner side of each side-plate. In this case, the narrow way of the timbers takes the lateral strain of any crush of the shaft walls; but sometimes the broad way is opposed to this, as being stronger, though the resisting power of the struts should be all that is necessary. Fig. 66 represents another way of making the joint, which is distinctly superior, in that it provides accommodation for the studdles or corner-posts; the side-plate *a* is not only halved at

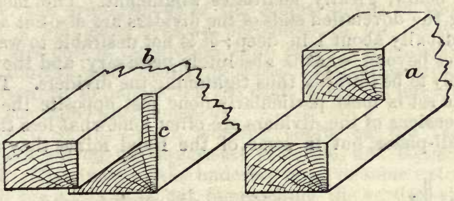


FIG. 65.—FRAME-SET JOINT.

b, and bevelled at *c* (instead of the shoulder *c* in Fig. 65), but also has ½ in. removed at *d*; so also the end-plate *e* is halved at *f*, bevelled at *g*, and checked at *h*.

The timbers of each set are cut to gauge with great accuracy, at surface, care being taken that when dimensions vary, as they almost always do somewhat, the variation does not affect the inside measurements. Each series of sets is made to depend from the bearer-sets, the first of which is the collar-set of the shaft. In getting frame-sets into place, a side-plate is first swung down and suspended from the bearer-set by hangers. These are bolts of ¾ to 1¼-in. round iron, hooked at one end, and threaded at the other for some 6 in.; they are used in pairs, and are of a length to give about 4 in. margin over half the outside height between two sets. Through holes bored towards each end of the side-plate, a pair of these bolts are passed, an iron washer interposing between the timber and the nut which tightens the bolt; a similar pair of bolts hang from the next side-plate above, and the two are united midway by the hooks. When the side-plates have been hung in this manner, the end-plates are laid in position on them, the studdles are inserted in the corners and at the ends of the dividers, these latter are dropped in, and the set is then wedged firmly in position, and trued by plumb-line and straight-edge, the hangers remaining as additional stays till the

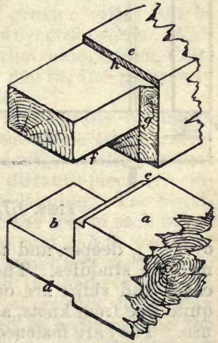
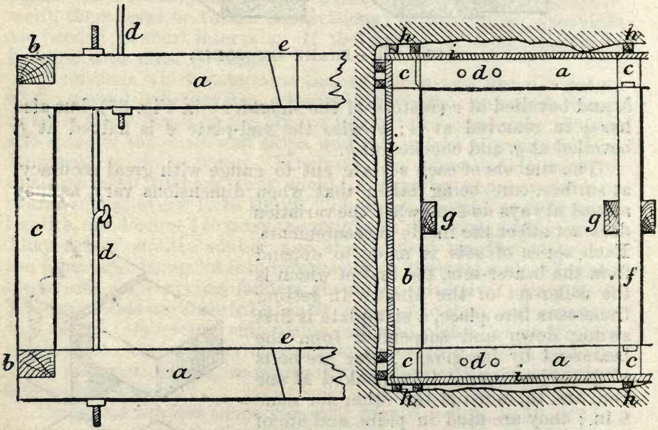


FIG. 66.
FRAME-SET JOINT.

next bearer-set is reached, and all wedges being invariably driven downwards. By degrees the hangers are removed for re-use. A saw-cut across the face of each timber at truly right-angles and at a uniform exact distance from the shoulder, made before the timbers are sent below, greatly facilitates alignment. The mortices for receiving the dovetailed ends of the dividers are also cut at surface, and are usually about 1 in. deep; it is not desirable to weaken the side-plates beyond what is absolutely necessary, and the thrust is most likely to be inwards, thus tightening the dividers. The wedging of the set is most particularly done just opposite the dividers. The dimensions of the dividers are often somewhat less than those of the wall-plates, but in some of the Rand mines they are made



FIGS. 67, 68.—FRAME-SET IN PLACE.

over 50% deeper, and thus present a broad shoulder to the intermediate studdles. The guides which occupy the runners of the cages and skips are cut out of well-seasoned hard-grained wood quite free from knots, and are planed smooth and greased when in use. They are fastened to the dividers by long coach-screws, joints being always made at dividers. A common size is 3 to 4 in. diam. each way, but care must be taken that they are large enough to ensure rigidity and absolutely secure hold. Lagging may consist of stout planking (up to 3 in. sometimes) in bad ground, or light laths (split round timber) when only small stuff is to be held in place, or may be dispensed with entirely in solid rock. Fig. 67 shows an elevation and Fig. 68 a plan of one end of a set in place:

a, side-plates; *b*, end-plates; *c*, studdles; *d*, hangers; *e*, mortices for dividers *f*; *g*, guides for cage or skip; *h*, blocks and wedges; *i*, lagging.

When the scale of operations makes a 4-compartment shaft necessary, the arrangement of the compartments in the form of a square is for many reasons preferable to the usual parallel system. Not only is the ground more easily and cheaply broken, and the timbers more readily procured of suitable strength and more convenient to handle, but the shaft is a much more solid and cohesive structure when finished.

Underlay Shafts.—Main working shafts on the incline are always timbered on the square-set principle, and lagged at least overhead. The angle of the underlay must to some extent determine how the sets will be cut, approaching the vertical-shaft style as it becomes steep, and the level- or drift-style as it flattens. In very good ground, it is possible to almost dispense with timber, only a single stull-piece or studdle, and a cap or head-board, being used to check the "winding" of small pieces of rock from the roof, with sills or sole-pieces for carrying the rails on which the skips or cages run, and a row of dividers carrying a brattice to separate the ladder-way from the skip-roads.

While, in the majority of cases, the underlay shaft is sunk with the several compartments ranged side by side on the same plane, there are occasions, such as bad ground, when the work can be done much more cheaply, and much greater safety can be secured, by arranging the shaft narrow way downwards. This was done by D. W. Brunton at the Smuggler mine, Aspen, Colorado, on an angle of 55° and to a depth of 800 ft., the pump-way being at the top, with most gratifying results.

When the full frame-set is required, it generally resembles Fig. 69: *a*, cap or head-piece: *b*, sill or sole-piece (these corresponding to the side-plates in a vertical shaft); *c*, stull or post (representing the divider or end-plate); *d*, distance-piece (resembling the studdle); *e*, head-boards, lining, or lagging. The heaviness of the ground controls the nearness of the sets to each other, 3 ft. and 6 ft. being the ordinary extremes and 4–5 ft. the most common distances. Both cap and sill are sometimes mortised to receive the tenons on both the dividers and the end-posts, when, if the ground is heavy, the caps need to be of greater depth than the sills; they are also mortised or checked to accommodate the distance-pieces, as in Fig. 70: *a*, cap; *b*, sill; *c*, end-post; *d*, divider. But more often the joint between cap and sill and the end-posts is effected as in Fig. 71, the cap *a* being checked about 1 in. deep with a bevel on the inner side, and the sill *b* more simply treated. When caps and sills are mortised, they must extend beyond the end-posts some few inches; when only checked and bevelled, they need not. The distance-pieces in a flat shaft are of least importance, and often only nailed in position. When not stayed by posts and dividers, sills should be let into hitches cut in the solid rock at each end, and may be of larger scantling, so as to admit of heavy rails being held by coach-bolts. A collar-set and bearer-sets are necessary, as in vertical shafts, but the latter need not be so frequent. Guides are sometimes attached to the posts when safety-catches are used on the skips or cages; but a better plan perhaps is to apply the safety-grip to the rails, which are necessarily the most capable of resistance.

For procuring true alignment of the timbering in an underlay shaft, the straight-edge and plumb-bob again come into play. The former is made somewhat longer than the interval between 2 sets, and to the side opposite the true edge is attached a frame, so that a plumb-line suspended from one end will hang truly vertical along a marked line, when the straight-edge is set at the

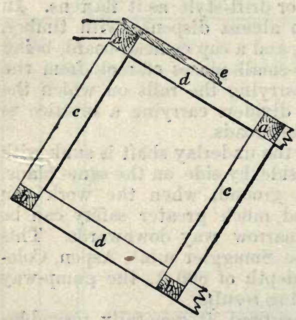


FIG. 69.

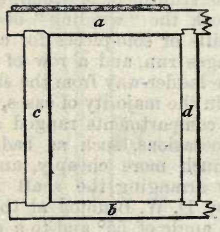


FIG. 70.

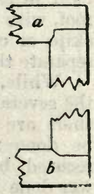


FIG. 71.

FRAME-SET FOR UNDERLAY SHAFT.

correct angle of the incline resting on 3 sills simultaneously. As each set is placed in position, it is held by hangers or tie-rods until permanently wedged-up.

Although it is customary to suspend timbers on hanging-bolts, and then wedge up, sometimes sinking is continued until it is necessary to put in a new set of bearers, and on these bearers the shaft-timbering is built up until the sets reach the bearers above. Possibly the timbers are more solid this way, but this is doubtful, and it has the great disadvantage that should it be necessary, owing to bad ground or other cause, to rush down the timbers as soon as possible near to the sinkers, this cannot be done without cutting hitches for new bearers, and entirely stopping all sinking at the bottom of the shaft. A very expeditious method of timbering, in vogue at the Simmer Deep Levels, is as follows. Four sets of wall-plates are let down at one time (between changes of shift) in this way: The engine is uncoupled, and the east and west drums are run alternately; the north side wall-plate is hitched on to the east rope with a chain and shackle about 13 ft. from the centre of the timber, with a catch chain round the end to keep the timber plumb with the rope. On arriving at the bottom set of wedged timbers, the top west hanging-bolt is put in, and in the hole for the lower west hanging-bolt an eye-bolt with nut is inserted. To this eye-bolt is attached a rope which is hitched round the west hanging-bolt of the last north wedged wall-plate. The wall-plate is lowered, the catch chain is loosened, and the east hanging bolt is inserted. The rope is then lowered, and the hemp rope is raised until the hanging-bolts are in a position to hook on. Two Kaffirs immediately jump down on to the suspended wall-plate, to guide the hooks into place, the engine rope holding one end and the hemp rope the other. Both ropes are then slacked off, the shackle and eye-bolt are loosened, and the centre hanging-bolt is put into place. The rope is pulled up, and, while the engine is being clutched for the west drum, the upper hanging-bolts for the next north wall-plate are put in. The south side wall-plate is then lowered by the west drum, and secured in place. As soon as the 8 wall-plates (4 on each side of the shaft) are in position, the stage planking (3×9 in. Oregon) is sent down in the bucket, and off-loaded on to the top set of hanging wall-plates forming the working platform. The east kibble then brings down the east end-plate and the divider for the east compartment, the kibble being half dumped, and the timber drawn straight out on to the stage planks, two Kaffirs holding the kibble in a horizontal position. The same method is pursued with the west end-plate and other dividers, only these are landed at the opposite side of the shaft. A hitch is taken round each end of the last wedged end-plate, with two ropes tied to the new end-plate, which is lowered into position on to the hanging wall-plate, and spiked in approximate position (with two 6-in. wire nails) to the wall-plate, while the dividers, which are slung the same as the end-plates, are placed. The studdles are then put in, each being temporarily secured by a rope passing round the set above; the hanging-bolts are tightened up, and the set is ready for blocking. The timbers for the three other sets are then put in similarly. Blocking is commenced from the bottom set, which is roughly blocked into line by wedging at the back of the end- and wall-plates at the centre; the sets are then permanently blocked at the corners, and behind the dividers, the timberman holding the block with his foot while he wedges up. The bottom set is not permanently blocked until the next set of wall-plates are lowered, owing to the danger of a chance block dropping on the sinkers below. The first set of the four is then permanently blocked, a Kaffir on the stage planks below holding the block until it is firmly wedged. Wherever possible, the end of the block next the rock is slightly lower than the end next the template, this being accomplished by nailing a wedge to the back of the plate. Under these conditions, should anything happen to the hanging-bolts, the sets tend to tighten themselves. Dry soft-wood wedges are driven in between the wall-plate and the block, but only behind the dividers and at the corners, never at the middle of a compartment. Four plumb-lines are used, two in the centre of the end-plates, and two in the centre of the wall-plates, set out from the last bearer set, about 3 in. from the plate. In the case of the end-plates, a square is used to set the saw-cut marking the centre of the new end-plate to the plumb-line, and the wall-plates are sighted across, and wedged up until in alignment with the line. Bearers are put in at every 16 sets, 6 × 12 in. in section under the end-plates, and 6 × 10 in. under the centre divider. The hitches are cut by drilling out the bottom completely, and blasting out the centre. Each end bearer is carefully plumbed from the bearer above, and wooden brackets are nailed on to hold the plumb-line about 3 in. from the plate. (A. E. Pettit, Tr. Inst. M.M.)

Another method, probably quicker than the last, is as follows: A hole is bored exactly in the centre of the wall-plate, in which is placed an eye-bolt attached to a chain. This chain is rather longer than half the wall-plate, and is attached to the hauling-rope by a shackle. In place of the usual two holes at the ends of the wall-plate, three holes are bored, to receive the hanging chains. Two wall-plates are hitched on to the winding-rope at one time, lashed at the top to the rope with a hemp cord, to keep them perpendicular in the shaft. On reaching the bottom set of timbers, the hemp cord is released, and the plates are allowed to swing horizontally. Into each of the extra end holes is inserted an eye-bolt, with a chain attached (the same length as the studdle), and terminating with another eye-bolt, which is put into the extra holes in the last set of wedged timbers. The winding-rope is then slackened, leaving the two plates hanging on their respective chains. The main lowering chains are then released, and the rope is hauled up. During the time the next set of wall-plates are being lowered, the hanging-bolts are put into the first pair of plates, and the hanging-chains are ready for the next pair. (A. E. Pettit.)

At the Village Deep, the usual timbering was followed: Wall-plates, 30 ft. 4 in. × 9 × 9; end-plates, 8 ft. × 9 × 9; corner studdles, 8 × 8; other studdles, 9 × 6; dividers, 9 × 7. All pitch pine. Sets, 6 ft. vertical centres. Owing to 3 compartments being used for hauling, much of the timbering had to be done while drilling was in progress. Wall-plates were hung 30 ft. on each side at a time, suspended from the rope of the centre compartment (bucket being first removed), swung into position by a rope at each end, and hung on 4 1½-in. bolts from the set above. With skilled timbermen, this occupied only 5 min. per wall-plate. The hanging was always arranged to follow the second blast, for machines, bars and drilling gear were all lowered before the second clean up was actually finished. Blocking the sets, and placing studdles and dividers, succeeded as soon as possible. Runners (8×4 in. wooden, 30-ft. lengths) were kept as close as could be to the bottom of the timbers. Sets of 6 single bearers were placed in suitable positions, at fairly regular 80-ft. intervals, in deep hitches; end bearers 14×9 in., centres 14×7 in. These in position and sets blocked, the hanging-bolts above were removed. The timbering averaged 60 ft. from the bottom, and was designed to carry 4-t. skips on 60-lb. rails, 52 in. gauge.

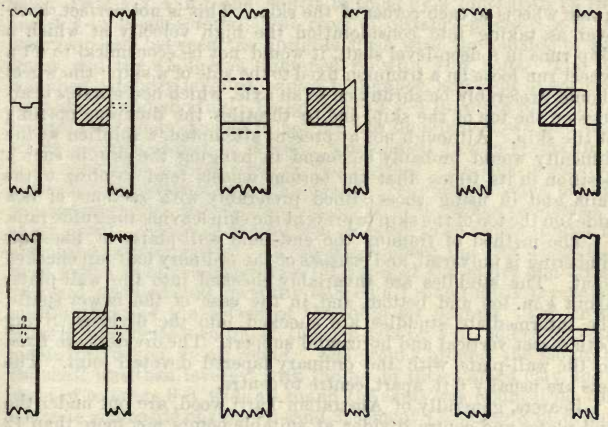


FIG. 72.—JOINTS USED FOR WOODEN GUIDES.

Guides.—Much variety is shown in the section of wood guides used, the sizes most met with being 8×4 in., 5×6 in., 6×4 in., and 4×5 in. Although of rather large area, the 8×4 in. section is probably safest, as a greater amount of surface is exposed for attaching to the dividers. Fig. 72 shows some of the many joints for wooden guides in use on the Rand. As considerable trouble has been caused by the coach-screws snapping (resulting sometimes in a serious wreck to the shaft), it is now customary, where narrow guides are used, to bolt or spike an angle-iron on top and bottom of the dividers, and to hold the guides to this with an extra bolt.

In the Robinson Deep No. 2 shaft, steel guides, 5×4 in. outside section, are used with satisfactory results.

Steel rails are also much used as guides, and are obviously necessary when the shaft goes round the curve to the underlay. Usual sections are 45, 50 and 60 lb. per yd. In many of the older shafts, the wheels of the skip are kept on these rails by an angle-iron fixed to the divider on the opposite side of the wheel to the rail; but this has a disadvantage, as a skip running at 2000–3000 ft. per min. in the vertical develops a tremendous velocity on the periphery of the wheel, and, when the wheel leaves the rail and touches the angle-iron, the momentum stored up in this spinning wheel is suddenly reversed, and sets up an undue strain on the angle-iron. Probably the most satisfactory way of using rails as guides is to fix one at each corner of the compartment, and attach guide wheels to each corner of the skip. This is not perfect, however, as, taking into consideration the high velocity at which a skip runs in a deep-level shaft, it would not be economical to let a wheel run loose on a trunnion fixed to the side of a skip; the wheel should preferably be shrunk on to an axle, which necessitates bearings on the top of the skip, and so throttles the dumping opening of the skip. Although not at present attempted, a solution to the difficulty would probably be found in hanging the skip in such a position in its frame that the bottom wheels tend to cling to the rails, and in using shoes (lined preferably with sections of raw hide) on the top of the skip to prevent the skip leaving the guide rails.

The method of framing the end- and wall-plates of the shaft timbering is universal, and consists of the ordinary half-lap checked joint. The studdles are invariably checked into the wall-plates about ½ in. top and bottom, and, in the case of the newer shafts, the intermediate studdles are checked into the dividers, giving both direct vertical and horizontal support. The dividers are fixed to the wall-plate with the ordinary tapered dovetail joint. The sets are usually 6 ft. apart, centre to centre.

Bearers, generally of Australian karri wood, are put under the end-plates and centre divider at suitable points not more than 12 sets apart, but in the near future it is probable that these karri bearers will give way to steel I-beams.

The hanging-bolts are usually 1⅛ in. diam. round iron, or, in the larger shafts, 1¼ in., with punched iron washers. (A. E. Pettit.)

Crib-Setting.—It is usual on the Rand to rest the collar-set of a deep-level shaft upon a wall of concrete, 3 ft. thick all round the shaft, built on solid rock. The height of such a wall is regulated by the depth at which solid rock is encountered, and seldom exceeds 10 ft. Its object is to give a good bearing for the collar-set; to avoid disturbance of the shaft by foundations for permanent heavy head-gears; and to prevent water percolating through the surface drift and rotting the timbers. The arrangement answers well, and is almost universally adopted. Where this is inconvenient, crib-setting is resorted to, as described below. The excavation was made 3 ft. 6 in. greater all round than the outside dimensions of the timber sets, in the case illustrated (37 ft. × 14 ft. 6 in.), and was carried down 23 ft. The formation cut through consisted of 6 ft. black mud (not very wet), followed by kaolin and clay, requiring careful staying and shoring all round. It was not disturbed, and (except a very little surface scaling on drying) no trouble was encountered. At 23 ft., a ledge 3 ft. wide was chiselled and levelled; on this 1 in. cement mortar (3 sharp sand to 1 Portland cement) was laid and allowed to set. The excavation was then resumed, but 3 ft. less all round; and the timber for the crib-set was prepared. This was made of 12 × 12 in. pitch pine 30 ft. long, with 12 uprights of same dimensions, and jointed the same as the shaft timbers. Each upright was mortised into the sets (the mortices being 12 × 4 × 4 in.), and bolted to them by 2 plate bolts ($\frac{7}{8}$ in.) and 4 $\frac{3}{4}$-in. coach-screws. Two rows of distance-pieces 8 × 8 in. were mortised between the uprights flush with the outside of the frame, and the whole was tightened up with $\frac{3}{4}$-in. bolts. The set was then carefully levelled, squared, plumbed, and held in position by wedging and shoring. Lagging of 12 × 4 in. pitch pine in one length, previously steeped in hot tar, was then placed, wedged, and where possible nailed to the set. Corrugated iron in sheets 10 × 2 ft. was fixed at the back of the lagging. As this proceeded, concrete was simultaneously rammed in on opposite sides, behind the lagging; this concrete was composed of 4 broken sandstone, 3 sharp sand, 1 cement, and was well tamped in to a height of 5 ft. above the bottom crib-set. Above the concrete, the clay which had been taken from the excavation (and spread out to dry and crumble) was puddled and carefully tamped all round until it reached within 12 in. of the bottom of the collar-set bearers. The cross-bearers were bolted on top of the crib-sets by 1 in. bolts, and were carefully wedged up all round. The ground around the mouth of the shaft was then levelled with the top of the rammed clay, and a sheet of concrete 6 ft. wide all round the shaft was rammed in; this concrete was made of stone 4, sand 3, cement 1. After 8 in. of this had been rammed in its place, two rows of 40-lb. rails were wedged against the bearers, and concreting was continued to the top of the cross-bearers. The concrete rested practically upon the original undisturbed surface of the ground, and was only completed after the tamped clay at the back of the lagging had properly settled.

The collar-set was next slipped into place on the cross-bearers, followed by the other sets carefully wedged against the uprights of the crib-set. No lagging at the back of the timbering was used until the ledge of the crib-set was reached. A bearer-set was established in the hard sandstone at 34 ft. from top of collar-set—4 bearers 7 × 12 in and 2 ends 12 × 12 in., let 16 in. into each wall, and fixed up in concrete. The space between the lagging and the walls was filled with puddled clay, tamped with long sticks, to the bottom of the bearer-set. This tamping effectively stopped the dripping which was encountered during sinking. The crib-set used 494 cub. ft. pine, and the total weight was 18,000 lb., and the grand total weight of timbering hanging from top cross-bearers was 15½ short tons. The cost was 386*l*., as against 711*l*. for the same height of concrete wall, concrete being 59*s*. 6*d*. per cub. yd. and cement 39*s*. 6*d*. per bbl. The work occupied 4 day-shifts. (H. D. Griffiths.)

Concrete Linings.—The increasing cost and diminishing supply of good mine timber compel attention to other kinds of lining, where the life of the mine is likely to outlast the timber. The decreased cost of cement has added to the weight of argument in favour of concrete. Brick and stone are very pervious to water, and, in cold weather, their roughness helps the formation of icicles, to the great danger of the shaft, machinery, and pipes. The cost of brick or stone linings is sometimes excessive, due to original cost of materials, necessity of using an excessive amount of mortar, and high cost of skilled labour. The time required to place brick or stone is double that in lining with concrete, and the job is not so strong, by reason of the difficulty in securing a bond to the walls of the shaft in the presence of much water. The usual timber lining lasts 12–15 years; in 6–8 years it becomes necessary to replace individual timbers and sections, causing temporary shutdowns. In the life of a good mine, say 30 years, it is necessary to re-timber the shaft entirely at least once, beside making many minor repairs. The cost of re-timbering is much higher than that of the original lining; and to this must be added loss of income during repairs. Another disadvantage with wooden lining is the danger from fire. The chief advantages of timber are lower first cost, greater speed in placing, and better adaptation to square shafts. Framing and placing double the original cost. But timber can be placed in about one-third the time required for concrete, or a gain of 13 days per 100 ft. of ordinary shaft size, a matter of importance. For concrete, masonry, or iron linings, an elliptical or circular section is compulsory, though it causes some increase in excavation beyond the area required to accommodate the cage. Roughly speaking, a square shaft lined with concrete would require walls twice as thick as an elliptical.

The following comparisons (by F. R. Dravo, E. & M. Jl.) of shafts of the usual design with timber lining, and elliptical and circular concrete-lined shafts, are most instructive.

Two shafts were sunk through one seam of coal at 100 ft., and continued through a second seam at 175 ft. Shaft A was 14 ft. 2 in. × 20 ft., and was lined for 45 ft., to shut off surface water, with concrete 12 in. thick. Shaft B was 17 ft. 4 in. × 33 ft., the concrete being 12 in. thick at the sides and 18 in. at the ends. It was a 4-compartment, and was concreted throughout. An average progress of 16 ft. per week was made, 20 ft. being the maximum; and the total excavation amounted to 21 cub. yd. per ft. of depth. A paddle concrete-mixer was placed at the head of the shaft, and the

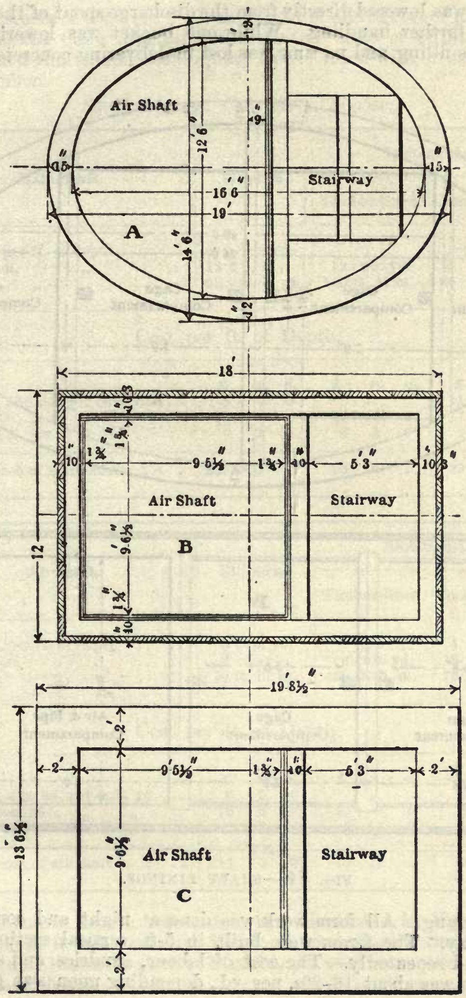


FIG. 73.—SHAFT LININGS.

concrete was lowered directly from the discharge spout of the mixer without further handling. While one bucket was lowering, the other was filling, and no time was lost in delivering concrete to the

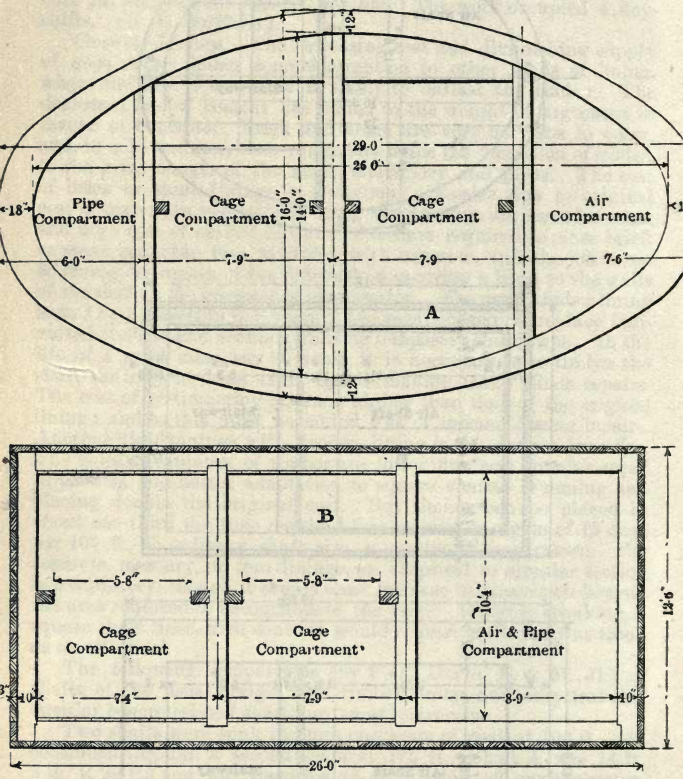


FIG. 74.—SHAFT LININGS.

placing gang. All form work was done at night, and concreting in the day. The forms were built in 5-ft. vertical sections, and were used repeatedly. The cost of labour, sundries, and superintendence was about 16–20s. per yd., depending upon size of shaft and thickness of concrete. To this must be added cost of materials. The annexed table illustrates the relative costs of elliptical concrete-lined shafts, and rectangular shafts lined with concrete and with timber.

Shaft Lining: Concrete v. *Timber.*

Main Shaft.	Elliptical.	Rectangular.	
		Timber-lined.	Concrete-lined.
Concrete, per ft. of depth	4½ cub. yd.	..	5·9 cub. yd.
Excavation, " "	13·5 "	12 cub. yd.	15 "
Timber, " "	90 ft. b. m.	500 ft. b. m.	80 ft. b. m.

Cost per ft. of Depth.

	£	s.	d.	£	s.	d.	£	s.	d.
Concrete, 36*s.* 9*d.* per cub. yd. ..	8	8	9		..		11	11	3
Excavation, 23*s.* " " ..	15	9	4	13	15	0	17	3	9
Timber, 12*l.* 10*s.* " M	1	2	6	6	5	0	1	0	0
Total cost of main shaft	£25	0	7	20	0	0	29	15	0

Air Shaft.	Elliptical.	Rectangular.	
		Timber-lined.	Concrete-lined.
Concrete, per ft. of depth	3 cub. yd.	..	4·3 cub. yd.
Excavation, " "	8 " "	8 cub. yd.	9·9 "
Timber, " "	70 ft. b. m.	450 ft. b. m.	70 ft. b. m.

Cost per ft. of Depth.

	£	s.	d.	£	s.	d.	£	s.	d.
Concrete, 41*s.* 8*d.* per cub. yd. ..	6	5	0		..		8	19	2
Excavation, 25*s.* " " ..	10	0	0	10	0	0	12	7	6
Timber, 12*l.* 14*s.* 2*d.* " ..	0	17	6	5	12	6	0	17	6
Total cost of air shaft	£17	2	6	15	12	6	22	4	2

The shafts herein referred to are illustrated in Figs. 73, 74. Fig. 73, A is oval concrete-lined; B, rectangular timbered; C, rectangular concrete-lined; Fig. 74, A, oval concrete; B, rectangular timbered.

Steel Lining.—An example of steel "timbering" in a Lake Superior incline shaft is described by F. Drake (E. & M. Jl., Sept. '02). This shaft is 6 ft. × 17 ft. 6 in. inside, dips at 70°, contains two hoisting compartments (each 6 × 6 ft.) and a smaller compartment for ladder-way and pipes, and is intended for 2 5-t. skips, on tracks of 4 ft. $7\frac{3}{8}$ in. gauge. Back runners are provided to guard against the skips leaving the track. The hanging-wall and footwall plates are of 30-lb. rail; the end-pieces and one divider of 25-lb. rail, and the other divider of 3-in. $7\frac{1}{2}$-lb. I-beam. Beams could be used for wall-plates and ends also, but as a rule, for such light members as are required in this case, they would be more expensive than rails, and they are employed for the divider between the skip compartments only because of their more convenient form. Where conditions require rail of 50 lb., or greater weight, beams can be economically employed. The various members of the sets are joined by connecting-pieces of $3\frac{1}{2} \times 3\frac{1}{2} \times \frac{1}{2}$ in. angle-iron secured by $\frac{1}{2}$-in. rivets, 2 rivets in each leg of the angles. Pieces 3 in. long, of 8 × 8 × $\frac{3}{4}$ in. angle, are used as brackets for supporting the back runners, and are riveted to the end, or divider, as the case may be, with 4 $\frac{1}{2}$-in. rivets. The studdles are merely pieces of rail, 4 ft. long, and having the ends slotted. The head of the rail forming the studdle faces inside the shaft, and extends down between the head and flange of the end-piece (or between the two flanges, if an I-beam is employed), and rests on the web of the latter; thus the studdle is prevented from moving in a direction parallel to the wall-plate. The studdle is also prevented from moving in a direction at right angles to the wall-plate. When there is pressure between the sets, therefore, the studdles cannot be knocked out by anything less than a force that will bend them. For the studdles, almost any size rail can be used; and, as only short lengths are required, pieces of old rail can thus be utilised.

In the upper portion of the shaft the rock was of such a character that no small pieces were likely to fall from the walls. Here the shaft was not lagged tightly, but instead, old wire ropes were stretched longitudinally along the ends and back, closely enough together to prevent the falling of any large pieces that might become loosened. This answered every purpose until there was danger of small pieces falling, and then ordinary 2-in. plank was used. It would be desirable to use metal lath, were it not for the expense, as the employment of wood diminishes the fire-proof quality of the lining. To partially offset this, it has been suggested to introduce sections of metal lath at intervals, to act as breaks for retarding the spread of fire, for a length of 4 sets (16 ft.) in every 100 ft. Corrugated steel and buckled plates are about the only materials available for metal lath. Ordinary flat plates, stiffened by angles, could be used, but for equal strength they are much more expensive than either of the other materials. No. 16 steel (the heaviest corrugated) is $\frac{1}{16}$ in. thick, and, if galvanised, will probably last as long as the steel of the framing. It would be much cheaper than any other fire-proof material proposed. A single thickness would not have the strength of sound 2-in. plank, being capable of carrying but 200 lb. per sq. ft. distributed load, as against 360 lb. for the plank, but would be sufficient for most ordinary shafts. Where this was not the case, two thicknesses could be used, or recourse be had to buckled plates. The latter are not ordinarily made less than $\frac{1}{4}$ in. thick, and will sustain about 560 lb. per sq. ft. Buckled plates of $\frac{3}{16}$ in. can, however, be had on special order, and would carry about 400 lb. per sq. ft., or somewhat more than 2-in. plank.

When wooden laths, corrugated steel, or buckled plates are used, the pieces may either be cut of such a length as to fit, at both ends, against the flanges of the members of the sets, or so that they rest against these flanges at their lower ends, but at their upper ends against the heads of the set-members. The former method has the disadvantage that the lath cannot very easily be taken out or replaced; and, when wooden laths are used, that unless the ends of the laths are specially prepared, their thickness is too great to allow them to fit down between the head and flange of the wall-plate or end-piece; thus but little of the flange (only $\frac{1}{2}$ in., with 25-lb. rail) is available for supporting the laths against pressure tending to force them into the shaft. This last objection would not hold in the case of metal laths, as, being thin, they would fit down against the web of the plates, so that the full height of the flange would support the lath.

In sinking, the steel sets are suspended by hangers, in the same manner as wooden ones, each hanger consisting of a bar, formed into a hook at the lower end, and having its upper end bent over to hold a vertically moving screw.

The amount of shop-work required for making the sets is so small, if the materials are ordered from the mills cut and drilled to pattern, that the ordinary mine staff can easily equip one or two shafts without neglecting their ordinary duties.

Stations.—At fixed intervals in shafts, varying from 50 to 200 ft., but most usually about 100 ft., levels or drifts are run. These necessitate provision of additional space alongside the shaft, known as stations or chambers. The entrance to such a chamber from the shaft must be of sufficient height to admit the longest stick of timber used in the workings, unless special timber-ways are provided, and its area must suffice to accommodate timber-carriage and ore-trucks, if cage-hauling is adopted, or bins and shoots for skip-loading. The ordinary studdles of the frame-set are replaced by longer and stouter posts, properly joined to the end-plates, while the side-plates are omitted, false timbers being inserted temporarily, if necessary. Under exceptional circumstances, a chamber may be 40 ft. high next the shaft, but 20 ft. is fully sufficient in most cases; the height is graduated down to that of the diverging levels, so as to avoid overhead timbering as much as possible. In width, the chamber accords with that of the whole shaft, its posts being aligned on the shaft studdles. Chairs are inserted in the shaft for supporting the cage or skip while at rest, and the chamber is floored and laid with flat-sheets for convenience in handling trucks. In incline shafts. the truck may be run on to a suitable cage, or be tipped directly into a skip, or into a bin for feeding the skip. The last arrangement is best when the output is large, as it permits of holding a greater reserve of ore, and of hoisting independently of getting. (See Hauling and Hoisting.)

At the Alaska Treadwell, below the 440 level, stations are only 150 ft. apart, and are 40–60 ft. long, and with an average height of 8 ft. The main cross-cuts run parallel to the wall-plates of the shaft, and as far as possible it is aimed to have the station on the side opposite from that toward which the skips dump. The station-set is 14 ft. high, and tied in two directions by iron bolts. To protect the front wall-plate that serves for the sill of the station, a stull 10 by 14 in. is put in between the wall-plate and the edge of the station.

Connections between Vertical and Underlay.—The development of the composite shaft system has been most marked on the Rand, where probably the best mining practice is to be found, and from which the best examples may be taken. Special timbering is generally involved where the incline merges into the vertical. A substantial bearer should mark the commencement and the end of the curve which forms the junction. This curve is usually a circular one at about 50 ft. radius, as shown in Fig. 75, representing the Jumpers Deep (S. J. Truscott), but a parabolic curve proposed by R. M. Catlin has been sometimes adopted, and is said to admit of much higher speed in winding. In Fig. 75, the point of ground *a* between the two shafts is squared off solid, and cemented masonry *b* is built up to support the main bearer *c*, which is also hitched in at the end *d*. This main bearer, 12 × 6 in., carries two main posts *e* of similar dimensions, which extend in one piece about 26 ft. long to the upper bearer *f*, and carry a series of 12 × 6-in. cross-pieces *g*. The main bearer *c* is extended across the incline by another length *h*, being supported at one end by the post *i* footed in the masonry *b*, and at the other by the top end of a bearer *k* hitched in the underlay. The back of the underlay is good standing ground, and needs no lagging, nor do the posts *ij* require any staying against lateral thrust. The whole structure is wedged tightly in place, and tied together by numerous bolts *l*. Guide-pulleys *m* carry the rope over the curve; rails *n* are fastened to sleepers *o* resting on the cross-pieces *g*, and guides *p* are attached to the cross-pieces; *r* is a sump.

Sumps and Lodges.—A sump or well is provided at the bottom of a shaft by sinking it somewhat below the bottom level, and into it the drainage water is led for pumping. But in the majority of cases nearly all the water made by a mine comes from superficial strata, and can be gathered before it reaches a great depth. Lodges are gutters surrounding a shaft, made by cutting a ledge all round, and building an edge of planks laid in clay or of bricks laid in cement; a pipe conveys the water caught to a convenient

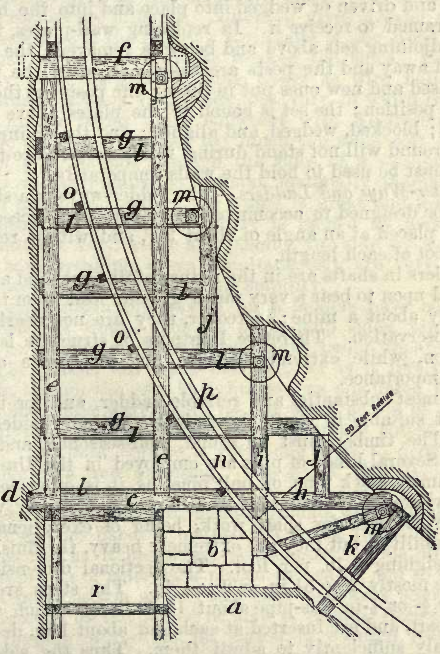


FIG. 75.—UNDERLAY MEETING VERTICAL SHAFT.

sump for the pumps. It is often desirable to keep surface-water undefiled by the highly-mineralised water of the lower levels, it being applicable to various purposes for which the other is unfit. Sumps should always suffice in size for at least 12 hours' accumulation, in case of pumps breaking down; in many mines, 3 to 4 days' water can be retained without hindrance to work.

Renewing Timbers.—When a weakness manifests itself in shaft timbering, the faulty member must be replaced by a new one. First of all, several sets, particularly those next above that at fault, are tightly bound together by hanging-bolts. When posts only are to be renewed, the adjacent lagging is removed, and enough ground is excavated from behind each post to allow of its being driven back from the shaft until clear of the timbers, or it may be chopped out; the new post is then placed in position from behind, and driven or wedged into place and into the hitch in the plates framed to receive it. In replacing wall-plates, the lagging of the adjoining sets above and below is removed; the blocks are knocked away and the posts are taken out, when the plates may be released and new ones put in place; the posts are then returned to their position; the set is bound to the plates above and below by bolts; blocked, wedged, and aligned; and the lining is put in. If the ground will not stand during this process, false timbers and lining must be used to hold the walls temporarily.

Ladder-Ways and Ladders.—The ladder-ways in a shaft should always be designed to accommodate ladders not exceeding 20 ft. lengths, placed at an angle of about 30°, and with a resting-stage at the foot of each length.

Ladders in shafts are in the nature of fixtures, and are liable to be called upon to bear a very much greater load than those in use generally about a mine; moreover, they are not nearly so much under observation. Therefore lightness of structure is not a consideration, while extreme strength and endurance are of the utmost importance.

The most substantial and reliable ladder, and for that reason the most suitable in shafts, is made with timber sides and iron steps. The timber must be sound, well-seasoned, and free from knots. Several kinds of pine are employed in the United States, the common black pine, or bull pine, as it is often called, being most favoured. In Australia, the harder and denser varieties of gum are the best for shaft work, being of exceptional strength and durability; but they are extremely heavy, the finished ladder often weighing 3 lb. per foot. The sectional dimensions of the sides are mostly $2 \times 3\frac{1}{2}$ in. or 2×4 in. The steps are pieces of ordinary $\frac{3}{4}$- or $\frac{7}{8}$-in. gas-pipe about 14 in. long, which are spaced 10 in. apart, and are inserted at each end about 1 in. deep in holes bored only sufficiently to admit them. Thus the sides are not appreciably weakened. The width between sides is 12 in.; it may be reduced to 10 in. without detriment, and the strength and lightness of construction will both be increased. To maintain the sides parallel, and prevent spreading, a round-iron rod of $\frac{1}{2}$–$\frac{5}{8}$ in. diam. is passed right through both sides and one of the pipe steps; it is secured by washers and a bolt-head at one end and a nut at the other, so that it can be screwed up quite tight. In a 20-ft. ladder, there may be 3 of the setic-bolts, one in the centre and one towards each end. For all positions where the traffic is constant and heavy and the ladder is comparatively a permanency, this style of building is much the best unless the corrosive nature of the mine-water puts iron out of the question.

Then, as also in most situations where the ladder is only temporarily located, the wooden slat or rung is preferable. This may be simply nailed on, or supported by a recess cut in the face of the sides, or inserted in holes bored laterally. The last-named are least satisfactory, as the sides are much weakened by the boring, and repairs are difficult. Notching gives greater security against a loosening of the nails by shrinkage of the wood in drying, but must not be so deep as to weaken the sides. Cut nails have nearly twice as great holding power as wire nails. The slats are almost always 1 in. thick, but vary in width from 2 to 4 in., the former being quite sufficient with Australian hardwoods, while the latter is none too much for some soft timbers. A couple of ½-in. tie-rods through the sides of a wooden ladder add greatly to its endurance without materially increasing its weight. In bad places, where the traffic is great and iron rungs cannot be used, much greater security is gained by making the ladders of half the usual width and placing two side by side; the chances of both breaking down simultaneously are very remote, and the narrowed tread adds to their stability.

Various novelties in the way of mine ladders have been proposed at different times, such as paper-pulp sides, aluminium rungs, and so on, the object mainly sought being apparently to reduce weight. None has come into use, nor is it likely to, one would think, though the gain in lightness has amounted to 38 %. Experience has pretty well taught us the weak points of wooden ladders, and their cheapness and easy repair are potent recommendations. It would require a very long trial to demonstrate the reliability of a new material. (See En. & Min. Jl., June 12 and Dec. 25, 1897.)

Bad Ground.—In loose and running ground, which breaks away before a set can be fixed, a method of lining the sides of the excavation as fast as it progresses is necessary. This often consists of sawn planks or split slabs placed vertically, and so blocked and wedged into position as to press each outwards against the walls; the top end of the plank or slab is blocked from the wall against the lagging of the last set placed in position, reaching a foot or so above the wall-plates of that set, and is further blocked and wedged away from the wall-plates themselves, the effect of this being to throw the foot of the piece backward from the shaft against the side of the excavation. Such a lining is often carried completely around the shaft. In ground which is too heavy for this method, a system of spiling, similar to that employed when driving levels under such circumstances, is used, and consists in forcing down a shield, slab by slab, ahead of the excavation. The slabs, planks, or poles are sharpened at foot, and sometimes iron-

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shod at top, for driving down by a sledge-hammer. The spiling is started from behind the wall-plate of the last set, and with considerable lean inwards, but attains a steeper angle as it is driven down; it is often supplemented later on by additional planking. Hangers are generally left permanently in such ground.

In swelling or creeping ground which expands with great force, resistance is hopeless, because no timber could be got to withstand the pressure. This difficulty is overcome by using a strong open lagging, which will maintain its position, but afford an outlet for the displaced ground, the creep being in fact regulated rather

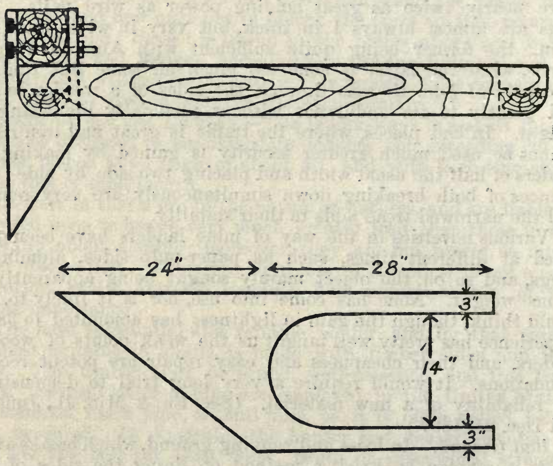


FIG. 76.—STEEL SHOE FOR DROP SHAFT.

than opposed; these conditions are observed so long as the necessity for them exists, the extruded material being removed. An arrangement successfully adopted at several mines is to place narrow square-sets with open lagging outside the true shaft-timbers, space being thus provided for a certain amount of accumulation.

In the Marquette iron region, where much trouble is caused by the boiling up of quicksand and water, filling the shaft sometimes 25 ft., experience with the older system of "drop-shaft" with bevelled bottom timbers was that on striking a boulder of a few feet diameter, the great weight of nearly 100 tons per 75 ft. of shaft would crush the bevelled edge of the bottom set, and hold the whole structure there until the boulder had been removed. The new type of drop-shaft has a chisel-shaped steel shoe, bolted to the bottom sets. This shoe will easily cut "hard-pan," and, striking a small boulder, will either crush it or push it aside—if too large, it can be blasted out with no injury to the bottom sets. When the "ledge" or ore-body is struck, the shoes may be taken off, one at a time, and shaft bearers put in their places, thus doing away with cutting out the bottom sets of the ordinary drop-shaft. These shoes may be used again in other shafts. In a shaft 17 ft. 4 in. × 10 ft. inside, the first 20 ft. of timbers will be of oak, the rest of best pine. The shaft sets are bolted together, 5 sets deep, with heavy bolts, and corner pieces 8 × 8 in. are spliced and bolted to keep them from pulling apart. To lessen the friction of the set descending through the sand, the outside may be sheathed with steel plates. At every 15 ft. is a ladder platform. The timber cage is large enough for 9-ft. timbers placed horizontally on a truck. Fig. 76 represents two views of the steel shoe (64 in. long by 24 in. wide), which is fastened to the bottom sets by 1½-in. bolts. (A. Foomis.)

The "telescope" or "travelling" shaft, sometimes adopted for sinking through drift, in deep alluvial mining in Victoria, means starting a shaft of such dimensions that successive smaller linings may be constructed inside the preceding one, and forced down by hydraulic pressure; but a very slight "rush" or "boil" will twist the timbers, and prevent further sinking. In this method, it is well to maintain considerable hydrostatic pressure by keeping the shaft pretty full of water, and thus preventing undue caving Excavation is performed by a "miser"—a kind of dredger or bucket scoop.

The salt beds of Detroit, Mich., at 1200–1500 ft., present great difficulties in shaft-sinking. Borings showed overburden 86–90 ft. thick; the first 30 ft. of this is hard clay, then 56–60 ft. of mud or silt (semi-liquid); at 60 ft., a seam of sand and gravel, 6 in. thick, from which flows sulphur water; again, other seams of sand and gravel, and more sulphur water; then 4–5 ft. of gaseous oil-rock, full of hydrogen; and, from there to the salt bed, limestone and sandstone strata. To work this salt bed, a sort of drop-shaft (Fig. 77) was constructed, consisting of a crib of 12 × 12 in. timbers *a*, bolted together with 8 1½-in. bolts *b*; tar and oakum were used at every joint, and the crib was made absolutely water-tight from top to bottom, in a construction similar to the hull of a ship, but 2 ft. larger each way on the bottom, and gradually tapered up for 18 ft., so that, as the shaft was pushed downward, it had a bell shape for the lower 18 ft. After rock was reached, the shaft was carried on the same size as the bottom of the crib, or 2 ft. longer and 2 ft. wider than the shaft was intended to be when completed. It was sunk in this size 18 ft. into the rock, and two issues of water were passed successfully. Then a crib *c* was started in the bottom of the shaft of the intended size of the shaft (6 × 16 ft. inside); and 18 in. of Portland cement *d* was used between the crib and the rock, and built up to the second watery seam. Then 2-in. pipes *e* were passed through the crib and into the crevice; the crevice was caulked between the pipes with oakum and tar, and the water was forced to pass out through the pipes. Then the crib and the cement work were carried up to the first

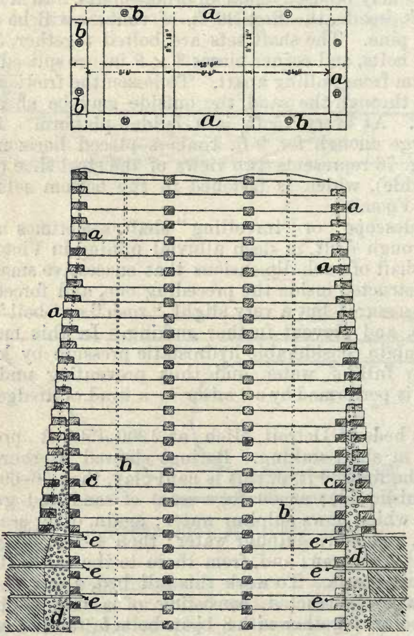


FIG. 77.—CEMENT AND TIMBER CRIBBING.

water crevice; pipes were again put through the crib into this, and the water was carried off as described. Then the crib was built up into the bell of the drop-shaft, a distance of 6 ft., and cement mortar was filled tight between the two cribs. After all the cement had set, gate valves were used on the pipes, and four-fifths of the water could be shut off; the pipes were then connected together, and 1 pipe was carried to the surface, and attached to a pump; 75 bbl. of cement (of about the consistency of cream) were pumped into these pipes, when the flow of water was stopped. (E. & M. Jl., Nov. '05.)

Occasionally recourse is had to sinking a cylinder of masonry, built upon a massive iron cutting-shoe, until skin friction prevents its further descent. Tie-bolts, attached to the cutting-shoe, are built into the brickwork, and, by means of these, a heavy iron anchor-ring is secured to the upper part of the shaft. A cylinder of iron is then built up inside the first cylinder, and is forced down by powerful hydraulic presses which work against the anchor-ring. The ground inside the cylinder is excavated by some form of dredge.

Another scheme consists in forcing cement slurry through bore-holes into the soft, fissured strata, thus forming a wall of concrete, within which sinking can be performed. This was successfully done at the Lens Collieries.

Sinking through swamp, ooze, and quicksand, by the Minnesota Iron Co., was effected as follows: Two heavy sets, or shaft-frames, cased on the outer side with steel, were started, and outside of them long steel-lined laths were driven downward. The laths fit closely together, and may be driven or jacked independently, which allows for the surrounding of any obstructions that may be encountered. The material inside the laths being removed, the inside sets were driven down, this being repeated to the bottom.

To reach rich gold alluvial beneath a swamp 600 ft. wide and ½ mile long, in Nova Scotia, the following caisson system was adopted. The first of these built was 14 ft. long, 7 ft. wide and 10 ft. high, with 3 sets of 6 × 9 in. spruce timber, planked with 2½-in. spruce, closed joints, but 3 in. smaller at top than at bottom, and the plank ends projected 6 in. above the top set. The windlass gear and sump box were placed on the top, and, as the excavating proceeded, the caisson was gradually lowered until the top was level with the surface. Another caisson was then built, the same as the first, but smaller by the thickness of the plank, so that it could telescope the first one 6 in., with the gear on top, as before. But many smooth boulders were encountered, too large to lift with the windlass gear; and it was almost impossible to drill them for blasting, on account of rising quicksand. The surface of the ground for many feet around the shaft gradually lowered, and the caisson began to stick fast; and ultimately the rising of the sand and the pressure of the large boulders upon the sides of the caissons prevented progress. Then, by sounding the sides of the caissons with a hammer, each large boulder was located; the planking was bored through with an auger, and a hole 15–20 in. deep was bored into each rock, according to its estimated size; the holes were charged with dynamite and long fuses; and timbers were cut to exactly fit endways between the sides of the caissons, and were driven into place until there was a solid wooden wall to resist the force of the blasting. This was entirely successful, and the caissons, after the boulders were shattered by firing, were relieved at once. (G. W. Stuart.)

The Honigmann differs from other sinking methods in that no casing is built until the shaft has been completely sunk through the sand to the solid rock formation. To prevent caving in, Honigmann proceeds as follows: (1) The water level in the shaft is raised (d) 70–80 ft. above the natural water level e in the quicksand outside, and (2) the sp. gr. of the water in the shaft is increased to about 1·2 by mixing clay with the water. Thus a pressure is obtained, which forces the clay contained in the water into the shaft walls, making a more solid mass of the quicksand. The increasing pressure of the quicksand toward the inside of the shaft, is held in equilibrium. The sludge loosened by the shaft percussion-drill is conveyed to the surface by a continuous water-flow, rising through the hollow cylindrical drill-rod, as shown in Fig. 78. A small gas-pipe a extends into the drill-rod for a short distance, and through this pipe is forced compressed air, which bubbles into drill-rod b, thereby making the water column in b lighter than that in the shaft. As a result, the water rises into b from the bottom of the shaft. The water and sludge flow through pipe c into a basin f, where the sludge is allowed to settle while the separated water flows back into the shaft. The velocity of the rising water in b can be increased by forcing more air through pipe a. Clay is mixed in the basin with the water returning to the shaft, to maintain the necessary sp. gr. When the shaft has reached solid ground, a casing is built in the usual way by means of iron tubbing. In passing through the quicksand, constant care has to be exercised to keep the artificial water level and sp. gr. of the water up to the predetermined standards, to prevent the shaft caving in. Several

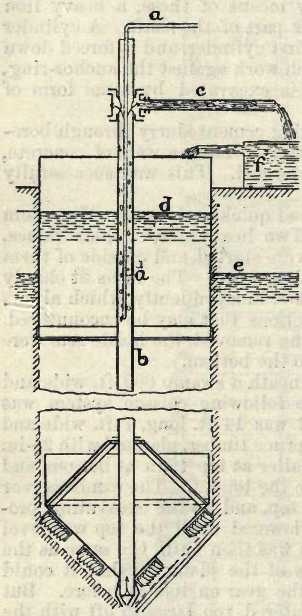


FIG. 78.—HONIGMANN METHOD.

shafts 12–18 ft. diam. have been sunk recently in Germany by this method, through 600 ft. of quicksand. (A. E. Hartmann.)

At the Susquehanna mine, a drop-section of shaft timbers, of full size of shaft, was framed and placed in position, 4 ft. in depth, of 12 × 12-in. timbers, bolted together and braced; the lower two sets were bevelled, flush outside, and a ⅛-in. steel plate, shaped to fit the edge, was fastened by spikes, making a shoe or edge 3 in. wide, to cut the sand. On the outside of this drop-section was bolted a sheeting plank, with lower edges hewn thin, so as to offer less resistance. The upper ends of the plank extended 3 ft. 6 in. above the timbers, and behind the last permanent shaft timbers. Some 20–30 jacks, 18 × 2½ in., were operated between the drop-section and the last permanent timbers, the plank sheeting serving to retain the sand. As the jacks forced down the drop-section, they were removed temporarily, and timbers were placed and bolted to the last permanent timbers; and, to bring the permanent timbers close together, pockets were cut out for nuts and washers. The greatest possible care in the use of jacks was necessary to secure straightness, and, as an additional protection against a crooked shaft, 8 × 8-in. diagonal braces were put at each end from the collar of the shaft down to depth of 85 ft., suggested by previous experience in the washing away of sand behind the timbers (and the caving) causing the entire shaft to surge several inches. The shaft timbers were hung by wire ropes from the bearers at the collar, but it was not necessary to truss the bearers. An improvement might be the substitution of steel sheeting for the plank, as less liable to break or get loose; and the steel, being thin, would allow the drop-section to cut the sand easier. (H. B. Sturtevant.)

In the sinking of several coal-mine shafts, where heavy water-bearing strata were encountered, much success has attended freezing the troublesome stratum, by the Poetsch system; but this method seems never to have been applied on any metalliferous mine, and it is in any case much more costly than the devices just described.

An interesting detailed article on sinking through very wet sands, by E. Mackay Heriot, appeared in E. & M. Jl., Dec. 1906.

Speed of Sinking.—This is subject to great variation, chiefly dependent on the nature of the ground. Generally speaking, inclines can be advanced more rapidly than verticals, because there is less time wasted in moving gear up and down, and often better opportunity for utilising bedding-planes in excavation.

Examples from Californian practice are as follows. The general average on the Mother Lode for ordinary 3-compartment verticals is 40 to 100 ft. a month. In the Oneida, a big shaft going to 2500 ft., encountering very hard rock and much water, with hand-drilling (3 shifts of 6 men each, necessitated by stratification being ill-adapted for machine work), the speed varies from 40 to 80 and averages 60 ft. a month. The Kennedy, in very hard grey slate, makes 80 ft. a month.

Some Transvaal figures are:—

Area of Excavation.	Ft. Sunk and Timbered per Month.	Remarks.
ft.		
28 × 8	55*a*, 40, 94*b*, 96*b*, 70*b*; av. 71	*a.* Heavy water.
23 × 8	65, 60, 60; „ 62	*b.* Hand-drilling.
18½ × 8 to 18 × 7	91, 66*c*, 87, 65*c*, 89, 86, 67, 67 } „ 81	*c.* Very hard ground.

Much higher averages have been attained more recently, 100 to 150 ft. being quite common.

Observations (at the Lake copper mines) on the time occupied in the various steps of loading and hoisting ⅓-ton buckets during shaft-sinking, showed the following averages—placing buckets, 40 sec.; filling, 2½ min.; hoisting, dumping and lowering, 1 min. 37 sec.; delays, 13 sec.; total, 4½ min.

Cost of Sinking.—On the Lake Superior copper mines, shaft-sinking is let on contract at 66*s.* 6*d.*–79*s.* per lin. ft., all supplies (except drills and steel) being furnished by contractors; timbering, tracklaying, etc., being additional. (W. R. Crane.)

Cost of Shaft-Sinking: Transvaal.

Area of Excavation, 28 × 8 ft. Cost per ft. Sunk and Timbered.	Area of Excavation, 23 × 8 ft. Cost per ft. Sunk and Timbered.	Area of Excavation, 18½ × 8 ft. Cost per ft. Sunk and Timbered.	Area of Excavation, 18 × 7½ ft. Cost per ft. Sunk and Timbered.	Remarks.
£34 4 3 *a*	£31 1 4	£22 4 2	£20 14 0 *c*	*a*, Heavy water; *b*, Hand-drilling; *c*, Very hard ground; *d*, Depths ranged from 691 to 1546 ft.; *e*, 911 to 1352 ft.; *f*, 714 to 1923 ft.; *g*, Recent.
41 7 11	24 9 0	24 2 10 *c*	18 12 4	
22 7 11 *b*	26 3 5	24 3 7	15 5 9 *b*	
23 3 3 *b*	22 17 7			
28 12 2 *b*		18 × 8	18 × 7	
	22 × 8	26 16 5	19 6 0	
29 19 1 *d*	33 14 8	19 10 5		
15 17 2 *g*	27 13 2 *e*	23 7 6	21 3 11 *f*	

Incline shafts cost much less than vertical shafts, because work proceeds more rapidly (there being a reduction of time lost at each blast), and there is a much smaller consumption of timber.

The Robinson main underlay shaft, 15 × 5½ ft. in the clear, cost about 9*l.* per ft. railed complete.

On the Mother Lode of California, the cost of sinking and timbering the ordinary 3-compartment vertical shaft averages between 6*l.* and 10*l.* a ft.; the Gwin, to 1800 ft., cost 7*l.* 6*s.* to 8*l.* 7*s.*; the Oneida, to about the same depth, with hand-drilling and many difficulties, reached 13*l.* 10*s.* per ft.

With increased depth, costs are augmented in more than proportionate ratio; Ihlseng states the increase for each 100 ft. to be "almost as the square root of the depth," but this is liable to considerable modification, and no rule can hold good in all cases. Costs of sinking in hard ground ("blue"), including raising and pumping, run mostly between 6*l.* and 10*l.* per ft.; timbering ranges from 3*l.* to 8*l.* per ft. In 3 shafts where contractors received 6*l.*, 7*l.* 10*s.* and 7*l.* 10*s.* per ft. respectively (including labour, explosives and lights), finished costs were 15*l.* 10*s.* 7*d.*, 20*l.* 1*s.* 3*d.*, and 20*l.* 11*s.* 2*d.* per ft. When on wages, drill-men generally get 30*l.* a month, and a bonus of 10*s.* per ft. for each foot over 70; white helpers, 15*s.* per shift and 2*s.* 6*d.* per ft. bonus; shovellers, 15*s.* per shift and 2*s.* 6*d.* extra if they finish their work under 10 hours. The smaller shafts do not exhibit a diminution in cost commensurate with their size. Hand-drilling is cheaper than machine in ordinary quartzite, but in very hard ground machine-drills are faster and cheaper. Explosives used in hand-drilling amount to about ½ to ¾ lb. blasting gelatine per ton of rock cut; in machine-drilling, to 1¼ lb.; moreover, in the latter case, the walls are less regular, and involve more labour in timbering, while more trouble is caused in damages and displacements by the heavier blasts. Some comparative costs are:—

a. 1896. First 500 ft. (hand), 20*l.* 18*s.* 5*d.*; next 650 ft. (machine), 26*l.* 19*s.* 3*d.*

b. 1897. First 99 ft. (hand), 12*l.* 5*s.* 3*d.*; next 88 ft. (machine), 16*l.* 1*s.* 10*d.*

c. 1898. First 450 ft. (hand), 17*l.* 5*s.* 10*d.*; next 760 ft. (machine), 18*l.* 5*s.* 4*d.*

On the Cinderella Deep, the estimated costs per ft., at each 1000 ft., including timbering, bratticing, and maintenance and office charges were—for 1–1000 ft., 18*l.* 2*s.* 10*d.* per ft.; 1000–2000 ft., 19*l.* 16*s.*; 2000–3000 ft., 22*l.* 15*s.* 6*d.*; 3000–4000 ft., 26*l.* 5*s.* 5*d.*

It is apparently no more expensive to put down a large shaft than a small one, notwithstanding the extra cost of timber, which in a large (28 × 8 ft.) shaft is 19 % of the cost, and of coal (10 % of the sinking cost); as the size increases, so the price of ground per cub. ft. decreases, and the rate of sinking increases, within certain limits, very much depending on facility for getting rid of broken ground.

Cost of Shaft-Sinking: Hand-Drilling.

For 89 ft. of 23½ × 8.	Cost per ft.			Cost %.
	£	s.	d.	
Sinking	7	5	4·5	48·3
Drill-sharpening	0	8	4·4	2·8
Timbering sets	2	15	0·0	18·3
„ ladder-landings	0	5	7·3	1·9
„ bearers..	0	1	9·0	0·6
Hoisting	1	7	4·7	9·1
Pumping	0	15	1·0	5·0
Shaft-top expenses	0	16	4·5	5·5
Management and salaries	1	5	8·0	8·5
Totals	15	0	7·4	100·0
White labour	5	16	5·5	38·8
Native „	4	0	6·6	26·8
„ food	0	11	6·7	3·8
Compound expenses	0	9	0·1	3·0
Timber	1	15	1·9	11·7
Explosives	1	2	10·3	7·6
Lubricants	0	1	0·1	0·4
Miscellaneous stores	0	7	5·6	2·4
Fuel	0	16	6·6	5·5
Totals	15	0	7·4	100·0

Cost of Shaft-Sinking: Machine-Drilling.

For 760 ft.	Cost per ft.			Cost %.
	£	s.	d.	
Wages, white and black	8	7	5·5	45·7
Timber	2	7	10·6	13·1
Explosives	1	16	10·4	10·1
General stores..	0	14	6·1	4·0
Hauling and pumping	2	1	0·2	11·2
General	1	5	6·7	7·0
Shops (drill-sharpening, &c.)	0	5	4·8	1·6
Rock-drills and compressors	1	6	7·7	7·3
Totals	18	5	4·0	100·0

This shaft (p. 284) is an incline, 21 × 6 ft. in the clear, and was already down 1228 ft.; rock, hard chloritic shale; sinking was all done by rock-drill, 6 machines in the face at one time, and 28 holes drilled per round; "sinkers" were got in as deep as possible (up to

Cost of Shaft-Sinking: Transvaal. (A. E. Pettit.)

Name of Mine.	Total Depth.	Size of Excavation.	Cost of Sinking per ft.	Average Number of ft. sunk per Month.	Cost per cub. ft. of Rock taken out.	Area of Excavation.	Remarks.
	ft.	ft.	£ s. d.		s. d.	sq. ft.	
Rose Deep ..	911	23 × 8	31 1 4	—	3 4·5	184	
	714	18 × 8	26 16 4	—	3 8·5	144	
Glen Deep ..	1005	18½ × 8	22 4 2	91	3 0·0	148	
	1017	18½ × 8	24 2 10	66	3 3·3	148	Mostly in dyke.
Durban Rood. Deep	1444	18 × 8	19 10 5	87	2 8·5	144	
Robinson Deep	2389	22 × 8	25 14 7	69	2 11·0	176	Price includes depreciation of sinking plant.
	1923	18½ × 8	24 3 7	69	3 3·2	148	
Vogelstruis Deep	891	18 × 7½	20 14 0	65	3 0·8	135	
	1055	18 × 7½	18 12 4	89	2 9·1	135	
	960	18 × 7½	15 5 9	86	2 3·2	135	Hand labour alone.
Knights Deep ..	1244	28 × 8	36 17 6	55	3 3·5	224	Very wet.
	1217	28 × 8	48 18 1	40	4 4·4	224	,,
Simmer East ..	1925	28 × 8	25 3 9	96	2 3·0	224	
	1802	28 × 8	29 6 0	70	2 7·3	224	Wet shaft.
South Rose ..	2392	28 × 8	21 2 3	78·1	1 10·6	224	Hand labour.
Simmer West ..	3408	28 × 8	21 1 7	129·8	1 10·5	224	,,
Jupiter	3752	28 × 8	20 13 2	135	1 9·2	224	,,

11 ft.), and where almost always blasted before the "lifters" and "side-holes" were started. Each round averaged about 6 ft., so that there were 60–70 tons of broken rock to shift after each blast, reckoning 11¾–12 cub. ft. to the ton. No regular shift hours were kept, drillers and shovellers following each other at any hour of the day or night, as the work demanded.

The main (vertical) shaft of the Victoria Quartz Co., Bendigo, from 4048 ft. to 4300 ft., cost—wages, 62*s.* 6*d.*; firewood, 25*s.*; timber, 13*s.* 3*d.*; explosives, 7*s.* 4*d.*; sundries, 7*s.* 11*d.*; total, 5*l.* 16*s.* Rock, hard slates, sandstones, not hard to drill but blasting badly (lying at about 78°). (W. Richard.)

Detailed Cost of New Kleinfontein Shaft (858 ft.).
(E. J. Way.)

			Cost per ft. sunk. £	s.	d.
White wages			3	16	9
Salaries			0	6	5
Native wages			1	7	9
Compound			0	14	5
Workshops			0	2	6½
Steaming station			1	11	6
Rock-drills			3	1	1
Surveying			0	0	11¾
Transport			0	0	3¾
Travelling expenses			0	0	1
Stores:—					
Description.	Quantity.				
Candles	207·2	boxes	0	3	1
Detonators	86	"	0	0	4¾
Fuse	2,475	coils	0	1	0
Gelatine	418	cases	1	14	1½
Iron bars, etc.	696	lb.	0	0	2
Steel bars	2,528	"	0	1	0¾
Rails	65,445	"	0	7	6½
Sleepers	259		0	0	5
Bolts and nuts	1,263		0	0	5
Dog spikes	363		0	0	2
Fish plates	3,011		0	0	10¼
Nails, assorted	432	lb.	0	0	1½
Coach screws	82			..	
Oils, grease	..		0	0	2
Piping	270	ft.	0	0	6
Tools	..		0	0	3¼
Timber, assorted	1,404	cub. ft.	0	1	8
Sundries	..		0	2	1
Total			£13	15	11

Cost of Shaft-Sinking: Lincoln, California. (Voorhies.)

	Per ft. sunk s.	d.
Labour: 1 foreman, 350 days at 16*s*. 8*d*.; 1 ditto, 282 days at 13*s*. 6½*d*.; and 2956 shifts (8 hr.) at 11*s*. 5½*d*., mining and timbering	53	2
Hercules powder, per ft., 16·8 lb. at 5·25*d*. per lb.	7	4½
Fuse, 48·4 ft. at ·17*d*.	0	8½
Caps	0	3
Candles, 3·2 lb. at 6*d*.	1	7½
Timber, 148 sets at 105*s*.	21	0
Surface labour, engineers, smiths, carpenters, etc.	35	0½
Fuel	33	2
Total	£7 12	4

Dimensions, 17 × 8 ft.; depth, from 1260 ft. to 2000 ft.; country, greenstone and hard black slate.

Cost of Shaft-Sinking: Golden Eagle Mine, Calif. (Benjamin.)

	Shifts.	Prices. s.	d.	Cost per ft. sunk. s.	d.	s.	d.
Labour:—							
Miners (9).. ..	423	12	6	35	3		
Topmen (2).. ..	94	10	5	6	6		
Engineers (2).. ..	94	12	6	7	10		
Smith (1).. ..	47	14	7	4	7		
Foreman (1).. ..	47	20	0	4	9½		
						58	11½
Stores:—							
Timber	10,976 ft.	54	2	3	11½		
Lagging	2,520 „	1	5½	2	5½		
Lining boards	2,270 „	58	4	0	10½		
Cordwood blocking		..		0	5		
Wedges	3,000		0½	0	10		
Wood fuel ..	25 cords	12	6	2	1		
Oil	..	..		0	5		
Kerosene ..	6 cases	17	3½	0	8½		
Candles	6 „	26	8	1	1		
Powder	600 lb.		7	2	4		
Fuse	2,500 ft.	15	5	0	3		
Caps	550 boxes	26	0½	0	1		
						15	6
Total						74	5½

Work was commenced at 7 ft. below the 400 level, and the shaft was sunk 150 ft. from that point; hand-drilling, working 3 8-hr. shifts, with 3 men on a shift; ground taken out 7 × 12 ft. in the clear for double compartment; hoisting by bucket; country, hard andesite; no water; timbered with 10 × 10 in. sawed timbers, end-plates and centre-braces dovetailed in, centre and corner posts gained in; sets placed 5 ft. centres, filled and lagged solid behind timbers, and each compartment lined with 1 × 12 in. lining boards set 3 in. apart; work completed in 47 shifts, making an average of 3·2 ft. per shift, hoisting 20 tons material per shift; timbers framed by hand, by foreman who directed the work; 18 holes drilled per round; No. 2 Giant powder; ground drilled hard, but broke well; outside holes kept high, and centre broken first.

DEVELOPING.

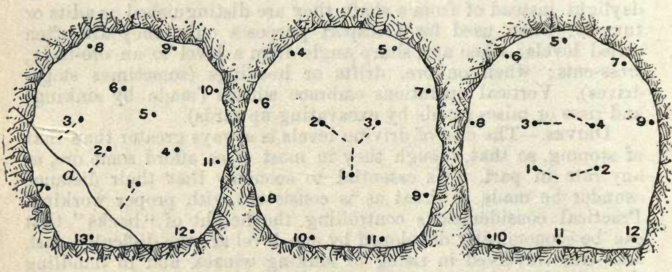
IN systematic mining, before the winning of ore is commenced, the deposit is "developed" or opened out by a series of lateral and vertical workings, so as to afford data for computing what the scale, cost, and profit of operations are likely to be, and to give ready access, so that a steady output may be maintained.

Lateral operations are known as drives; when they start from daylight instead of from a shaft, they are distinguished as adits or tunnels; when used for transport purposes only, they are often called levels; when at a sharp angle from a level to an ore-body, cross-cuts; when on ore, drifts or headings (sometimes stope-drives). Vertical operations embrace winzes (made by sinking) and rises or raises (made by excavating upwards).

DRIVES.—The cost of driving levels is always greater than that of stoping, so that, though they in most cases afford some ore, at any rate in part, it is essential to economy that their distance asunder be made as great as is consistent with proper working. Practical considerations controlling the height of "backs" that can be economically developed by one level are the difficulty, cost, and time involved in rising or sinking winzes, and in handling the ore in long stopes. In many mines, the ore-bodies dip at angles which are too low to allow of the unaided movement of broken rock down the stopes, and consequently a great deal of shovelling is involved in the removal of the ore. It is chiefly owing to the time and labour that would otherwise have to be expended on this operation that the intervals between successive levels seldom exceed 130 ft., and are in many cases as little as 50 ft., though 100 ft. is perhaps the most common average. Many of the S. African mines range from 100 to 200 ft., and have a mean of 140 ft. on the reef-plane; but in the "deep" mines, there is a growing tendency to increase this to 300 ft. for the main levels, and, if necessary, intermediate levels will be added so as to afford about 150 ft. of backs.

Drives are usually carried in the reef itself, so as to expose it as much as possible (in order that some idea of its value may be obtained by sampling), and to make its productiveness in some measure compensate for the cost. In exceptional cases, however, the ground stands so badly that it is unwise to make a roadway on the vein, and then the level is driven parallel with it, and cross-cuts are put out to the vein at intervals for measuring and sampling purposes. Where a reef is small and rich, the level is so driven that the reef is just exposed in the hanging-wall corner of the drive, and as little as possible of it is broken; with a preponderance of waste, the ore which is broken in driving becomes almost worthless from admixture of barren rock, or is entirely hidden and lost, especially as ore is generally pulverised much more easily than the neighbouring country.

Size.—Dimensions of drives vary somewhat with the potentialities of the mine. On small undertakings, operated by hand labour, and never calculated to put out large quantities, the size may be kept down to a minimum that will accommodate little trucks for manual service, for the sake of economy, however trifling. The least dimensions in which hand-drilling can be conveniently done are 4 ft. high and 3 ft. wide. Where machine-drills are

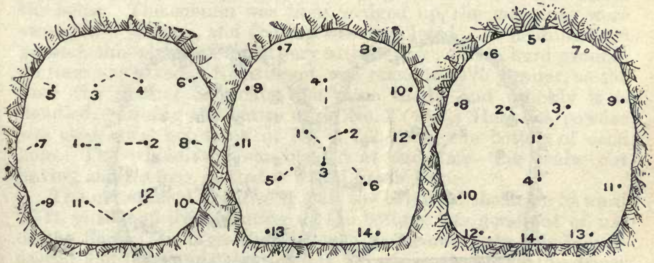


FIGS. 79, 80, 81.—EXCAVATING DRIVES.

employed, the amount saved by reducing the area of the drive a foot or so either way is of no moment; and if the output is to be large, there is appreciable ultimate gain in having so much room in the level as will admit the largest-sized truck that can be handled, and will give abundant facility for rapid transport. For single-track levels, common measurements are $6\frac{1}{2}$ to $7\frac{1}{2}$ ft. high in the gross (making about 6 ft. in the clear between roadway and roof) and about 5 ft. wide, the width being contracted somewhat above the middle of the drive, to give additional stability overhead, and sometimes under foot as well, though frequently the full width is needed to make room for gutter, air-mains, etc. For double tracks, the height is increased to 7 or 8 ft., and the width to 9 or 10 ft. While the cost per linear ft. is increased by augmented width, the cost per cub. ft. is lessened, timbering is not materially added to, and the gain in speed is marked.

A gentle gradient is necessary to induce a natural flow of water towards a pump-station, and it is always contrived where possible that this shall accord with the direction in which loaded trucks are to travel, so that it will facilitate tramming. The most suitable grade for the latter purpose, keeping in view the up-hill return of the empty trucks, is a fall of about $\frac{3}{4}$% (1 yd. in 133, or 1 in. in 11 ft.) towards the shaft or adit mouth, but sometimes this is increased to 1 yd. in 114 or 1 in. in $9\frac{1}{2}$ ft. Water will readily flow at 1 in 150 (1 in. in $12\frac{1}{2}$ ft), if the drain is kept reasonably free from obstructions.

Excavating.—In breaking ground, much the same principles apply as in shaft sinking (p. 242). Where a bedding-plane or joint presents itself, it is generally utilised in taking out the cut, though this is so to a much less extent with machine-drilling than with hand-work. An example is shown in Fig. 79, where the cut-holes 1 to 3 are inclined towards the joint *a* and fired together; the cut



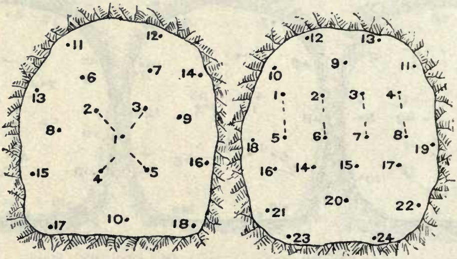
FIGS. 82, 83, 84.—EXCAVATING DRIVES.

is next enlarged by means of holes 4 to 6, also blasted simultaneously; and these are followed by the outside holes 7 to 13, the order depending partly on how the ground has broken, and partly on the facilities for clearing away the fallen rock, but in most cases leaving the bottom holes till last, so that they may loosen the mass for the shovellers.

With machine-work, the cut is generally made more or less central, the number and arrangement of the holes depending on the tightness of the ground. Figs. 80 and 81 are examples of schemes in easy ground: in the former, converging cut-holes 1 to 3 suffice, and are followed by the outsides, without the aid of any enlarging holes, consecutive numbers (opposite sides) being fired as pairs; in the latter, cut-holes 1 to 4 are drilled to meet or stagger in pairs, a back-hole 5 coming next, then the outsides 6 to 9, and finally the lifters 10 to 12 together.

Arrangements for ordinary ground are illustrated in Figs. 82, 83 and 84. In Fig. 82, a long vertical cut is made by 2 pairs of holes 1 to 4, inclined as shown and fired in pairs; the outsides follow in pairs in rotation, being all inclined slightly outwards to prevent a tendency to contract the face; the risers 11 and 12 are given a downward pitch and converge. In Fig. 83, a more circular cut is made by converging and simultaneously-fired holes 1 to 3, followed by easers 4 to 6 still looking inwards, and then by the outsides in sequential pairs. Fig. 84 differs in placing a short central hole 1 to help the cut-holes 2 to 4, which may be 6 ft. deep and almost meet behind 1; they are fired as a group, but No. 1 should go first; No. 5, in the back, paves the way for other outsiders in pairs right and left, finishing as usual with 3 lifters.

In very tight ground, the number of holes may run up to 18 or even 24, and such distributions as are shown in Figs. 85 and 86 are adopted. Fig. 85 indicates a vertical cut with 5 holes, the central



FIGS. 85, 86.—EXCAVATING DRIVES.

one being about ⅔rds the depth of the other 4 which converge behind it; 6 to 9 are easers, fired in pairs, and are followed by the outsides, 11 to 16, in the same fashion, last coming the lifters, 17 and 18. Sometimes 10 forms one of the easers, and sometimes it is the final hole, according as the ground breaks. A horizontal V-cut of 8 holes, as in Fig. 86, is occasionally better suited to the ground, though it may mean breaking wider than usual; an easer 9, and outsides 10 to 13 in pairs, will probably be fired and in part cleared away before the remaining holes are charged.

An interesting description of driving a 7 by 8-ft. tunnel in the Melones mine, California, is given by W. C. Ralston (1898). It was 2608½ ft. long, and was advanced on an average 10·22 ft. per diem, passing through diabase, brown slate, and heavily-mineralised talc-schists with quartz stringers The working force consisted of 29 men, divided into 3 8-hour shifts of 7 each (4 machine men and 3 shovellers on each shift); 2 drivers with horses working 12 hours; 2 engineers working 12 hours; 1 blacksmith, 1 helper, 1 mechanic, and 1 outside man, working 10 hours. The machine men worked at drilling only, and, to save time, 2 machine drills were mounted on one column placed horizontally across the tunnel as near the top as possible. This permitted the men to fix the column and get to work immediately after blasting, without waiting for the dirt to be shovelled back from the face; and by the time the shovellers had the rock cleared away, the machines could be swung under the column to put in the centre cut and middle side holes. After these holes were drilled, long drills were inserted in the machines, which were then swung to a vertical position, with the drills resting on the bottom of the tunnel; the wedges were then removed from the column, and the whole weight now being on the drills, the machines were cranked down until the column was in a position low enough to drill the balance of the holes. The column was then wedged up, the machines were swung into position, and drilling was completed. In medium hard ground, this plan was found very satisfactory; but in hard ground, no time was saved. No attempt was made to save powder, as the finer the ground is broken the more easily and quickly it is handled. During the entire work, No. 2 (40 %) Hercules powder was used, with one stick of No. 1 (60 %) in the bottom of each hole. The whole face was blasted at one time—the centre cut having shorter fuse, so that it would break first.

The old method of carrying half the height of the drive forward 20 ft. or so, and then blasting up the bottom, has gone out of use on the Rand. In high tunnels, such as for railroads, this is successful, but it is a mistake to apply it in a mine, where the drives and cross-cuts are only 6 ft. high.

It is common for rock-drill men to work two machines on one bar, and, in order to expedite drilling, a short horizontal bar is rigged before the dirt is cleaned out, and drilling commenced on the back or top dry holes. These holes are well advanced by the time enough broken ground is thrown back to permit the erection of the vertical bar.

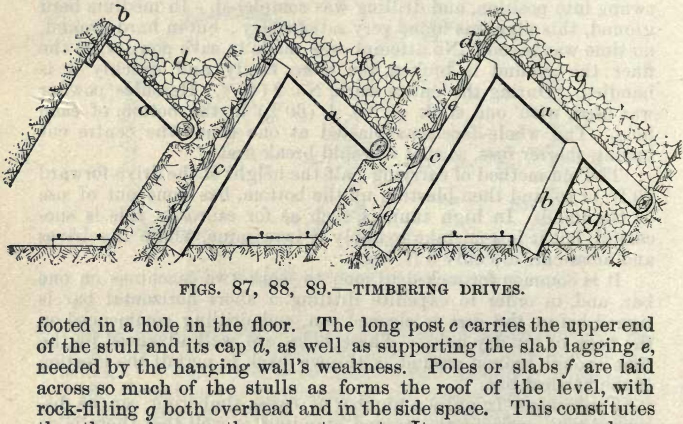
Timbering.—In quite a number of cases, timbering can be dispensed with entirely, the ground standing so well that there is no risk of fall from either walls or roof; but it is advisable to keep a constantly watchful eye on the latter, lest pieces of rock become loose by winding, and need taking down ere they fall.

With good walls, and not too great a width, a single piece or quarter-set may suffice; but sticks much exceeding 16 ft. in length are very awkward to handle, and too weak for resisting a lateral strain of any moment. The arrangement is shown in Fig. 87. The foot of the timber *a* is let into a hitch cut in the footwall, and its upper end supports a cap or head-board *b*; pole lagging *c* is laid lengthwise of the level, and is well covered with rock filling *d*.

both to prevent a squeeze from pushing up the top end of the stull *a*, and to shield the lagging *c* from falls of rock.

When either wall is incapable of providing a sound hitch for one end of the stull, a leg or post is added. Thus in Fig. 88, the stull *a* is footed in the footwall, while its upper end and the cap *b* are carried by the post *c*, forming a two-piece or half-set, *c* being notched into *a* as shown. The hanging wall is weak, and requires to be lagged with slabs *d*; smaller slabs (or poles) *e* are laid across the stulls, and sufficient rock filling *f* is used to make all secure.

If neither wall can be relied on, two posts will be required. In a level of no great width, these posts may be at each end of the stull; but if the width is excessive, and only partially required for a gangway, the second leg may be at some distance from the wall. Such an example is given in Fig. 89. The stull *a* is hitched into the footwall, but a short post *b* takes most of the weight, and prevents sagging in the middle; it is notched into the stull, and



FIGS. 87, 88, 89.—TIMBERING DRIVES.

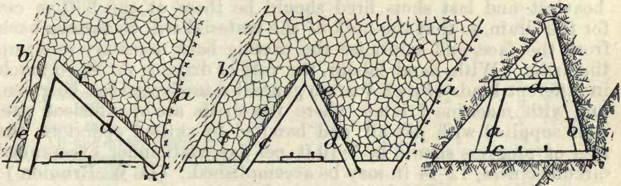
footed in a hole in the floor. The long post *c* carries the upper end of the stull and its cap *d*, as well as supporting the slab lagging *e*, needed by the hanging wall's weakness. Poles or slabs *f* are laid across so much of the stulls as forms the roof of the level, with rock-filling *g* both overhead and in the side space. This constitutes the three-piece or three-quarter set. It consumes a very large quantity of timber, and that mostly in long pieces.

A much stronger and more economical arrangement of timber for a level of unusual width was successfully adopted by the author in an Australian mine, and is shown in Figs. 90, 91. The footwall *a* is diorite, and stands very well, but the hanging wall *b* is serpentine, full of "soapy heads," and "winds" very badly, coming away in masses of ½ cwt. to ½ ton. The post *c* is footed in rock, and held at the top by the strut *d*, which is hitched into the diorite; substantial lagging *e* keeps up the serpentine wall; and the lagged roof *f* is well covered with waste 3 or 4 ft. thick, obtained by breaking from the hanging wall, if not otherwise available. Where the width becomes too great for the strength of *d*, both posts *c d*, as in Fig. 91, are footed in the floor, and inclined to each other at the top, where they are scarfed and sometimes bolted together; lagging *e* and rock-filling *f* are added as usual. The timber used was box and red gum, 10 in. diam. After 5 years the work stood as good as new.

Fig. 92 illustrates a complete set, the legs *a b* resting on a sole plate or sill *c*, and the stull *d* being hitched at one end in the hanging wall, and forming a stay about midway in the length of leg *b*, to resist the pressure of the vein or packing behind it; *e* is filled with waste.

With full sets there is so much more labour entailed in fitting and joining, and these operations are so much simpler with square than with round timber, that the former is generally employed.

When the level accommodates a double track, the stull-piece is reinforced by a post midway separating the tracks.



FIGS. 90, 91, 92.—TIMBERING DRIVES.

In loose ground, recourse must be had to spiling or fore-poling, which is conducted on exactly the same principles as the corresponding operation in shaft-sinking (see p. 273), viz. by driving planks or logs, pointed at one end and sometimes iron-shod at the other, over the cap of the last set, and in advance of the heading, giving them an upward tendency. When about half-set distance has been covered, a false or dummy set is introduced to relieve the spiling of some of the weight, and then the work is advanced till the time comes for inserting a true set. It is often necessary to join the last 3 or 4 sets with stout iron dogs, removing them as the settling ground fixes each set firmly in place. Sometimes the spiling must fit very closely, and packing of moss, stringy bark, or old bagging is required to keep the fine dry sand from running into the drive.

Arch sets, as in Fig. 93, are both easier to erect and stronger than square sets.

In hand-driving, the rate of progress is usually in inverse proportion to the hardness and toughness of the rock, but where air drills are used, the retardation due to excessively hard rock is not so severely felt. With heavy drills, better progress can be made in the hardest rock than in soft material where the timbering has to be kept so close to the face that heavy blasting is impossible. Heavy blasts, where possible, are the key to rapid progress in tunnelling. When sufficient powder is used only to break the rock, and the material comes out in unwieldy pieces, the time required for loading is prolonged; it is better and more economical to use enough explosive to thoroughly shatter the rock, so that shovels can be forced into it as if it were sand or gravel [that is, where labour is high and explosives are cheap, as in the United States]. The heaviest and last shots fired should be those at the bottom cut for the drain, so that the bulk of the material will be thrown back from the face, and minimise the casting back necessary to set up the drills. With hand labour, the rate of driving varies so much in different kinds of rock that no general rate of speed can be given, but with machine driving, where the drills are of sufficient size, well supplied with dry air, and handled by skilled operators, the rate of progress should be 7–10 ft. per day, and, under exceptional circumstances, 12–14 ft. may be accomplished. (D. W. Brunton.)

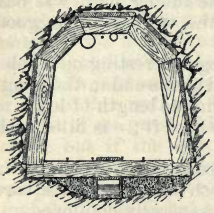


FIG. 93.—TIMBERING DRIVES.

Cost.—So many conditions go to determine the cost of all mine work that a wide range of figures is encountered.

At the Lucknow mines, New South Wales, with wages at 7*s.* 6*d.* per 8-hour shift, the author found the cost of driving in the serpentine, along a good "joint," and just breaking the diorite footwall, to be about 24*s.* per ft.; with hand labour, sometimes as low as 17*s.* 6*d.* In driving through diorite, air drills were generally employed, the cost of passing through ordinary rock of this class being about 30*s.* per ft. On the other hand, in some varieties of the diorite, short distances cost up to 5*l.* per ft. with machine drills. As an example of very hard and tough rock, cases may be cited where it has taken as much as 16 hours to get down a hole of 30 in., hundreds of drills being blunted; 10 drills to an inch of progress may have to be used at starting, and often a hole has to be abandoned and another commenced.

At another mine (Myall's Reef) in the same Colony, with a reef 2–20 ft., and averaging 8–9 ft. wide, driving cost 20–30*s.* per ft.

During operations in the Potsdam sandstone beds of the Black Hills, in 1894, with wages ruling at 14*s.* a day, the author paid contractors generally about 10*s.* to 12*s.* 6*d.* per ft., miners furnishing all their own supplies except timber when needed.

In the Transvaal, contract prices (men finding all supplies, but company furnishing machines and air) mostly range from 35*s.* to 47*s.* 6*d.* per ft. On the average, a 4-ft. hole takes 5 plugs, and a 5½-ft. hole, 6 plugs of blasting gelatine; in ordinary ground, a squared cut of 5 ft. consumes some 50 lb., and in very tight ground 60 to 75 lb. blasting gelatine. Allowing a mean sectional area of drive to be 36 sq. ft., and length of cut 4½ ft., the normal consumption of blasting gelatine, using machine drills, is 3½ lb. per ton of rock excavated.

According to Hatch and Chalmers, the cost in blue ground ranges from 45*s.* to 65*s.* per ft., including track-laying. These authorities consider hand-drilling in drives 15 to 25% cheaper than machine, and give the cost of explosives at 4*s.* 6*d.* to 6*s.* 6*d.* per ton broken.

Tunnel Costs: California. (W. C. Ralston.)

	Melones.		Hogsback.	
	s.	*d.*	*s.*	*d.*
Labour	31	1½	32	4½
Powder	5	5	3	9
Fuse and caps	0	9½	0	5
Firewood	2	7½	3	10
Coal	0	3	..	
Charcoal	..		0	10
Water	1	4	..	
Planks, ties and timbers	0	3	0	6
Candles	0	5	0	7
Steel rails	0	11	2	1½
Air and water pipes	1	10½	2	0
Horse feed	0	5	0	11
Steel, tools, etc.	0	6	2	5
Total	45	11	49	9

Their dimensions were alike (7 by 8 ft.), and the average hardness of the ground did not vary materially, though in the Hogsback it broke bigger, consuming less explosive, but creating more work for shovellers. At the Hogsback, all men received 12*s.* 6*d.* a day; at Melones, 6 miners got 12*s.* 6*d.*, 15, 10*s.* 5*d.*, and carmen. 8*s.* 4*d.*

WINZES AND RISES.—At variable intervals, to secure proper ventilation and to open out the ground for stoping, connection is made between the levels. These constitute winzes, and they may be made either by sinking or by rising or by a combination of the two. In general, perhaps, sinking is more common than rising, at any rate so far as the broken rock (and water) can be conveniently handled. With machine drills, sinking is rather more costly than rising; but often with hand labour, and particularly so with black or yellow labour, sinking is cheaper. In many mines a difficulty in the way of economic rising is the heated condition of the atmosphere where the men are working, but this may be at any rate partially remedied when air drills are employed. Sometimes the relations of development to mill-demands do not permit of waiting for a level to be run before starting a winze upwards, and then sinking has to proceed concurrently with the driving; but where the conditions allow of rising, the work can be done better, more cheaply and more quickly, if the "cribbing" or "box-rise" system, described below, be adopted.

As to distance apart, no rule can be laid down. Every additional winze adds to the cost of development, and, where the call for starting points for stopes is not urgent, there is a tendency to increase the distances between them to 500 or 600 and even to 800 ft.; but in some instances this will be a false economy, because of excessive shovelling involved in removing the broken dirt. About 200 to 400 ft. is a very general average on the Rand.

Dimensions vary in like manner. A good working size is about 6 × 4½ ft., but it would not be good policy to leave ore on the walls, supposing the vein to be more than 6 ft. thick, nor would

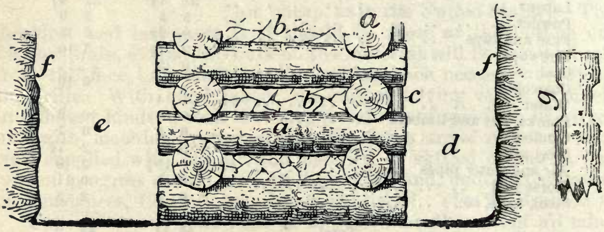


FIG. 94.—BOX-RISE.

anything more than was absolutely necessary be cut out, if the vein was small or the country very hard.

The "box-rise" consists in the adaptation of "cribbing" or "penning" to the needs of the situation. When the rise has reached to a convenient height above the level, a crib or pen is built up, dividing it into 2 or 3 compartments, according to circumstances. At one end in any case is left a man-way. If the vein is large, and all material broken will be milling dirt, all but a small portion of the remainder of the rise will be occupied by the crib; if very little ore is anticipated, the same rule will apply; if something like equal quantities of ore and waste are expected, more space must be left outside the crib. The object to be attained is that the ground broken, whether ore or waste, shall serve to keep the crib constantly full, and, when the conditions permit of shooting ore and waste separately, or nearly so, to keep them at once apart and obviate handling. The crib is built of short logs laid as shown in Fig. 94, and with just enough of a notch cut in the corresponding upper and lower sides of each to ensure their security from displacement, either by accidental force from without, or from pressure of the rock-filling within; *a*, logs; *b*, rock filling; *c*, ladder; *d*, man-way; *e*, open pass for ore or rock; *f*, walls; *g*, manner of notching logs. At a suitable height for truck-filling, the crib is provided with a sloping iron-shod bottom, a shoot, and a door. The ladders in the man-way are secured to the logs forming one side of the square crib. On two other sides, the crib is wedged quite tightly between the standing walls. Short ends of heavy timber are kept at the top of the crib for covering the man-way when firing, and machines, etc., are secured there. Sufficient ore or waste, as the case may be, is drawn from the shoot in the crib before firing to leave accommodation space for all that is likely to be broken. Men and machines have thus always a firm foundation to work from, the cribbing being carried up as the winze rises. When the great bulk of the material broken is of one class only, and the third compartment of the rise is left open, a very simple means of ventilation is at once afforded, by placing a door in the level and causing the air to circulate through the winze, rising up the man-way and descending on the other side. In the author's experience this has proved most efficient in cases where previously men could not work at all, even with air-drills. When two classes of dirt (ore and waste) are broken in considerable quantity, economy of handling may be secured by building a second shoot in the third compartment. This, however, prevents effective ventilation by natural means, and, in such a case, it will probably be found necessary to keep a jet of compressed air constantly blowing. A highly satisfactory aid to this is to carry up the man-way a 1-in. gas-pipe filled with water under considerable pressure, controlled by a cock, and furnished for the last 20 ft. or so with rubber hose and a fine rose or sprinkler. This freshens the air and condenses the fumes after firing, costing nothing but an occasional hour's labour for the pipe-fitter, and interest on the price of the pipe. The additional volume to be pumped is not worth mentioning, and in many cases there is none, it being absorbed by the dry broken rock, and thus moderating the dust nuisance which is sometimes a serious matter.

On the Rand, winzes have been put down over, under, and on the reef. Under the reef is probably best, and a distance of 20–30 ft. seems most desirable, and is sufficient to leave the road intact after working out the reef above it. The angle of a winze should always be greater rather than less from the horizontal than the dip of the reef, to allow for any possible variation in the formation, as it is simpler and better mining to flatten an incline than to increase the angle. Inclines are usually sunk with machine drills, from two vertical bars, but in some places a horizontal bar (with a central support) reaching the whole width of the shaft has been used. (A. E. Pettit.)

GENERAL COSTS.—There is so much variation of practice in segregating mine accounts that comparisons must always be regarded as having only approximate values, yet they are interesting.

Two groups are selected for illustration—the Rand and the Lucknow mines (N.S.W.). The former is distinguished by a large and consistent ore-body, well-marked regularity in the features of the country, operations conducted on a large scale, machinery of the best and newest type, fairly cheap native labour, fuel of medium quality, and explosives at very costly figures. The latter offer a great contrast, the ore-bodies being exceedingly erratic in size and disposition, the country ranging from the hardest diorite to soft and even rotten serpentine, operations on a lesser scale, machinery old and wasteful of power, labour expensive, fuel consisting of dirty coal eked out by wood, explosives of best quality and at reasonable prices. In both cases the sinking of main shafts and the outlay on general mine equipment are charged to other accounts (generally against "capital"), and the debits to "working" embrace only drives, cross-cuts, winzes and rises, these forming the sum of the development work properly incidental to maintaining the output of the immediate future (1 or 2 years).

Transvaal.—The average cost per ft. driven, sunk and risen in the blue quartzites at 300 to 1500 ft. deep is quoted by Campbell at 6*l.* 15*s.* 6*d.*; and the apportionment of this total is—white labour, 40 %; native labour, 16; explosives, 22; fuel, 9; stores, 10; general expenses, 3.

Truscott quotes the following:—

Crown Reef: 10,232 ft., at 3*l.* 3*s.* 6½*d.*

Ferreira: 6406 ft. (3651 ft. drives, 740 cross-cuts, 1566 winzes and rises, 449 shafts), at 4*l.* 12*s.*

Geldenhuis: 7096 ft. (4295 drives, 839 cross-cuts, 1064 winzes, 449 rises, 449 shafts), at 3*l.* 11*s.*

George Goch: 9655 ft. (6873 driven by air-drills), at 3*l.*

Henry Nourse: 8650 ft. (6580 drives, 1613 cross-cuts, 457 winzes and rises), at 4*l.* 3*s.*

Meyer and Charlton: 3758 ft. (2081 drives, 326 cross-cuts, 630 winzes, 721 rises), at 3*l.* 3*s.* 9*d.*

Princess: at 2*l.* 15*s.*

Robinson: 11,193 ft., at 2*l.* 16*s.* 8½*d.*

Simmer and Jack: 25,708 ft. (17,003 drives, 2597 cross-cuts, 2706 winzes, 3402 rises), at 4*l.* 18*s.*

Development Costs: Transvaal.

	Meyer and Charlton. Cost per ft.	Princess. Cost per ft.	Robinson. Cost per ft.
	£ s. d.	£ s. d.	£ s. d.
Labour, white	2 3 10½	0 15 6	0 16 4
" native		0 5 9	0 5 8
Explosives	0 4 3	0 13 6	0 16 4
Tools, etc.	0 12 3½	0 8 6	0 5 10
Smithing	0 3 4	0 1 9	0 3 3½
Fuel	*a* ..	0 10 0	0 9 3
Totals	3 3 9	*b*2 15 0	*c*2 16 8½

a Included under "tools."
b Compressed air costs 7*s.* per ft.; tramming, 2*s.* 3*d.*; both included in above.
c Compressed air costs 7*s.* 9½*d.* per ft.; tramming, 2*s.* 8*d.*; hoisting, 3*s.* 2*d.*; all embraced in above.

Development Costs: New South Wales.

	1899. Cost per ft.	1898. Cost per ft.	1897. Cost per ft.	1896. Cost per ft.
	£ s. d.	£ s. d.	£ s. d.	£ s. d.
Labour	1 18 7	2 12 9	2 11 3	2 6 5
Fuel	0 4 10	0 7 5	0 3 8	0 4 0
Explosives	0 2 7	0 3 3	0 3 8	0 4 8
Timber	0 0 11	0 0 10	0 1 4	0 1 0
Sundries	0 0 10	0 1 1	0 1 8	0 1 8
Totals	2 7 9	3 5 4	3 1 7	2 17 9
	3935 ft.	4343 ft.	6695 ft.	8962 ft.

NOTE.—Labour: miners, 7*s.* 6*d.* per 8 hours; shovellers, 6*s.* 6*d.* Fuel: coal (weak and carrying 12% ash), 14*s.* per long ton; wood, 10*s.* per cord; steam pressure, 45 lb.; air compressor 55 lb.; transmission, 2000 to 5000 ft. Explosives: mostly gelignite, at about 1*s.* 6*d.* per lb.

Development Costs: Center Star, Rossland.

—	Cost per ft.					
	Sinking Winzes.		Rising.		Driving.	
	s.	d.	s.	d.	s.	d.
Drilling	31	8½	31	8½	22	8
Blasting	7	4½	8	6½	3	11½
Explosives	14	1½	13	7½	11	8½
General mine supplies	6	5	4	0½	2	9½
Mine lighting—candles	1	5½	0	8	0	8
Mine lighting—electric	1	1½	0	9½	0	8
Smithing	4	11	3	10	2	10½
Shovelling—direct	24	8	3	5½	4	7½
Shovelling—apportioned	2	3½	2	0	2	4
Timbering—labour	5	1½	12	8½	0	8½
Timbering—material	0	6	3	1	0	2½
Machine drill fittings	5	1½	4	3	2	8
General mine labour	18	0½	11	9½	8	9½
Hoisting underground	30	3½	..		0	8
Hoisting main shaft	3	10	2	11½	2	8½
Compressed air	5	8	6	5	4	5½
Mine ventilation	3	4½	1	11½	1	4½
Assaying	1	5	1	3½	0	9
Surveying	1	0	0	11½	0	8½
General Expense	18	9	15	5	9	6½
Total	£9 7	2½	£6 9	6	£4 4	10½

MINING METHODS.

No strict and rigid classification of the methods adopted for mining metals, ores, and other minerals, is possible, because it frequently happens that a mine which, at the start, would fall into one category, during later periods of its history passes through several others. But convenient for arranging descriptions of the various operations is the grouping into three main headings—Superficial deposits, Beds, and Veins.

SUPERFICIAL.

This section is pre-eminently concerned with alluvial mining, especially for gold and tinstone, the term "alluvial" being employed in its widest sense to include detrital deposits. It admits of sub-division into Shallow Open Mining, Drifting, Frozen Ground, Hydraulicing, and Dredging.

Shallow Open Mining.

This can scarcely be termed "mining" at all, it being simply digging or raking in most cases. Yet a vast proportion of the world's mineral output is obtained in this way, and it cannot be ignored. It has been very briefly mentioned already under Man Power (pp. 2-3).

In gold-mining (U.S.) it is reckoned that an average gang of 6 men can shovel 15–20 cub. yd. of gravel per 10-hr. day, in two stages, from the face to the sluice-box.

In tin-mining (Malaya) about ⅔ of the total world's supply is obtained by Chinese labour with hoe and carrying basket, as seen in Fig. 95. The waste of labour in long carries, erecting stagings and gangways, and lacing-up tailings, is fabulous. Coolies are paid 2*s.* and upwards for 7 hr. work, and do not average more than 2 cub. yd. each per diem; so that working costs, with a depth of washdirt of 20–60 ft. and a mean carry of 50 yd., reach 16–20*d.* per cub. yd. In Banka (tin), it is reckoned that 2 men, 1 filling and 1 carrying baskets, can excavate about 10 cub. yd. per diem = 5 cub. yd. per man; but it is admitted that ordinary mines remove 10,600–14,000 cub. ft. (= 390–520 cub. yd.) per ann. per man on pay-roll: at 300 days per ann. = 1·3–1·73 cub. yd. per man per diem. In Sarawak (gold), coolies are paid 27c. (6·8*d.*) per cub. yd. for excavating clay or earth; and in Malaya (tin), 25c.

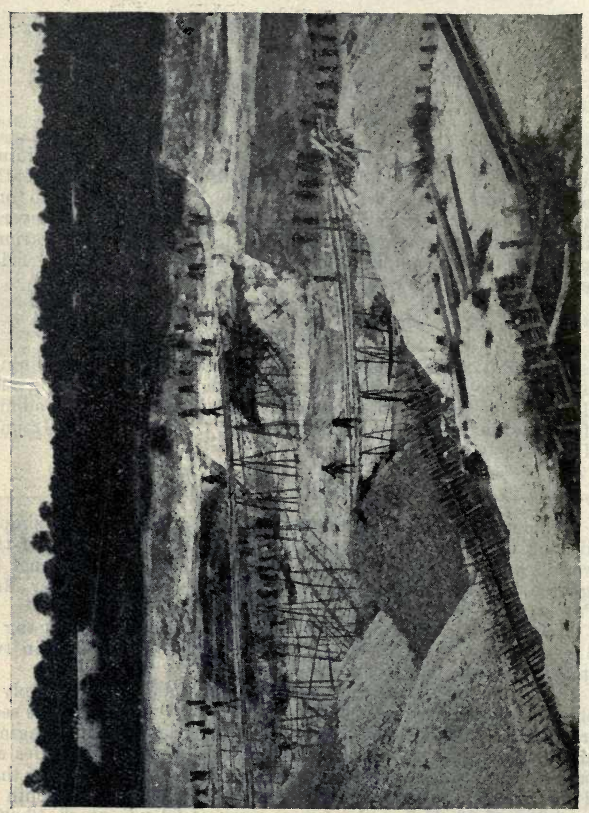


FIG. 95.—CHINESE TIN MINE, MALAYA.

(6·75*d.*) for removing overburden up to 30 ft. deep; or, if food (cooked rice) is provided, 15c. for overburden and 19c. for wash-dirt, 6 hr. a day, rainy days paid for (no work done).

Where sufficient fall and dumping ground are not available for ground-sluicing on bed-rock, the system shown in Figs. 96, 97, is adopted. The upper stratum of gravel is regarded as worthless overburden, and therefore is washed away rapidly in advance of

FIG. 96.—CHINESE TIN MINE, MALAYA.

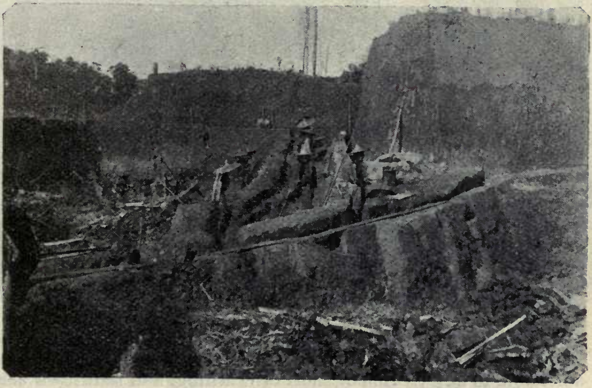


FIG. 97.—CHINESE TIN MINE, MALAYA.

the washdirt. All sluicing, both of overburden and of washdirt, is done by means of little channels cut in the top of the washdirt itself. The overburden is easily hoed down into the sluices, and got rid of without passing through the ditch where the tinstone is saved. But the washdirt is taken out in successive little paddocks,

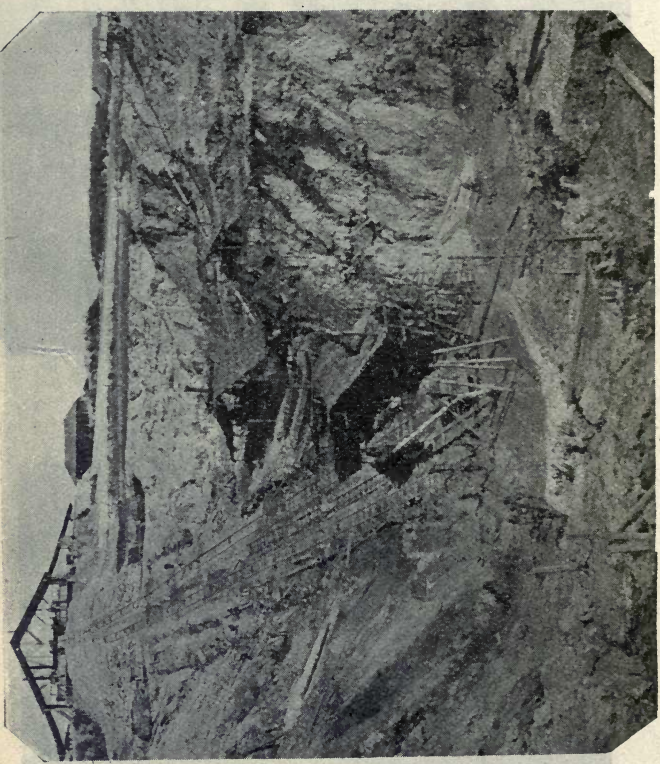


FIG. 98.—TIN MINE, MALAYA.

and shovelled up into the sluices. The ground is very tough, and in part consists of a layer of iron-cemented wash, which is not amenable to any treatment short of crushing, and is thrown aside. The working costs are higher than usual, but, failing a much more abundant water supply for hydraulic sluicing and elevating the tailings, no better way of working (without machinery) could be devised.

In one or two of the larger Malayan tin mines under European management, there is the usual swarm of Chinese or Tamil coolies hoeing and basketing the washdirt, but they carry it only to trucks, running on tram-rails and hauled to the washing plants by steam engines actuating drums and ropes, as in Fig. 98. Trucks contain 16–27 cub. ft.; and costs are about 20–22*d.* per cub. yd. when the workings are 80–100 ft. deep Contract prices to coolies are 30–35c. (8½–10*d.*) per cub. yd. to fill and tip trucks.

Both for the stripping of overburden and the removal of the washdirt in Alaskan gold-mining, also, machinery is largely employed. The method particularly in vogue is called "derricking."

Derricking.—Hand labour is used in excavating, while transport of washdirt and disposal of tailings are accomplished by derricks (Fig. 99) An area of 30 ft. beyond the end of the derrick boom *a* is worked. A pit *b*, roughly circular, 140 ft. diam., is the result, since the boom reaches approximately 40 ft. in its sweep, and the buckets *c* are hauled 30 ft. from the bank. Under a stratum of sand and soil, 4–5 ft. thick, the gravel is usually 30 ft. thick, and for the most part small, not over 10% exceeding 6 in. diam., and nothing above 18 in. Excavation is accomplished entirely by pick, the gravel being shovelled into the derrick buckets. These are hauled by the derrick line, guided by hand, upon wooden skids *k*, to a point directly beneath the end of the boom, where they are hoisted and carried to the dump box *d*. They are of 11 cub. ft. capacity, and are made of crude oil drums or gasoline tanks, cut to a height of 2 ft. 8 in., and 2 ft. 5 in. across the top. Two lugs to hold the bale are set opposite each other ¼ distance up from the bottom. The bale is made from the original hoops of the drum. The bottom edge is strengthened by the original flange of the tank, while on the upper edge is riveted the flange originally at the top of the tank. The bale is supplied with a catch which, when the bucket is travelling, rests in a notch constructed on its edge, which holds the bucket in an upright position. On reaching the dump box, a man on the platform with his shovel frees the catch, and the bucket dumps in turning bottom upward. In fitting the lugs holding the bale, a piece of iron 9 in. square is riveted to the inside of the material composing the drum, which is ⅛ in. thick. To the outside of the drum, a strip 9 in. long, 2½ in. wide, and ½ in. thick, is also riveted, and to this the lug (2 in. long) is welded, making a very strong construction. These buckets weigh 140 lb.

The derrick is very simple and practical, having but one haulage line. It leans towards the pit, i.e. when at rest, the boom swings away from the hoist. The hauling line passes through a block at the foot of the mast, and a few feet in front of it, or toward the direction of haul. In this position, immediately upon receiving a hauling strain, the boom tends to move toward the dump box, and once there, having disposed of its load returns by gravity to the pit. To the end of the boom is attached a rope, by which a man in the pit can regulate the swing of the boom to a nicety. By snubbing this line about a post set firmly in the ground,

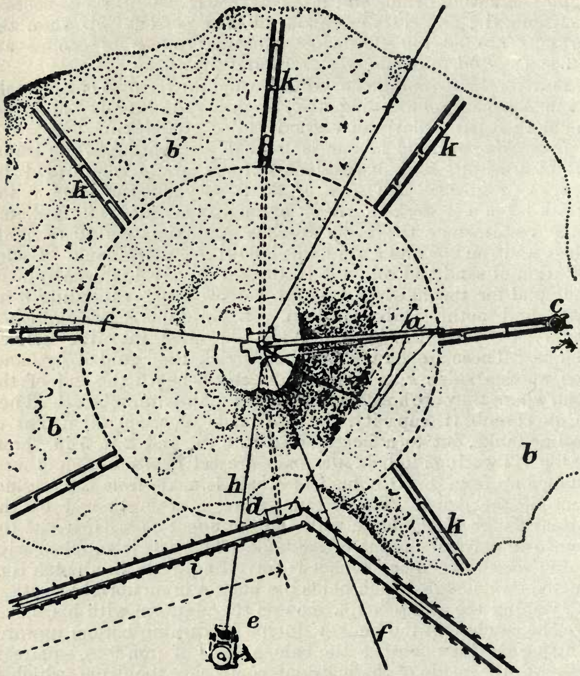


FIG. 99—DERRICKING.

hauls can be made in any direction, with no inconvenience from the swinging of the boom, and the dumping of the load on the platform can be accurately adjusted. The derrick on an average raises material about 25 ft. and carries it 85 ft. to the sluices, handling generally 500 cub. yd. in 22 hr. Power is obtained from a 15-h.p. boiler and 8-h.p. double hoist *e*, burning ½ cord of wood in 24 hr.

Both boom and mast are of fir, 40 ft. long. The hauling line *h* ($\frac{1}{2}$ in. diam.) is of crucible steel wire; the guy lines *f* ($\frac{3}{8}$ in. diam.), also of steel, are tightened by watch tackles. About once a month, 18 hr. are wasted in moving forward 110 ft. The stacking of tailings may be effected by a similar plant. The dump-box *d* feeds the sluice *i*.

Sometimes the bucket is replaced by an iron skip holding $1\frac{1}{4}$ cub. yd., run on trucks to the working faces, and this gives 15–20% greater capacity.

Derricking is a simple, efficient, adaptable, and comparatively cheap method of working open cuts where gravel must be first shovelled by hand, and the bed rock cleaned up afterwards.

Scraping.—A very common excavating appliance in America is the "scraper."

Ground which can be worked by men shovelling into sluices can, under certain conditions, be worked satisfactorily by horse scraping (see also p. 12). The most important governing condition is the looseness of the gravel and bed-rock. Two horses, or, better, mules, with driver, will scrape 30–40 cub. yd. of gravel a day over a distance of 75 ft. Generally, the horses travel in an elliptical track, passing the end of the tail-box and scraping the tailings from it, then entering the pit, scraping up the washdirt, and afterwards delivering it to the sluice.

Steam scrapers have also come much into use, both on washdirt and on tailings. For the latter, their capacity is $\frac{1}{3}$–$\frac{1}{2}$ cub. yd., and a 25–30-h.p. engine handles about 250 cub. yd. loose material per 24 hr., employing 1 fireman, 1 hoistman, and 1 or 2 men to fill, guide, and dump. The arrangement is shown in Fig. 100: *a*, 25-h.p. double-drum hoist; *b*, anchor, which by lengthening or shortening permits position of scraper to be varied; *c*, scraper; *d*, $\frac{3}{4}$-in. hauling cable; *f*, pole carrying 8-in. sheave with 12-in. conical plate beneath. The scrapers drag the material from the pit to the dump, 100–300 ft. horizontally and 20–50 ft. vertically. They are not always provided with teeth, but this is advisable. A rigid bale should never be used, as flat stones catch between it and the body of the scraper.

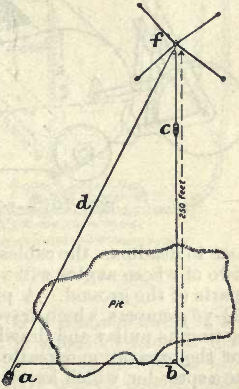


FIG. 100.—STEAM SCRAPER.

An example dealing with washdirt 60 ft. wide and 5 ft. deep, having a capacity of $\frac{1}{3}$ cub. yd., and using 6 h.p., moved 100 cub. yd. per 24 hr.

By replacing the sheave anchor (*b*, Fig. 100), here a rock-filled crib, by a truck on a track, made fast by cable and deadman, greater mobility would be attained. The scraper should deliver into the sluice-box, by ascending an inclined plane. A scraper of large capacity, to handle 700 cub. yd. a day, should be preferably of the bottomless, self-dumping type, of 6 cub. yd. capacity; by it actually $3\frac{1}{2}$ yd. will be delivered each time. The operations will require a 60-h.p. boiler and double-drum hoist, and the services of 7 men on a shift. The hoist can be mounted on skids, so as to be easily moved by a sheave and deadman. The sheave through which the drawback cable runs may be anchored to a weighted car running on 200 ft. of track laid parallel with the cut on the side opposite that occupied by the sluice and hoist; and the drawback cable may pass through two sheaves, one anchored to travelling anchor and

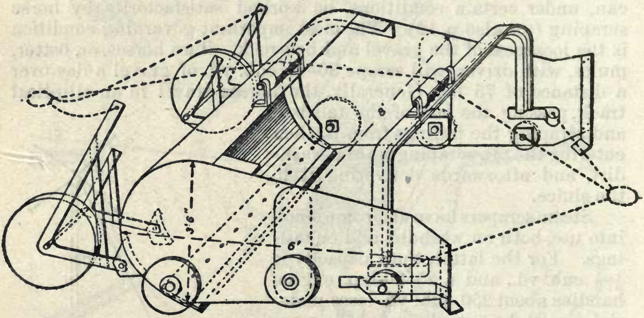


FIG. 101.—BOTTOMLESS STEAM SCRAPER.

one to deadman, the cables thus forming a triangular arrangement, two of whose angles will vary as the car is moved to cover various parts of the ground. A plant working on Klamath river has two $1\frac{1}{4}$-yd. scrapers, which travel back and forth alternately, both cables acting as pulley and drawback. Two sheaves are used on the side of the excavation opposite to the washing plant, and are attached to a spreader, which keeps them a given distance apart; to each end of the spreader is fastened a tackle, which runs back at an angle to deadmen, to which it is securely anchored. The bottomless scraper shown in Fig. 101 has a theoretical capacity of 6 yd., and actually handles a little over half this amount. In seven 10-hr. days, stripping to 4 ft. in depth, it handled 400 cub. yd. per shift, making furrows over 300 ft. long. A 60-h.p. boiler was used, and 1 fireman, 1 winchman, and 2 scraper men were employed. In a haul of 150–200 ft., it would deliver 2 cub. yd. per min.

Steam Shovel.—When a daily output of 1000 cub. yd. or more is feasible, the steam shovel comes into play, especially if the bedrock is sufficiently soft to allow the dipper lip to dig far enough into it to recover all the values; otherwise a gang of men will have to follow the shovel to clean bedrock. A prime essential to success is that the washing plant shall be isolated, the gravel being conveyed from the dipper to the sluice by some form of tramming. If cars are used, they should be 2 cub. yd. capacity, or larger. Under ideal conditions, the dipper dumps into cars run by gravity to the hopper of the washing plant, the full cars pulling the empties back to the pit. Tramming, even when it must be up an incline to a height of 35 ft. above the pit floor, adds

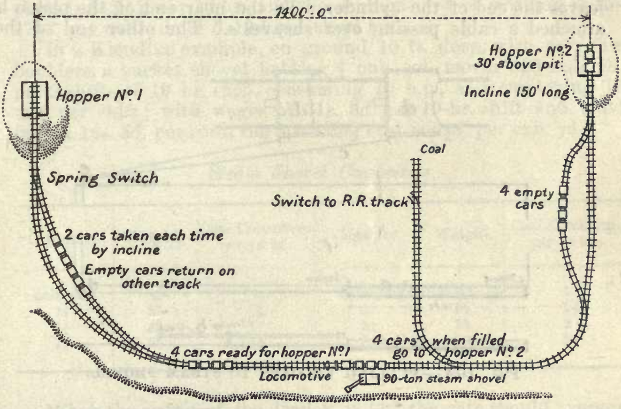


FIG. 102.—STEAM-SHOVEL OPERATIONS.

proportionately little to the expense. It is a common fault in all steam-shovel operations that the shovel is ahead of the car discharge. A partly idle shovel, however, is not so serious as idle men, since the shovel draws no pay. With a 2-yd. shovel, fitted with extra long boom and $1\frac{1}{4}$-yd. dipper, a 25-ft. bank can be dug and caved. Tramming may be more cheaply accomplished, where there are several years' work ahead, by a small locomotive in the pit running to the bottom of the incline or directly to the washing plant. In Fig. 102, the locomotive occupies a position intermediate between the two trains of cars, which deliver to the bottom of two inclines leading to hoppers at each end of the pit. If conditions admit of dividing the water to two washing plants, this system allows rapid delivery of cars from the shovel. In this plant, the locomotive keeps 20 cars going (each of 2 cub. yd.), alternately in trains of 6 and 4, two ways to the ends of the pit, whence they are hauled, two at a time, to the hoppers. When empty, they run down, and are switched automatically to the empty tracks. With a 90-ton shovel, of 5-yard dipper capacity but fitted with 2-yard dipper, the actual yardage of firm shale moved, working 9 hr. a day, is 670, or at the rate of 1488 yd. in 20 hr.; this is less than half what it could do with loose gravel.

A most convenient device for moving up the cars within reach of the dipper is shown in Fig. 103. The long cylinder *a*, made with casting to attach to the shovel *f*, contains a piston of equal length, which is supported on suspended track and wheel *b* as it leaves the end of the cylinder. To the near end of the piston is attached a cable passing over sheaves *e*. The other end of the

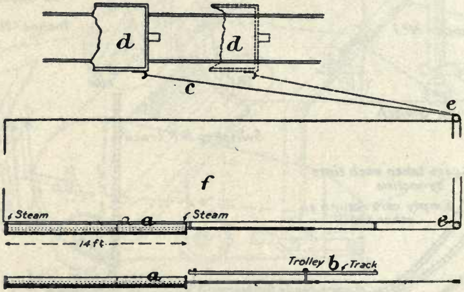


FIG. 103.—PULLING UP CARS TO STEAM SHOVEL.

cable *c* is hooked to the corner of the gravel car *d*. When steam is turned into the near end of the cylinder, the piston travels back toward the forward end of the shovel, and the gravel car is hauled by the cable an equal distance, 5–7 ft., as may be required. Steam is then turned into the cylinder, allowing the piston to return and the cable to free itself; the cable is unhooked and pulled by the car man to the following car, and hooked in readiness to pull it along. The steam and time consumed are so small as to be almost negligible. By passing the cable around the body of the shovel car, over a second sheave, the cars on the opposite side of the shovel can be moved, when the relative position of the shovel is reversed. Dipper chains are generally preferable to cables: a link can be repaired, while wire cable cannot. The moving of the shovel as the digging progresses may be done economically and rapidly by skids of 6-in. timber hung from the jack frames by chains (thus requiring no attention when the shovel is moved up); on these skids, rest 6-in. blocks, 2 × 2 ft., on which the jack shoes bear.

The actual working costs (Illinois), in moving 17,422 cub. yd. shale per month of 26 9-hr. days, digging from 50-ft. bank with a shovel fitted with 2-yd. dipper, delivering to 20 2-yd. cars trammed 2 ways, 1500 and 2000 ft., respectively, to bottom of 2 inclines, hoisted by cable to hoppers at elevation of 20 ft. above the track, and dumped to hoppers, was 2·16*d.* per cub. yd., including 3 t. coal per month at 8*s.* 4*d.*, and wages ranging from 18*l.* per mo. for engineers (3) to 10*d.* per hr. for fireman, trackmen, and hoistmen (7 in all).

At the Minnesota iron-mines, the average work of an ordinary plant is 3458 t. per 10 hr., and the average cost of working is 15–17½*d.* per cub. yd., or 5–10*d.* per short ton.

In a Klondike example, on ground 10 ft. deep, with very few boulders, a bucket shovel holding ¾ cub. yd. moves 800 cub. yd. per 2 shifts of 10 hr. each, consuming 10 h.p., and employing 10 men per shift; with wages at 31*s.* 3*d.* per 10-hr. shift and wood fuel at 16*s.* 8*d.* per cord, the operating cost is 6½*d.* per cub. yd.

Steam Shovel Capacities.

Capacity.	Weight.	Coal Consump. per 10 hr.	Capacity.	Weight.	Coal Consump. per 10 hr.
cub. yd.	t.	t.	cub. yd.	t.	t.
1¼	35	¾	2	65	1¼
1½	45	1	2½	75	2
1¾	55	1¼	3	90	2½

When the swing of the derrick is 90°, the rate should average 3 shovels per min. Using rails in 5-ft. lengths, shovel may advance 5 ft. before changing. Change occupies 3–5 min. Width of cut, 15–40 ft. Height of face 15–30 ft.; below 12 ft., not economical, because of disproportionate time in changing, and inability to *fill* the shovel. Most important to ensure plenty of cars and easy access. On average work, in America, it is estimated that a 1-cub. yd. shovel kept actually shovelling for 50 % of the time, with easy gravel, will deal with about 50 cub. yd. (in place) per hr., falling to 35 cub. yd. with tough or cemented ground; and the corresponding figures for 1½ and 2½ cub. yd. shovels, respectively, will be 80–60 cub. yd., and 140–100 cub. yd. The lesser sized shovels are applicable to small cuts; extra weight is necessary for tough or hard ground.

Benching.—The Mount Lyell copper mine, Tasmania, has a lens-shaped pyritic body, 600 ft. long by 30 ft. wide in extreme parts, resting on schist and overlaid by massive conglomerate. The deposit was first attacked by sinking a shaft in the centre of the ore-body and connecting this by a tunnel through the mountain-side at a depth of about 450 ft. Then the ground was excavated around the top of the shaft, and was worked back toward a bench. As soon as this bench was far enough back, a second was opened, and others followed as facility offered; there are now ten in operation, the topmost about 800 × 600 ft. while the lowest is 100 ft. across at a depth of 266 ft. Ore quarried from the benches is trucked to a shaft and tipped to cars in a tunnel below or on lower benches; waste is taken through a separate tunnel, and run to a dump, costing the same as ore. Any large blocks of ground brought down by the blasts (fired in series), are separately broken up by "pops" to sizes convenient for loading into mine cars by steam travelling cranes or by hand. All drilling is done by machine. The output of mined material is 1000 tons daily.

The Mount Morgan gold mine, Queensland, has abandoned square-set timbering in favour of opencast benching, as worked at Rio Tinto and in the Minnesota iron-mines. Steam shovels are working at a depth of 315 ft., and preparations are being made for carrying the cut down to a depth of 500–600 ft. This will necessitate excavation of waste for a considerable distance from the ore-body, to prevent "creeps." Benches are 50–60 ft. deep. In preliminary operations, as much waste as ore has to be removed; but, when the cut is opened out further, waste will be largely reduced. The ore-body is about 800 ft. long and 500 ft. wide. The surface ore is of chalky consistency and much of the country is quartzite. The lower level sulphide ore is hard and compact.

Drifting.

In working the deep, often sub-basaltic "leads" of auriferous gravel, which attain very large dimensions in Victoria and in California, special means have to be adopted, which in some cases more nearly resemble long-wall mining, and in others are the counterpart of pillar-and-stall methods.

An example of Californian procedure is illustrated in Fig. 104: *a*, bedrock; *b*, rim or rising bank of channel, miscalled "reef" in some countries; *c*, wash-dirt; *d*, main drive or gangway, timbered with posts *e* and caps *f*, and having a car-track *g*; other fittings in the drive are a 10-in. air-pipe *h*, compressed-air pipe *i*, boxed-in steam-pipe *k*, and pump-column *l*, with a drain at *m*, and an electric bell wire; *n* is an ordinary post, with cap *o* and spreader *p*. A depth of about 6½ ft. of gravel is removed for washing, the remainder (above it) being too poor to pay for extraction.

In some instances the maintenance of the principal and permanent drive in the channel itself is so difficult and costly that it is preferable to run it in the bedrock, and to connect it by shoots *r* at intervals with the temporary track *s*, which serves the faces where

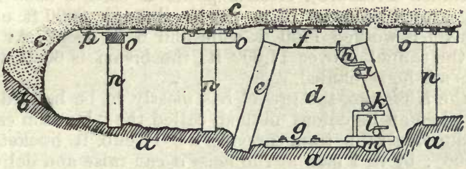


FIG. 104.—DRIFTING: CALIFORNIA.

operations are being conducted. This arrangement is shown in Fig. 105. The shoot *r* is about 3 ft. square inside, lined with boards, and these are covered at bottom and sides with $\frac{1}{16}$-in. sheet iron; it

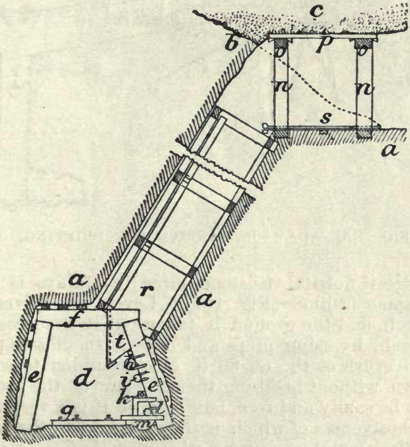


FIG. 105.—DRIFTING; BEDROCK TUNNEL.

is provided with a movable lip *t*, and a door for controlling discharge into cars on the main track *g*.

In California, drifting has been done in some instances at 60–150 ft. above bedrock. The topmost gravel is rarely paying to any method. Where the bedrock itself carries value, it is drifted out at same time; in hydraulicing, the bedrock may have to be blasted before it can be piped. At the largest Californian mine (the Hidden Treasure), electric power is used, a 6500 lb. motor hauling gravel trains at 12 m. per hr. through 8000 ft. of tunnel. The gravel is worked 1400 ft. wide and 6 ft. deep. At the Red Point, the tunnel is over 14,000 ft., the breast is 60 ft. wide, and the cars are horse-hauled.

In the Klondike, the gravel has mostly to be hoisted through shafts, when an ingenious method, called the "Dawson carrier"—a self-dumping cable tram carrying a 9–11 cub. ft. bucket—is used (Fig. 106): by it, 1 man at the hoist *a* can raise and deliver to the sluice *b* as much dirt as 8–10 miners can deliver at shaft bottom. The post carrying the cable is guyed (*c*) where necessary.

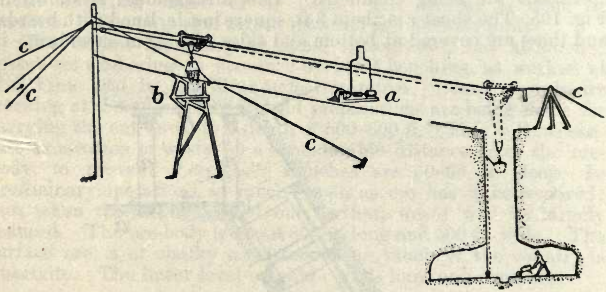


FIG. 106.—DAWSON CARRIER FOR DRIFTING.

The largest detrital tinstone mine in Malaya is worked in much the same fashion (Fig. 107). Levels are driven at every 20 ft. in depth, and the ground is taken out in 3 successive slices of 6–7 ft. each, by using props and caps without sole-pieces, and allowing the crush of the 60–80 ft. of overburden to squeeze the timbers down without breaking them. Some of the ground has a tendency to be sandy and free, but most of it is a very tough blue clay, the cohesiveness of which is the salvation of the mine. But for the regularity and evenness of the crush, both in direction and in speed, such a system would be impossible. As it is, the cost is enormous. "Stoping" alone, on contract, is paid for at the rate of 2*s.* 9*d.* per cub. yd., and stoping, in this instance, means chopping the clay out with a hoe, and basketing it to the truck which stands 20–30 yd. back from the face. The dirt is raised in bottom-discharge wooden skips, which empty into small trucks (16 cub. ft.), for rope haulage to the washing plant.

The Victorian deep leads are remarkable for the immense volume of water which they contain, and for the huge pumping plants required. From the Loddon Valley mine, 6000 million gal. of water

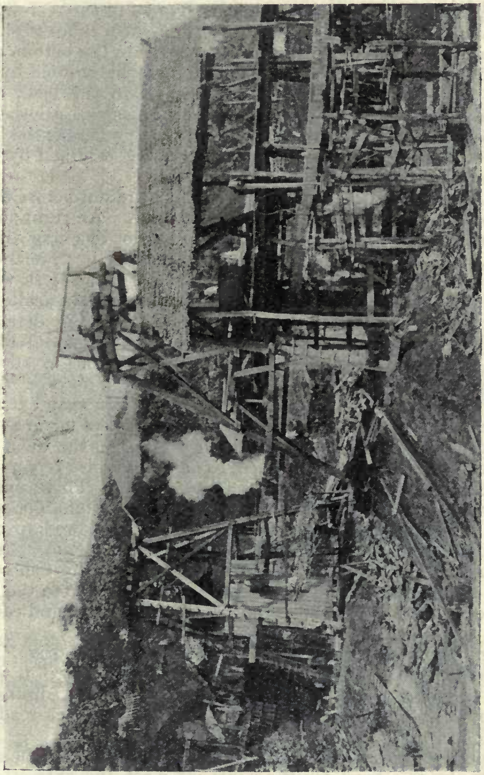


FIG. 107.—DRIFTING IN A MALAY TIN MINE.

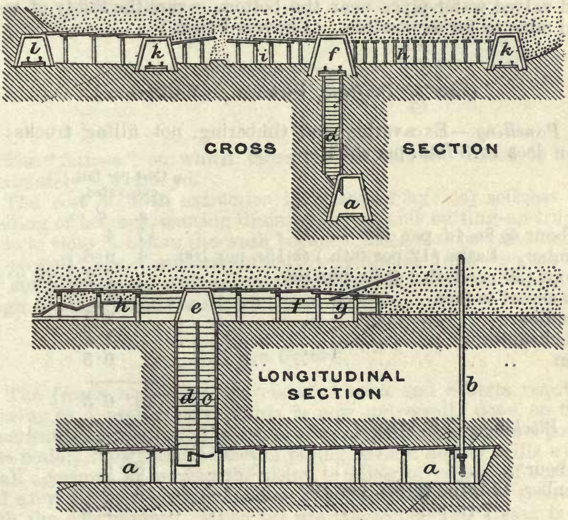
have been raised, and the present pumping capacity is 10½ million gal. daily. The initial water pressure is said to be 200 lb. per sq. in. on the bedrock of the wash, and this has to be reduced to 6 lb. per sq. in. to ensure safety in working. Draining and removal of washdirt are accomplished by two series of levels—"reef-drives" (from the main shaft) in the solid rock well below the washdirt, and "wash-drives" partially in the washdirt and partially in the bedrock. The water saturating the drift sands overlying the washdirt proper is lowered to within about 15 ft. of the bedrock by tapping. Boreholes are put up from the roof of the reef-drive into the water-bearing stratum by jacking 3-in. steel tubes through the soft slates, assisted by boring tools; a valve in the bottom of each tube controls the flow. The wash can then be entered by drives, which are extended across and up and down the lead, at wide intervals, as fast as the capacity of the pumps will allow. When the wash has been partially drained by these wash-drives, it is cut up by other drives, and, after further draining, is mined by "panelling" or "blocking."

At intervals of 100–200 ft., wash-drives are put across the lead, and connected by shoots with the reef-drive. At right-angles to these, and at 35–40-ft. intervals, truck roads or blocking drives are driven, and completely drain the wash; this is "panelled" on either side of each truck road by taking out strips 4½ ft. wide and 16 ft. long. The back is held up by laths placed parallel to the truck road. Each lath is supported by two legs, 2 in. diam. × 2½–3½ ft. long, and forms a separate panel independent of all other laths. In "blocking," strips of wash, 7 ft. wide, are removed, and the roof is supported by laths 4½ ft. long, resting on caps 7 ft. long, supported by legs. Blocking is used where the wash extracted is greater than 3 ft. thick. An endeavour is always made to get the "blocking" or "panelling" parties operating on the wash as soon as it has been thoroughly drained, so as to minimise repairs, which are great, especially if the ground is swelling. These operations are carried on up and down the lead, from a point opposite the shaft; after the wash is extracted, the roof is allowed to collapse immediately behind the blocking or panelling parties. From the working faces, the wash is hand-trucked to the nearest shoot ahead of the working faces, and conveyed by the reef-drive to the shaft.

The water is held sponge-like by the drift sand, and each tube drains only a limited basin-shaped area.

Figs. 108, 109, illustrate the operations: *a*, reef-drive, leading from main shaft; *b*, draining tube; *c*, ladder-way; *d*, washdirt shoot; *e*, cross wash-drive; *f*, leading wash-drive, draining the wash by tubes *g*; *h*, panelling; *i*, blocking; *k*, truck-road; *l*, air and timber way.

Panelling consists in simply taking the ground out in strips at right angles to a level for the width of a set, the extraction commencing away from the shaft and working toward it. The levels from which the panels are worked are generally driven in such an erratic manner that much ground is lost for want of system, and, as blocks of supposed poor ground are left standing, as well as blocks that cannot be safely attacked on account of the bad method of laying out, the roof does not cave uniformly. When panelling, the men work on their sides or squatting down, and scrape up about 6 in. of the floor. The roadway is a little below the panels, partly with the object of giving more head room, and partly to assist the men when loading the washdirt into trucks, it being shovelled from the face, or brought in wheel-barrows. Blocking costs 1*s.* per fathom more than panelling, as more ground is taken out, but the



FIGS. 108, 109.—DRIFTING, VICTORIA.

men have more room to work in. The pillar left after the ground has been opened up is divided longitudinally by an imaginary line; the mates of each blocking party of two pairs work up to this line from the blocking drives on either side of it, commencing at the end nearest the worked-out ground. All the blocks in a section are worked simultaneously, and the working face is kept even right across the lead. No dirt is stowed in the old workings, the ground overhead being allowed to fall in. At Chiltern, the miners extract about 2 ft. of wash, and 1–1½ ft. of "reef bottom." If the distance is not too far to truck, one trucker will do the work for two pairs of men. Miners support the roof with light timber as they proceed, and a temporary rail-track is laid, which is always kept at a convenient distance from the face. At Chiltern, electric locomotives are used underground. Each bore is provided with a quick-closing slide-valve, by which the flow can be cut off instantaneously. The bore-hole tubes are wedged with wood. If the rush of water and sand is so great as to scour the tubes, small tubes are driven inside the larger. Sand-locks are constructed between the bore-holes and the main level. And water-lock doors of great strength, and provided with air-escape pipes near the top and valved water-pipes near the bottom, ensure the safety of men and mine as far as possible. (Danvers Power.)

Costs of Drifting, Victoria. (Wilkinson.)

Panelling.—Excavating and timbering, not filling trucks; 1 man does 1·13 fath. per shift :—

	Cost per fath. excavated. *s.*	*d.*	
Labour @ 8*s.* 4*d.* per day	7	4	
Timber. Laths (12 per fath.) @ 15*s.* per 100	1	9·6	} 2*s.* 9·8*d.*
Props (20 per fath.) @ 3*s.* per 100		7·2	
Transport to face		5	
Candles		2·7	
Blacksmithing		1·5	
Iron		0·5	
	10	6·5	

Blocking.—7 ft. strips.

	s.	*d.*	
Labour	8	6	
Timber, 15 laths @ 15*s.* per 100	2	3	} 3*s.* 4·9*d.*
1 7-ft. pole		2·5	
3 props @ 15*s.* per 100		5·4	
Preparation		1	
Transport		5	
Candles		2·7	
Blacksmithing		1·5	
Iron		0·5	
	12	3·6	

These figures refer to wash extraction under best conditions, and do not include filling and trucking to shoots, which costs 3–5*s.* per fath. Contract costs, including trucking (excluding timber), vary from 13*s.* 6*d.* to 17*s.* 3*d.*, with ordinary ground and length of transport.

Total costs for well-opened-up mine, taking out full width of wash developed:—

	s.	d.
Development	8	
Panelling	15	
Timber	5	
Trucking	2	
Winding	1	6
Puddling	2	6
	34	
Managing, pumping, etc... ..	6	
	40	

The "fathom" on which these calculations are based is approximately = 4 cub. yd.

The cost of wash extraction is increased by: (*a*) softness or swelling of bedrock, causing draining-drives and cutting-up truck roads to close in before the wash between them can be extracted; (*b*) irregularities of surface caused by "potholes" and "bumps"; (*c*) actual bedrock surface which has to be removed being too tough to allow of easy work with shovel.

Frozen Ground.

The frozen auriferous gravels of Alaska and Siberia require thawing as a preliminary. This is now universally done, on the American Continent, by the steam thawer, which comprises a portable boiler, a few hundred feet of piping, several hollow drills with small apertures at or near the point to allow of escape of steam, and a few feet of rubber steam hose to connect the drills or "points" with the iron piping. When all are connected, and steam is at about 120 lb. in the boiler, the point is held against the face of the gravel, and the steam is let into it by a valve. The gravel in front quickly thaws, and the point can often be pushed in its own length, usually 4 ft., in a few minutes. When all the points are in, the steam is allowed to fall to 40 or 50 lb., and is kept at that for 6–8 hours, in which time each point will have thawed about 2 cub. yd. of gravel. The points are then moved to another part of the drift, or to another drift; and when the gravel cools, it is dug out and taken to the surface to be washed in the sluice-boxes.

As the steam thawer does not vitiate the air in the drift, it can be used in summer as well as in winter, and then the gold-bearing gravel can be mined, and immediately afterwards washed in the sluice-box, at much less cost than if it had to be handled twice.

In this work, no timbering of any consequence need be used, as the gravel and overlying peat are permanently frozen sufficiently hard to support the roof, if the drift is not made too large.

Where the depth of the gravel is not more than 10 or 12 ft., or where the gold is scattered plentifully throughout, it is often worked by removing all the peat from the surface; the heat of the sun and of the warm summer air then thaws the gravel quite quickly, so that it is possible each day, and day after day, to shovel off a little and pitch it into the sluice-box. In this way, ground can be very thoroughly worked over, as the work is done in summer and is all in sight from the surface. (Tyrrell, Tr. I. M. & M.)

In the Nome district, similar measures are in vogue. A small boiler (6–8 h.p.) supplies steam to sets (6–9) of thawing-points, and each "thaw" occupies about 1 hr. In shaft-sinking, the points are placed 18–36 in. apart, and in ordinary gravel, after 40 minutes' steaming at 40 lb., the ground is softened for 6 in. ahead. In breasting-out, 3 men can thaw and remove about 3 ft. in a face 6 ft. wide: assuming a pay-streak 5 ft. thick, this means 90 cub. ft., or say 3 cub. yd.; the cost, including labour, supervision and fuel, is about 6*l.* per diem, or 2*l.* per cub. yd., which may be reduced 15 % by running 4 gangs at a time. For merely testing ground, 12-in. holes are sunk by the aid of well-boring rigs and oil-engines, 25–50 ft. a day being accomplished. (A. L. Pearse, Tr. I. M. & M.)

Points 5 ft. long are used. "Batteries" or "crossheads" of 4 are supplied with steam from a crosshead of $\frac{3}{4}$-in. iron gas-pipe, fitted with elbows and short pieces of $\frac{1}{2}$-in. pipe leading to the steam hose to which the points are attached. The valves are generally set in these short pieces. The points are driven in with a mallet by the man on night shift, and are left in the bank 10–14 hr., the only labour being the point-man and the fireman. Each point thaws on an average 6 ft. into the bank, 18 in. on each side, and 4 ft. high.

It is good practice to start the points with hot water. They must be driven carefully and slowly, and, for 10 points distributed along a face, the average time needed is 1–3 hr. The amount of steam required for each point is 1–2 b.h.p. The amount of gravel which a point will thaw ranges from $3\frac{1}{3}$ to $4\frac{1}{2}$ cub. yd. When the gravel is very argillaceous, steam thawing sometimes results in a baking or cementing action, which increases the difficulty of mining and washing it. (Purington.)

Thawing with hot water by means of a force pump has been successful at a claim where it was desired to extract a 3-ft. pay streak capped by 27 ft. of barren gravel at a depth of 50 ft. A small force pump (ram pattern, outside-packed valves) placed in the main runway, drew water from a 6-ft. sump to which the workings drained. It had 4-in. intake, 3-in. discharge choked to $2\frac{1}{2}$ in., and the water was pumped to the face through cotton hose and discharged by a 1-in. brass nozzle at 40 lb.; 6000 gal. water were used over and over, and, by discharging the exhaust into the suction, the water was kept at 150° F. In a 10-hr. shift, the pump, using 30 h.p., thawed and broke down ready for the shovellers 175 cub. yd. This is vastly superior to the duty of the $1\frac{1}{2}$-h.p. steam point, even allowing 4 cub. yd. to the point, as the 30 h.p. would supply only 20 points, and the max. duty would be 80 cub. yd. But, for the hot water method, there must be no silt in the gravel, otherwise the water becomes thick, and cannot be settled in the sump. In this method of thawing by hot water piped against the bank, the gravel thawed can be selected in the face, and none other need be taken down, the rest remaining solidly frozen. It also avoids the unpleasantness of the steam method, which fills the workings with steam, and creates dampness; the walls and roof of the drifts are dry, and ditches cut in the bedrock on grade to the pump sump drain the floors. (Purington.)

In the frozen placers of Siberia, the crude and costly system of thawing by wood fires still obtains.

Hydraulicing.

The hydraulic method of working mineral deposits is most economical of power, capacity, and labour. But it is exceedingly wasteful of water, and the topographic conditions under which it may be successfully conducted are not common. Besides abundant water, it needs adequate slope of the bedrock for moving material, the lightest suitable grade being 220 ft. to the mile.

One or more jets of water under pressure are thrown from a pipe or pipes with great velocity against the face of a bank, and the gravel is loosened by the stream so that it falls or "caves," being struck with sufficient force to be disintegrated, so that it will be carried by the current into the sluice. For moving and washing the gravel after it leaves the bank, additional water is frequently used under slight head, and is known as "bank head" or "by-wash" water. Various devices for more easily disintegrating the bank, and for disposal of the boulders and tailings, are employed.

The water is conveyed to the points of application by ditches, flumes and pipes, as in the case of water to be used in generating power (see pp. 22–41).

Ditches.—When "scraping" can be adopted for cutting a ditch, a strong 2-horse plough first runs a single furrow, following, as closely as possible, from one survey peg to the next, the natural contour of the country, thus establishing the ditch line. This is next continued to a width sufficient (allowing plenty of slope to the inner bank) for the required depth of ditch. The "grader" is then used to remove what has been ploughed to the outer bank of the ditch. This done, the ditch looks like a wagon road. Then the plough is used again for a single furrow, following the first, and the loose material is "scraped" from the ditch to the outer bank, building it up. This is repeated until the ditch is almost completed. It remains to level up the bottom, and to slope the banks to required dimensions. This is done by hand, with pick and shovel, but plough and scraper should do almost all the work. Only a small head of water should be allowed to flow through the ditch for a few days; when it has become well soaked, the head may be increased a little daily, until full capacity is reached. (See also pp. 24–27.)

Flumes.—(See also pp. 28–30.) The first cost of a flume is often less than that of a ditch, and its repairs cost very much less under ordinary conditions.

Generally it is considered cheaper to commence at the head of a flume and build down stream, floating the lumber in the flume. But during excessive heat the lumber will "check," and anyway it will require more handling, besides which the workmen must walk in the water or the flume be completed, even to the walking-board, as it advances. By using a low truck, made with axles wide enough to run on the two outside stringers, lumber can be run out and the bents be kept a long way ahead. Immediately the floor is laid it may be used as a wagon road to haul lumber with a single horse in shafts, the wheels to the trucks being 10 in. wide, cut out of a tree, and with two 4 × $1\frac{1}{2}$-in. tires. (Ralston.)

Where the price of sawn lumber is prohibitive, small flumes can be built of comparatively rough wood, provided a good caulking material, such as some kinds of moss and bark, is at hand.

Flumes are usually built rectangular in section, but when the ground is unsuitable for excavating the necessary width for a large square flume, a section such as Fig. 110 may be adopted, and will actually afford about 1% greater flow for the same sectional area, while loss by evaporation will be less as the stream diminishes. Details are: stringers *a*, 12′ × 9″ × 7″; posts *b*, 6″ × 4″, on $1\frac{1}{2}$ in. block; sills *c*, 6′ 6″ × 6″ × 4″; posts *d*, 4′ 6″ × 10″ × 4″; cross ties *e*, 8′ 6″ × 6″ × 4″, with posts gained in 1 in.; lining boards *f*, 12′ × 18″ × $1\frac{1}{2}$″; lagging boards *g*, 12′ × 4″ × $\frac{1}{2}$″; floor boards *h*, 12′ × 24″ × $1\frac{1}{2}$″; angle struts *i*, 2′ $9\frac{1}{2}$″ × 4″ × 4″; gang planks *k*, 12′ × 10″ × $1\frac{1}{2}$″. (W. H. Radford.)

Piping.—(See also pp. 30–41.) The penstock, pressure-box, or sand-tank, as it is variously called (see also p. 22), should be of ample size, and the water should stand not less than 4 ft. above the pipe inlet, to prevent admission of air. If made in two partitions, it becomes more effective as a sand-box. An ingenious addition for disposing of floating impurities (p. 22) is a paddle wheel, driven by the current, actuating (by a chain and sprocket wheels) two pulleys which revolve very slowly and carry an endless belt of $1\frac{1}{2}$-in. poultry netting. This belt, of the same width as the flume, emerges from the water at an angle of 35°, and dumps its load on a steeply sloping apron, having a diagonal guide, which diverts it over the down-hill edge of the flume. The blocks to which are fastened the boxes of the forward pulley are adjustable to admit of tightening the belt.

Monitors.—Of the two types in general use, single- and double-jointed, the latter is superior in efficiency of the stream, but the former is easier of manipulation, as it can be lubricated without turning the water off. The double-jointed machine is safer under

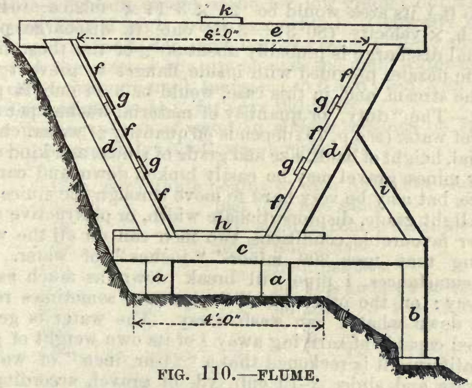


FIG. 110.—FLUME.

high heads. It is made with and without a king-bolt; the latter has a clear water-way, and is therefore more efficient; but it has the drawback of being ball-bearing, which, on account of damage to the balls, is often disadvantageous. The king-bolt type is undoubtedly best for all practical conditions. Eight sizes are in use, with nozzles of $1\frac{1}{2}$–10 in. diam.; for all except No. 1, a deflector is indispensable. The method of setting usually recommended consists in driving bars (iron or steel) through holes bored for the purpose in the ends of the bed-piece, into the ground. But stakes driven against the bed-piece hold better, and can be more easily removed.

The most modern deflector is electrically operated, and so arranged that the stream may be accurately directed to any desired point without the pipe-man being anywhere near the giant itself. Where banks are 200–300 ft. high, the giants have to be placed at some distance, and the full force of the water cannot be utilised, as it "feathers" or breaks before striking the gravel. With this device, it is possible to set the giant as close as desired, and deflect the stream in any direction, the operator standing out of danger.

To estimate the quantity of water discharged from a nozzle, extract the square root of the head, and multiply this root by 8·03; the product will be the velocity (ft. per sec.) with which the water escapes. Area of mouth-piece × this velocity = discharge (cub. ft. per sec.). *Ex.*—What quantity of water will be discharged from a pipe, under a head of 100 ft., through a 3-in. nozzle? *Ans.*—The square root of the head (100) is 10 × 8·03 = 80·3 ft. as the velocity per second. The diameter of nozzle being 3 in. (·25 ft.), its area would be ·25 × 3·14 × ·0625 = ·04906 sq. ft., which, × velocity (80·3) = 3·93 cub. ft. discharge per sec. The actual discharge is probably about 80 % of the theoretical, in well-made nozzles provided with inside flanges to prevent revolution of the stream, and, in this case, would be 3·14 cub. ft. per sec.

Duty.—The "duty," or quantity of material washed per miners' "inch" of water (see p. 19) depends on quantity of water, character of material, height of bank, size and grade of sluice, and kind of riffle. In many mines, gravel may be easily broken down and carried to the sluice, but may be very hard to move through the sluice, on account of light grade, disproportionate width, or obstructive riffles.

Under favourable conditions, two men can do all the work in a washing that uses 300 miners' "inches" of water. Under such circumstances, 1 pipe will break down as much as 3 can wash away; on the other hand, 3 pipes are sometimes required to break down what 1 can wash away. The water is generally considered capable of carrying away $\frac{1}{5}$ of its own weight of gravel.

In California it is reckoned that a "24-hr. inch" of water will disintegrate and sluice 2–10 cub. yd. of gravel, according to its character; 7 cub. yd. is a common figure, while, on occasion, 1 cub. yd. has required 534 cub. ft. of water, or 20 times its bulk.

In Klondike, 5 cub. yd. is usual.

Australian and New Zealand figures, worked out at "cub. yd. gravel washed per cub. ft. water per sec.," range from 160 to 178, according to grade; a safe average is 165 cub. yd. on a 5 % (7·2 in. per 12 ft.) grade. (H. L. Lewis.)

Saving Values.—The arrest and collection of the valuable contents of the gravel, whatever their nature—gold, tinstone, gems, etc.—are effected in sluices, which sometimes are merely long ditches, known as ground sluices, but more often are wooden structures or box sluices.

Sluices must vary in length according to the nature of the dirt being washed, to be determined in each instance by actual experiment, an increase being necessary if the tailings show a loss. The sluice is sometimes run on a wide curve, the outside edge being raised $\frac{1}{2}$–1 in. to equalise the wear and tear, because in a straight sluice the velocity of the current would be likely to carry away much of the dirt without disintegrating it. The dimensions of the sluice are determined by the quantity of material to be treated, which is governed by the water supply. One 6 ft. wide by 3 ft. deep, with a 4–5 % grade, will take about 3500 miners' "inches"; one 4 ft. wide by 2½ ft. deep, with 2½ % grade, 1200–1500; or with 4 % grade, 2000 miners' "inches." The water must be in sufficient depth to cover the largest boulder likely to be encountered, so that the body required will vary with the coarseness or fineness of the dirt. With too much water, the riffles are likely to pack, and the yield will be less, increasing with low grade and small body of water. If water is plentiful and cheap, it will to a certain extent atone for low grade; but with water scarce and dear, a high grade is essential. Generally speaking, ordinary gravel needs 4 %, and coarse gravel 6–7 %: the heavier the gravel, the steeper the grade, and the more water necessary; 4 % is very commonly used, increased to 6 and even 8 for clay, and reduced sometimes to 1½ % for very light dirt.

The moving power of water in sluices depends on speed and volume, approximately as follows—a speed of

16 ft. per minute	begins to wear away fine clay.				
30 „ „	just lifts fine sand.				
40 „ „	lifts sand as coarse as linseed.				
60 „ „	moves fine gravel.				
120 „ „	„	inch pebbles.			
200 „ „	„	pebbles as large as eggs.			
320 „ „	„	boulders	3–4 in. thick.		
400 „ „	„	„	6–8	„	
600 „ „	„	„	12–18	„	

Hence the following rule for establishment of grades in sluices when the velocity needed is decided upon (Van Wagenen): Multiply the velocity (ft. per sec.) by itself, and the product by the wet perimeter (ft.). [See p. 23.] Divide this result by twice the area (sq. ft.). The result is the total fall (ft. per mile). *Ex.*—What grade must be given to a sluice 12 in. broad and 6 in. deep, that it may carry a velocity of 320 ft. per min. (5·3 per sec.)? *Ans.*—The velocity (5·3) × itself, and the product × the wet perimeter (24 in. = 2 ft.) = 56·18. This ÷ area (72 sq. in. = ·5 ft.), and doubled = 56·18, which is the fall in ft. per mile. To reduce grades expressed in ft. per mile to in. per box of 12 ft., multiply by ·027. Thus, a grade of 56·18 ft. per mile = 1·5 (1½) in. per box. To reduce to in. per rod (16 ft.), multiply by ·036.

The maximum quantity of water which may be advantageously used in a sluice of correct dimensions, when the ground is ordinarily full of boulders, is set down at 1000 miners' inches. This corresponds to a discharge of 95,000 cub. ft. per hr., which, with gravel and boulders, would represent about double that amount of moving substance in the sluice. When more than this is used, the current will be so strong that men cannot work to any advantage in the head-box. The bottom should be 1¾–2¼ times the height of the side, or, taking the side at 30 in., the bottom should be $52\frac{1}{2}$–$67\frac{1}{2}$ in. wide. If, however, the ground is free from large boulders, it is merely necessary to ascertain the dimensions best adapted to carry the greatest economical quantity of water (1000 in.) or 27·1 cub. ft. per sec. Double this discharge to make room for the gravel. The flume must then discharge 54·2 cub. ft. of material per second.

Screening.—One of the first steps is the mechanical removal, as soon as they are quite cleaned from adherent clay, etc., of all boulders, inasmuch as they demand either heavy grade or much water for their transport, and they unduly wear out the sluice. The point at which they are removed will in part depend upon dumping space. The means of removal may be grizzlies (sloping steel-rail gratings), trommels, or shaking screens; the two latter are very useful in cleansing the boulders, and incidentally breaking up coherent masses of "cemented" or clayey gravel, and are adoptions from dredging practice. Sometimes everything over 1 in. diam. is thus early disposed of.

Puddling.—Unbroken clay lumps are fatal to the saving of values, whether gold, tinstone, or gems, and their disintegration must be accomplished, either by weathering (as with diamondiferous

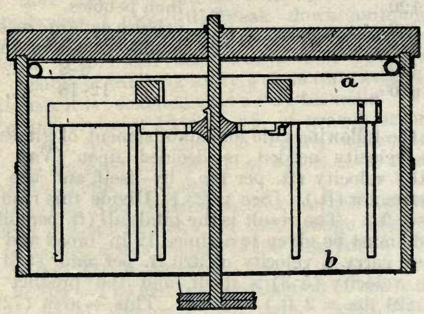


FIG. 111.—PUDDLER.

ground at Kimberley), or by puddling in some shape. The "mud-box" of the Klondike is a very rudimentary puddler, in which all the work is done manually by a fork-man, who breaks up the clay balls against "pole-riffles," or saplings studded with small squares of $2 \times \frac{1}{16}$ in. sheet iron driven in cornerwise. An elementary type of puddler, which can be built of materials at hand almost everywhere, is shown in Fig. 111. It is founded on the idea of the Siberian pan, and has a capacity of 100–200 cub. yd. in 10 hr. Assuming that steam power is already at hand, it requires no outside material beyond the iron shoes and simple castings, and the punched steel plate which forms the floor of the pan. Its operation requires 10 h.p. The device for automatically clearing the bottom of the pan of large stones is not necessary where hand labour is cheap enough to dispense with it, the large stones being periodically removed by lifting gates in the periphery of the pan. The amount of water used does not exceed ordinarily 125 miners' "inches." A 4-armed casting, keyed to the shaft and bolted to the horizontal timbers, is advisable. As used in Siberia, the central shaft is frequently a wooden beam, and heavy stones dragged with chains, as in the arrastra, may supplant the iron shoes. Water is supplied by a perforated pipe *a*, and the floor *b* is

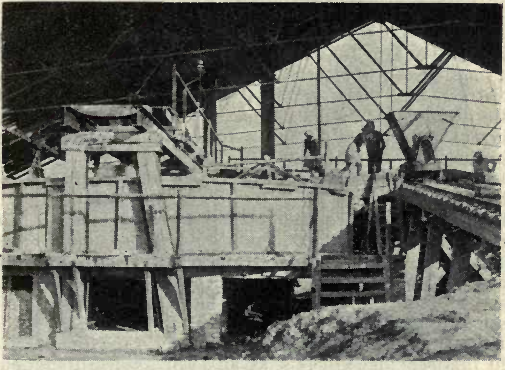


FIG. 112.—MALAYAN TINSTONE PUDDLERS.

$\frac{1}{4}$-in. steel plate inclined toward an outlet shoot at a grade of 3 in. to 8 ft., and punched with $\frac{1}{2}$–1-in. holes.

On the Victorian deep leads, the puddlers are usually shallow iron pans, 20 ft. diam., provided with revolving stone drags, overhead driven.

An example of the plant used in Malayan tin mines is shown in Fig. 112. It is usually 25 ft. diam., with 4 arms carrying a drag or harrow, steam driven, costing about 160*l*. complete, and having a capacity of 125 cub. yd. per 24 hr. Sometimes the harrow resembles a gate without a bottom bar. The top bar, of wood, is freely suspended by hooks and rings from the puddler arm, and, at equal intervals, 7 tines of $1\frac{1}{4}$-in. square iron are fastened by clamps. The tines also pass through a second bar about midway in their length, which keeps them in place. As the tines become worn at the lower end, they can be driven down; and when too short for further lowering, they are taken out and welded with other small ends, and again utilised. It is claimed that the action of the tines is to run over the gravel to a great extent, but to churn up the lighter and more clayey portions, whereas the usual harrow form penetrates the already washed gravel, and does less effective work while consuming much more power. Certainly the power required is very small indeed, less than 1 h.p. per puddler.

Riffles.—Usually some form of obstruction to the flow of the water and associated gravel and sand in the sluice is provided, with the object of assisting the arrest of the valuable contents. An exception is the Chinese *lanchut* or box sluice by which more than half the world's tinstone supply is collected in Malaya. This (Fig. 113) is simply a wooden box, 20–30 ft. long, set at a slope of about 1 in 10 to 1 in 16, with a perfectly smooth bottom, furnished with a small stream of clean water, and operated by coolies standing in the stream and wielding hoes of various patterns, by which they rake the down-flowing stream of mixed materials constantly against the current, an essential feature of the "manipulation" being their extraordinary cleverness in using their feet as aids to the hoe. Needless to say the losses of fine tinstone in this plant are never less than 4 %.

Elsewhere, as in Tasmania and Australia, the tin-sluice is several hundred feet long, and is furnished with cross-poles (saplings) at intervals of time and space, the sluice being allowed to almost fill up during a 3–4 weeks' run between clean-ups. The boulders contained in the wash are considered an important aid in themselves to the action of the pole riffles.

At the Bruseh tin-mine, Malaya, the sluice-boxes are 4 ft. wide and 20 in. deep, and are laid at an even grade throughout of 1 in 24. The head of each sluice is simply a conduit, continually advanced as the face recedes, and paved with 4-in. blocks of soft wood, set on end. The effective portion of the sluice-box is paved with riffles consisting of 16-lb. rails set head downwards at about 1½ in. apart and crosswise in the box. The rails are held in position in cast-iron holders 4–5 ft. long, having suitable recesses at the requisite intervals, and nailed to the sides of the box. The intention was to place the riffles only 1 in. apart, and this would probably have been preferable, but the available supply of rails did not suffice for the total length of sluices at that rate. Each sluice demands about 750 cub. ft. of water per minute. There are three: two, 300 ft. long each; the third, 420 ft. The lengths are dictated by the situation and character of the ground. A couple or so of Malays armed with small poll-picks are constantly employed in each, gently raking between the rails to dislodge such heavy waste matters as may have gathered. The clean-up takes place usually every 4 or 5 days, according to the amount of tin. The guiding rule is that work shall stop the moment any trace of tin can be found escaping. Such escape is observed in a very simple manner. In the last foot of the bottom of each sluice are bored 4 small holes about equidistant, across its width, and below each of these is set as a trap an old kerosene tin. A small stream of the heaviest

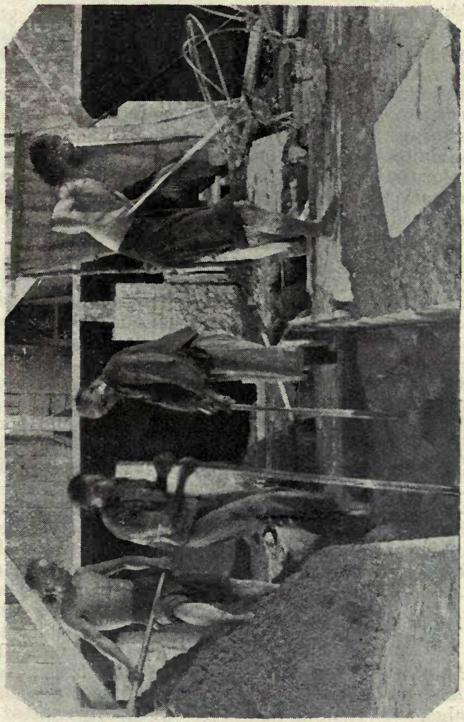


FIG. 113.—CHINESE BOX-SLUICE, MALAYA.

particles passing through the sluice thus falls into the trap, with water, and from that trap there is a constant overflow, so that very considerable concentration takes place. Samples from each trap are assayed at oft-repeated intervals, and, when they reveal any value in this concentrate, sluicing operations are diverted from that box, and its clean-up is proceeded with. The riffles are lifted individually by a pinch-bar and thrown out, and sufficient water is admitted to wash the arrested tinstone down for collection, a lead-loaded wooden slab, 4 in. square and about 4 ft. long, serving as a baffle-board in directing the current. Cleaning up proceeds as fast as a man can pick up and replace the riffles. About 75 % of the total catch is in the topmost third of the sluice. The clean-up occupies about $3\frac{1}{2}$ hr.

In gold-mining, some sort of riffle is always used. In the early days, paving with stones or wooden blocks was general, the interstices serving to catch the gold; but these entail enormous labour for setting and unsetting at each clean-up, and are in less vogue now, though useful in out-of-the-way places and where native labour is cheap. Wood blocks are spaced 1–3 in. apart near the top of the sluice, decreasing towards the dump.

If mercury be used for catching fine gold, it should be sparingly added in small and frequent (twice daily) driblets, well distributed, but not strained through canvas or buckskin and thereby floured. Incautiously employed, especially in a leaky sluice, it will easily cause great loss of gold.

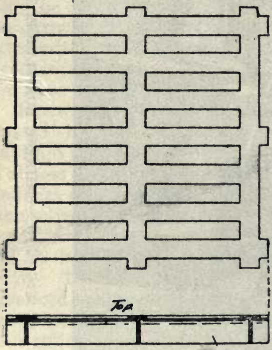


FIG. 114.—GRATE RIFFLE.

The convenience, endurance, and efficiency of the iron riffle have brought it into general favour. The tramway rail lends itself admirably to arrangement in sets giving minimum trouble in laying down and taking up. Another useful form is a sort of angle-iron grating in which both longitudinal and transverse bars are T-shaped (Fig. 114).

Sometimes, where much heavy iron sand accompanies the gold, a useful preliminary to the sluice-box is a sort of automatic agitator, consisting of a series of baffles.

For very fine gold, and for use in "under-currents"—all coarse material having already been eliminated by grizzlies,—coco-nut matting and canvas, sometimes covered by "expanded metal," and even amalgamated shaking tables, are occasionally resorted to; but to obtain efficiency with any of these delicate appliances, the wash-dirt must contain nothing larger than coarse sand.

¶ *Tailings.*—The disposal of tailings is a matter of primary import in hydraulicing, where such enormous quantities of gravel are dealt with. The two chief ways of disposing of them are by piling them in heaps and stacking them in hollows.

Elevators.—By the hydraulic elevator, which operates on the principle of an injector, water under pressure, discharged through a nozzle set within a steel jacket, creates a vacuum, and causes water and accompanying solid material, fed to its intake, to rise to a height corresponding to approximately 10 % of the head under which the water is applied. An essential is the constant admission of air with the tailings; this, being compressed, aids in the elevating process, and those elevators are considered best which admit air most freely. Though simple in operation, it consists of many large parts, some very heavy, and a number of rods, bolts, bands, and staves, costing about 250*l.* and weighing a ton. The pipe from the supply to feed the nozzle of the elevator is connected by a ball joint, allowing the pipe to enter the pit from several directions. Jacketing the nozzle is good practice, as it prevents "smothering" with gravel; and the various parts (lining, backing, first section above throat, top section, etc.) are best made separate, and merely held together by a ring of wooden staves, obviating heavy castings. The upcast pipe is made of $\frac{1}{8}$-in. lap-welded steel, and set at an angle of 60°. The hardest wear is in the throat and on the hood which receives the impact of the gravel in the head-box of the tail-sluice above; these, though manganese steel castings weighing 400–600 lb. each, may wear out in 6 months or less. Wherever possible, wear should be borne by renewable steel liners. There are three well-known makes—Hendy's, Evans's, and Campbell's. For best results, the maximum of gravel should be lifted for the minimum of associated water. It is well known that these machines will lift all the gravel that can be got to the throat. The 3 standard sizes are—10 in. throat, $3\frac{1}{2}$–$4\frac{1}{2}$ in. nozzle, using 300–500 "inches" of water; 12 in. throat, $4\frac{1}{2}$–5 in. nozzle, 500–750 "inches"; 15 in. throat, 5–$5\frac{3}{4}$ in. nozzle, 750–1000 "inches."

Examples of duties from actual experience are: (*a*) 600 cub. ft. of water, at 225 ft. head, lifted sand and gravel to a height of 52 ft. at the rate of 2400 tons in 24 hr.; elevator used 2812·5 gal., and raised its own water, the water coming from the monitor at the rate of 1687·5 gal. per min., together with material washed out; it lifted stones which passed through a 9 in. sieve.

(*b*) 400 "inches" of water, under 300 ft. head, lifted gravel 60 ft. at 72 t. per hr.; largest stones, 6 in. diam.

Dams.—Simplest of all is the brush dam, made by packing brushwood, laid with the cut ends down stream, in a kind of bank along the line beyond which it is desired that the tailings shall not go. Obviously this system would not be effective against a stream, and it is applicable only on flat ground above ordinary water level; nevertheless it is much employed in Australia and Tasmania for retaining the tailings delivered from hydraulic elevators and centrifugal pump dredgers, while allowing the superfluous water to seep away through the sieve-like mass, freed of all but very fine matter in suspension. To be effective, the brushwood should be as long as possible, and with a good proportion of twig to leaf: large and many leaves are not desirable; nor a lesser length than 6 ft. Dams fully 20 ft. high may be built in this way, and will endure for many years.

Next in scope and in cost is the log and brush dam, Fig. 115. It differs from the brush dam in being reinforced by logs, measuring not less than 4 in. thick, laid lengthwise with the face of the dam, and alternating with about the same thickness of brush, placed as in the first case. At every 4 ft., or so, in its length, each log is firmly lashed to the one below it by means of ordinary galvanised-iron fencing wire, of a size to be strong yet supple enough for straining and plying with the hands. Further addi-

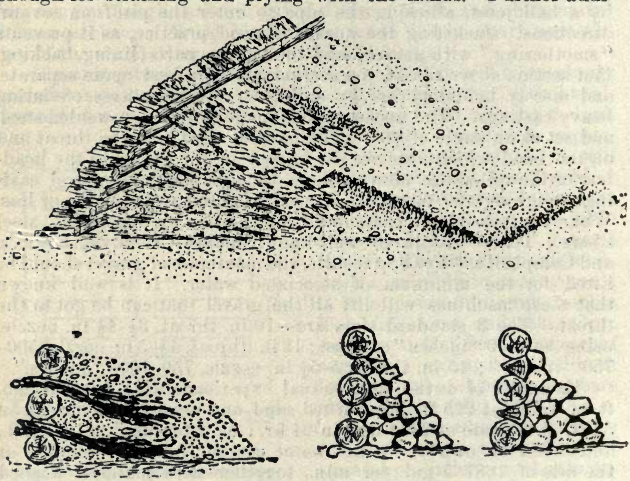


FIG. 115.—LOG AND BRUSH DAM.

tional strength and solidity are sometimes given by occasionally interposing heavy logs and branches between the layers of brushwood. As each bedding of brushwood is lashed in place, tailings are raked and scraped well over it, the dam building always being kept in advance of the growth of the bank of *débris*. The more it is trampled down, the better. While the up-stream face of the bank maintains a slope against the stream, the down-stream face of the dam should have a batter inwards of about 9 in. horizontally to every 1 ft. of vertical rise. When growing trees or stumps of reliable size and strength are available on the banks for tying to, they will obviously be beneficial. Such a dam, up to a height not exceeding about 20 ft., will tolerate a fair stream of water constantly flowing over it, besides what escapes through the interstices. If built in a stream with comparatively steep banks and a rapid fall, it is well to provide accommodation for storm water, when the dam has reached its maximum intended height, by cutting a trench

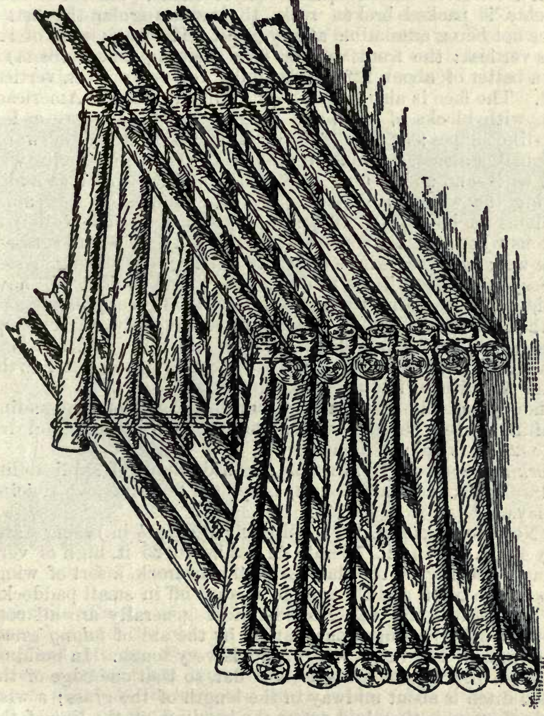


FIG. 116.—LOG OR CRIB DAM.

in one bank. Once the dam is filled, it can catch no more, and the principal object then is to safely hold what is already caught, providing new dams for further supplies. This type of dam is quite commonly used in all mining countries for retaining tailings, even tailings from stamp mills, which are never larger than sand and often mere slimes.

Most effective of all, except, of course, masonry dams, is the log or crib dam, illustrated in Fig. 116. This consists of a crib or "pigstye" built of large logs notched and fastened together by iron pins near their ends. This structure, commonly 15–20 ft. wide across the direction of the stream (i.e., from back to face), is in length divided up at similar distances by cross bracing, so as to form a series of united square compartments. Into these compartments is packed broken rock, the more angular the better, pebbles not being admissible at any price. The back wall of the dam is vertical; the front wall or face (looking down stream) is given a batter of about 1 ft. horizontally for every 10 ft. vertical height. The face is also packed, or "chinked" as the Americans call it, with blocks of wood and pieces of stone of more or less wedge-like shapes; or even with brushwood laid cut ends outwards and tightly jammed, the inner portion of the brushwood being well loaded with sand and stones. Old mine ropes are sometimes added for tying the whole back to big trees or stumps on the banks. Such dams are built 40 ft. high and more, and last indefinitely in situations where the timber—of decent quality to begin with—is always wet; and even where alternate wetting and drying hasten the decay of the logs, if the filling has been well done, it will have consolidated extraordinarily, and will withstand great pressure long after the logs have ceased to possess any holding power. The topmost set of the crib may well be made of much heavier timber, so as to admit of cutting channels for the overflow of the water in the last face sill without unduly weakening it.

When favourable sites can be found, the cost of impounding fine tailings is less than ·05*d.* per cub. yd.; with brush and log dams, ·125*d.*; with crib dams, ·15*d.*

Working Results.—To illustrate the wide range of applicability of hydraulicing, two examples may be quoted from tinstone sluicing in Malaya.

At New Gopeng (Fig. 117), 5 little monitors ($1\frac{1}{2}$ in.) using water at only 40 lb. per sq. in., operate on banks 10–25 ft. high of very free wash. Owing to the flatness of the bedrock, a sort of wing-dam has to be built, and the ground washed off in small paddocks. These wing-dams and the ground-sluices generally are all constructed in a highly ingenious manner by the aid of *lalang* grass, which attains a length of 7–8 ft., and is very tough. In building the ditch, a bedding of grass is laid flat, so that one edge of the intended ditch is about midway in the length of the grass; a wisp or hank of grass is then laid across the bedding, on the line of the proposed bank, and a certain amount of tailings sand is packed tightly all over the bedding; next, the loose half of the bedding is folded back, and more sand is packed on that. By repeating these operations, with plenty of trampling, the bank is gradually built to the required height, the face exposed to the wear and tear of the gravel sluiced through the ditch consisting of the rolls of grass.

These ditches are very quickly, easily and cheaply built, and endure for many months—more than long enough to fulfil their purpose. The gathering of the tinstone from the sluicing operations is carried on continuously by women and girls using *dulongs* or wooden pans.

At Bruseh (Fig. 118), stanniferous schists are dealt with. These are greatly hardened by metamorphism, and, *in situ*, are quite proof against removal by water alone, even at 225 lb. per sq. in. In such ground, the practice is to run in T-drifts, and shatter the mass by explosives. The drifts are carried in for about 25 ft., and then crossed. In the short ends are placed 2 to 6 cases

FIG. 117.— HYDRAULICING TINSTONE, NEW GOPENG.

of gelignite; the leg of the T is refilled with the excavated ground as tamping, and the charge is fired. The object is to produce a lifting and settling effect, but not to scatter the mass. Experience has demonstrated that dynamite is the best explosive. The drifts measure about 30 in. high and 18 in. wide. After the explosion has loosened up the ground, the larger stones are spalled to about 4-in. cubes, and the monitor is gradually applied, so as to produce a grinding and comminuting effect as the mass slowly slides down towards the sluices. A charge is used in one or other of the banks about once a month, and generally loosens about 25,000–30,000 cub. yd. The average working pressure is about 100 lb. per sq. in.; and the duty per monitor, including all lost time, 1150 cub.

yd. standing ground per 24 hr., maximum bank height being 200 ft. Working costs, including everything, average 1·68*d*. per cub. yd. without explosives, and 3·64*d*. with.

At two other Malay tin-mines, both hydraulicing a decomposed

FIG. 118.—HYDRAULICING TINSTONE, BRUSEH.

pegmatite (granite), one, using 2 monitors, treats 20,000 cub. yd. per mo., and the other, with 5 2½-in. monitors, 60,000 cub. yd.

A number of average figures for California (wages, 10*s*. per diem), quoted by Bowie, are:—

			Per cub. yd.
Grades, $1\frac{3}{4}$-$2\frac{1}{4}$ %;	banks, 20– 80 ft. high;	easy washing, few boulders:	2 – 4 *d.*
„ 4 -$4\frac{1}{4}$ %;	„ 50–150 „ ;	considerable blasting:	5 – 7 *d.*
„ $4\frac{1}{4}$-$4\frac{2}{3}$ %;	„ 20–100 „ ;	cement and boulders:	$7\frac{1}{2}$–$11\frac{1}{2}$*d.*
„ $4\frac{1}{2}$-5 %;	„ 100–300 „ ;	boulders and hard ground:	1 – $2\frac{1}{2}$*d.*

The La Grange mine (Cal.) in 2 yr. sluiced 2,275,967 cub. yd., on a sluice grade of $1\frac{1}{2}$ %, with a duty of 1·48 cub. yd. per "inch," at a cost of 3*d.* per cub. yd.

The N. Bloomfield mine (Cal.) in 3 yr. sluiced 7,071,000 cub. yd., on 4·33 % grade, with a duty of 4·43 cub. yd. per "inch," for 2·05*d.* per cub. yd.

A number of lumped returns (Cal.) showed 1,251,399 cub. yd. of gravel piped by 655,657 "inches" water per 24 hr. (or 1·91 cub. yd. per "inch"), in banks 50–130 ft. (av. 63 ft.) high, on a sluice grade of 7 in. per box (12 ft.) at a cost of: piping, ·96*d.*; bed-rock cuts, ·06*d.*; cleaning up, ·38*d.*; moving monitors, ·36*d.*; total, 1·76*d.* per cub. yd.

The Cariboo mine (B. Colum.), in 1904, sluiced 1,461,341 cub. yd., using 225,198 "inches" of water (average 6·49 cub. yd. per inch, ranging from 5 to 8·9), at an operating cost of ·97*d.* per yd.

Victorian figures, on 464,842 cub. yd., with an average depth of working face of 30 ft., and wages at about 8*s.* 4*d.* per 8-hr. shift, are about 3·3*d.* per cub. yd.. of which, wages represent 3*d.*, and maintenance and repairs ·3*d.* On "abandoned" alluvials, when water has to be pumped out and tailings elevated, the costs are 6*d.*–1*s.* per cub. yd.; and when using steam-driven centrifugal pumps for breaking down the bank and the same for elevating the tailings, costs are 3–6*d.* per cub. yd.

In Tasmania, the Briseis tin-mine (1905) sluiced 368,027 cub. yd. for a total cost of 27,059*l.* = 17·64*d.* per cub. yd.; but this cost included removing 551,000 cub. yd. overburden, mostly basalt.

Dredging.

The term "dredging" is applied to several very widely-different styles of alluvial mining. By far the most generally used, is the "bucket" dredge, but it is being rapidly followed by the "centrifugal pump" machine, while far behind come the "grab" and the "dipper." Each of these has its own particular merits and sphere of usefulness.

Bucket Dredges.—These consist essentially of a pontoon or hull to carry the machinery, means of anchorage, digging mechanism, value-catching appliances, tailings disposal plant, and motive power.

Pontoons.—These measure 80–120 ft. long, 30–40 ft. wide, and 7–9 ft. deep. They must possess great strength to withstand the enormous shocks to which they are liable. When of wood, they consume 50,000–70,000 ft. super.; hard woods are excellent for this purpose, but their weight might be an objection in shallow-draught hulls, and then pine may replace them for planking. Steel has come much into vogue (U.S.) lately, but is considered objectionable where submerged timber occurs, and the ground is shallow. Draught ranges from 3½ to 5 ft., with 24 hrs.' fuel on board. The bows should be the most heavily planked and framed, so as to resist damage by striking a face, as the dredge surges at her work; often they are sheathed with steel plates. The bottom, at the forward end, should be sprung or bevelled upwards, for clearance and handiness in working; and, from the bows aft to the position of the ladder at "maximum dredging depth," should have the planking considerably reinforced to obviate accidents through logs being caught in the buckets, and damaging the bottom. Longitudinal bulkheads, or fore and aft keelsons, are often introduced; transverse water-tight bulkheads at the bows would be advantageous for dredges working in rivers where there is danger of collision with floating trees and lumber. The size and stability of hull must be proportioned to the weight of machinery carried. Overhanging deck-houses give more room, and permit of installing a small though very convenient fitting shop. Crowning the deck makes it stronger and drier. More powerful gantries are now introduced: the bow gantry is not merely a support for the outboard end of the digging ladder, but is designed also to tie together the bow pontoons; the main gantry, which carries the upper tumbler and its driving gear, and the inboard end of the digging ladder, is made much stronger, when the upper tumbler has a gear drive (good alignment becoming essential), and it may well be made the vertical post in longitudinal and transverse trusses, tending to prevent distortion of the hull, by the great weights of the digging and tailing machinery at extreme bow and stern. Stern gantries are made to sustain longer stackers. Some form of derrick at the bow, to remove obstructions and to hoist machinery, is vital; and a travelling crane above the driving and other machinery is invaluable for saving time and labour during renewals and repairs. The machinery is best housed in most climates. The type of dredge and height of upper tumbler must determine the position of operating levers, controllers, and switches. Generally it would appear that advantages lie with a location behind the upper tumbler; but some operators prefer them close to the driving machinery (on the lower deck), and others in the pilot house (forward, on the upper deck). The main objective is that the winch-man shall see as much as possible both of the machinery and of the work being done. Really efficient lighting at night is most important, but rarely provided.

Anchorage and Manipulation.—Very diverse practice obtains in the two great dredging centres—New Zealand and California—in the methods of holding and bringing the dredge to its work. The

New Zealand dredge is held to the face of the bank by "headlines" run out over the bow to an anchorage, and the buckets are lowered in a vertical plane, the material being caved by undermining at the bottom. The Californian type is held to the face by a pivotal stern "spud," the buckets being side fed horizontally through an arc of 120°; the cut is begun at the surface, and the digging ladder is lowered about 1 ft. at the completion of each arc, no attempt being made to undermine or cave the material excavated. In the first case, all digging is done from the bottom; in the second, banks may be cut in terraces.

In the New Zealand method, the dredge is moored by two flexible wire ropes on each side, and a head rope, operated by steam winches having 6-8 barrels (one of which is used for raising and lowering the bucket ladder), and so arranged that all or any of them may be operated by one man. Assuming the dredge to be ready for starting, it is brought up to the point of operation by hauling on the head-line. The ladder is lowered, and the wash is raised until the buckets reach bedrock. It is then moved sideways, by gradually slacking out the mooring lines on one side, and hauling in those on the opposite side, until the end of the cut is reached. The head-line is then hauled in 1½–3 ft., according to the nature of the wash, and the cut is taken in the same way back to the starting point. The buckets must be raised or lowered as the surface of the bedrock rises or falls, in order to clean up the bottom. Thus everything depends upon the intelligence and skill of the winch-man, as to whether the dredge is kept lifting to its full capacity, and the bottom is cleaned up or not. For fairly level ground, with soft and shallow wash, lines are certainly preferable: they admit of more rapid change of position, allow the stern to be moved independently of the bow, and their resiliency lessens jars and shocks.

In the Californian method, two spuds (one steel, one wood) are employed. When the metal spud is lifted, the dredge is walked ahead by the wooden spud. Bow shore-lines serve to move the hull from side to side, through the arc of a circle, similar lines being fitted to the stern. On uneven ground, with deep and hard gravel, there is much to be said for this practice. Large dredges are often equipped with both.

The use of the pivotal spud introduces problems in the disposal of tailings. The spud should be so located as to make the radius of the arc described by the buckets in digging as long as possible, permitting a wide cut. This demands placing the spud at or near the stern. The old short stackers and tail-sluices resulted in a narrow distribution of tailings, more particularly the fines, and the stern of the dredge was frequently aground, unless kept clear by a sand pump. This has been partially remedied by lengthening stackers and tail-sluices, but will always be more serious for the spud dredge than for the headline dredge. The latter can use the maximum tailings room. In dredging a confined area, tailings can be deposited on barren ground outside the cut. To accomplish this with a spud dredge, numerous changes of position would have to be made, resulting in much loss of time and some loss of ground, because of the difficulty of relocating pivotal points. (Hutchings.)

Wasting of unworked ground by faulty tailings-discharge is shown in Fig. 119. Here the distance from pivotal point to lower end of bucket-ladder excavating at the maximum depth (called the digging length) is 90 ft.; *px* and *py* represent the distances from pivotal point to point of discharge of stacking ladder (called tailings lengths). The discharge is at points *z* and *z'*, only 18 ft. and 5 ft. from boundaries of cut, whereas they should be not less than 40 ft., if the dredge be digging in ground 20 ft. deep. (Hutchings.)

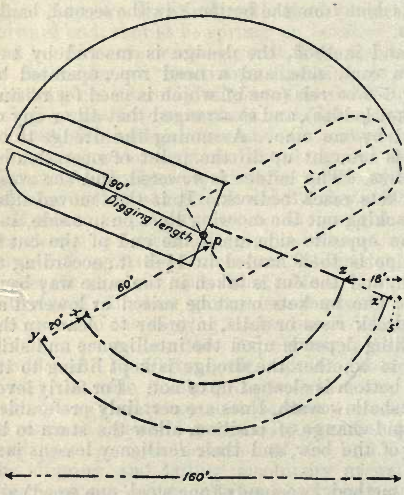


FIG. 119.—SPUD DREDGING.

Fig. 120 shows a method in general use to prevent tailings covering unexcavated material adjacent to cut. It necessitates moving the dredge as often as the distances from *a* and *a'* to *x* (while moving ahead in excavating arc *a*, *b*, *c*, and the advancing of tailings in arc *x*, *y*, *z*) become so short as to make it difficult to move to and dig in the other half of the cut. Shifting the dredge is necessary every few days, and consumes much time; besides the danger of the dredge not being located with the pivotal point in the right place, with repeated loss of ground. It is assumed that the dredge is digging 20 ft. below the surface of a pond on a bank 30 ft. deep. Keeping the lines of advance of the pivotal points too close together would result in grounding the dredge; if too far apart, ground adjacent to the cut will be wasted. In general, where the material to be dredged is free, the headline method is superior; and if "surging" can be prevented, the headline method will be superior in dredging indurated material, for there the problem of the disposition of the tailings is much simpler, permitting shorter stackers and tail-sluices, smaller hulls, and lighter construction. The pivotal spud should be used only by dredges of the double and single lift type, whose sluices are sustained by auxiliary scows. Several dredges of these types, having broken pivotal spuds while dredging tenacious material, have substituted a system

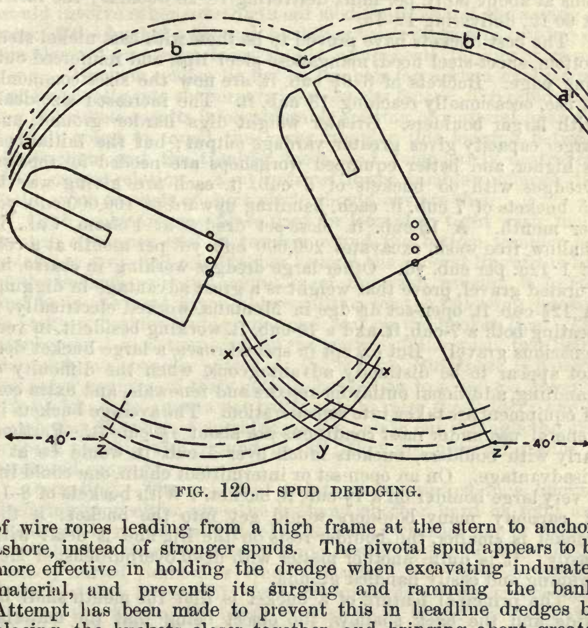


FIG. 120.—SPUD DREDGING.

of wire ropes leading from a high frame at the stern to anchors ashore, instead of stronger spuds. The pivotal spud appears to be more effective in holding the dredge when excavating indurated material, and prevents its surging and ramming the bank. Attempt has been made to prevent this in headline dredges by placing the buckets closer together, and bringing about greater continuity of contact between buckets and dredging surface; but not entirely successfully. It is possible that the introduction of rollers on the lower side of the bucket-ladder, and behind the lower tumbler, would result in holding several buckets firmly against the material being excavated, and prevent surging, as it is probable that this movement, in digging hard material, is caused by the alternate taking hold and letting go of the buckets, only one bucket at a time being held firmly by the lower tumbler. The buckets behind the lower tumbler dig by their weight only, which is insufficient to force them into hard material. (Hutchings.)

Digger.—The digger consists of an iron or steel ladder-frame, upon which travels a continuous chain of buckets. These buckets may be "close-set" (one to every link), or "open-set" (one to every other link). The former gives better efficiency in loose (and even in fairly tight) gravel where there are no large boulders; the latter lasts longer where there are many boulders: the former runs at about 50 ft. per min., delivering 18–25 buckets; the latter, at 60 ft., delivering 12–15.

The best buckets have proved to be those with cast nickel-steel bottom, sheet-steel hood, manganese-steel lips, and reinforced cutting edge. Buckets of 3–8½ cub. ft. are now the sizes commonly in use, occasionally reaching 13 cub. ft. The increased size deals with larger boulders. Greater weight digs harder ground, and larger capacity gives greater yardage output; but the initial cost is higher, and better equipped workshops are needed for repairs. Dredges with 65 buckets of 6 cub. ft. each are giving way to 75 buckets of 7 cub. ft. each, handling upward of 100,000 cub. yd. per month. A 13-cub. ft. close-set dredge at Folsom, Cal., in shallow free wash, excavates 200,000 cub. yd. per month at a cost of 1·12*d.* per cub. yd. Other large dredges working in coarse, indurated gravel, prove that weight is a great advantage in digging. A 12½-cub. ft. open-set dredge in Montana, worked electrically, is beating both a 7-cub. ft. and a 10-cub. ft. working beside it, in very tenacious gravel. But except in special cases, a large bucket does not appear to be distinctly advantageous, when the difficulty of handling, additional outlay for repairs and renewals, and extra cost of equipment are taken into consideration. The average buckets in general use under most conditions are about 4½ cub. ft. Particularly with boulders, buckets much over 5 cub. ft. would be at a disadvantage. On an open-set or intermittent chain, one could lift a very large boulder on a 4½-cub. ft. bucket. With buckets of 8–12 ft. capacity, many boulders would get into the bucket; if the bucket is smaller, the boulder rests on the lip, and is dealt with easily. It is quite another matter with deep overburdens, or free running and easily handled ground.

The principal feature of a bucket is that the mouth shall be wide and the back shallow, the object being that the material dredged may fall more readily when the point of discharge is reached. With the earlier designs considerable losses occurred by spilling, except when buckets were working at maximum depth: the bucket was long, narrow, and deep, and a clean discharge was not effected, especially when working in sticky or sandy material. The use of water jets to wash adhering material from both inside and outside of buckets, is now common; and a loose loop of chain across the bottom of the bucket is a great aid to complete discharge with tenacious dirt.

The bucket-chain passes, at top and at bottom, around "tumblers." These formerly were made with few faces—a square section being most approved, as being most capable of firmly holding the bucket, and compelling rotation of the bucket with itself; but the use of lower tumblers of more numerous faces causes more buckets to dig at a time, and results in less surging. Possibly a round tumbler of large diameter might entirely eliminate surging, for there would then be no raising and lowering of the buckets, as there is during the revolution of the tumbler of few faces. It would involve other constructional modifications, but the saving in wear of the bucket pins and bushings by deflection in turning on tumblers of numerous faces, should compensate for any extra expense. Buckets seldom lie flat on tumbler faces when excavating, but frequently ride the corners between the faces. Too rapid feeding of buckets results in considerable spilling of material upon the buckets about to round the lower tumbler, this lodging between the buckets and the tumbler faces, and temporarily destroying their proper relation. A cylindrical lower tumbler of large diameter, with buckets of short pitch, should obviate this.

Grab hooks are sometimes used for loosening tight wash, made in such a manner that in turning round the bottom tumbler the points are projected some inches beyond the general line of the lips of the buckets, so that they undercut the wash, causing it to fall, and be in the most favourable condition for the following buckets to pick up, besides saving wear and tear on the bucket lips. The number used is determined by the nature of the wash, two being the minimum, equally spaced in the chain. But there is danger, in timber country, of their getting hooked on to a log too heavy to lift, and in consequence not being able to raise the ladder; and in tough clay they are of no value, as the points cut through the clay without causing any ground to fall; they are, however, useful in cemented ground.

Operating.—The principal points to be studied in a bucket-dredging proposition are—the bottom (whether hard or soft), and the depth from the water level; the wash (whether fine or coarse), and whether there are many large boulders, bands of cemented gravel, buried timber, or other obstacles; the metal or mineral (whether fine or coarse); and the quantity of water available. Should the bottom be hard, or the wash coarse, or both, the buckets and driving gear must be made specially strong, and grab hooks or picks be inserted in the bucket chain; if a large proportion of sand in the wash, special provision must be made for getting it clear of the dredge, which ordinarily is not allowed for. Comparatively shallow gravels, with excess of water and soft bedrock, form ideal places. One of the greatest drawbacks to bucket-dredging is buried timber; in torrential streams flowing through timbered country, and in alluvials deposited in what were well-wooded valleys, imbedded trees are a great nuisance. When a log is struck, it always means a stoppage—and it is the difference between the stoppages (due to this and other causes) and the value of the wash put through, that makes a dredging proposition payable or not. Slinging out by a derrick, even after sawing up, is not always feasible. Large boulders can be dealt with, so long as they are not over a couple of tons, and are not too frequent. A large body of flotation water, such as a flowing river, is not at all necessary, provided water can be obtained at a reasonable distance. This "paddock" dredging has an advantage over river dredging, in that the dredge is immune from floods (which are always more or less dangerous, and certainly detrimental), and that the water level can be raised or lowered to suit the conditions. Yet every dredge needs to be carefully adapted to the conditions of the particular area to be dredged; of the hundreds at work, no two are exactly alike.

At starting, in the case of dry ground, an excavation is made equal to about twice the length of the dredge square, and 4–5 ft. deep according to the draught of the dredge. This excavation is filled with water from the nearest supply, either by pumping, races, or fluming; and the dredge is launched into it. Usually about 2 cub. ft. of water per sec. is all that is necessary for working the dredge. Once started, it cuts its own flotation way, since any material taken away from the face is treated and put over the stern. The dredge-master's great object is to open out a face across the ground for its full width, and to get the buckets down on to bedrock, as soon as possible. In carrying this out, attention and skill are required in manœuvring the dredge, as there is little space for stacking tailings. The face has to be opened out in the shape of a fan, and gradually worked ahead, lowering the buckets until the bottom is reached. In opening out deep ground, one is often a month in getting to bottom. If care is not taken in the initial starting, the position may become so cramped that the dredge will finally be immovable and have to be dug out. When once a wide paddock is opened out, there is very little trouble. The face must be kept square, and the corners of the paddock worked up, otherwise it will gradually narrow, and ground will be left behind and lost. The tailings must also be evenly distributed at the stern.

To facilitate working, the mooring lines, of which there are 5 (2 on either side at the bow and stern, and a head-line), have to be properly set, so that the winch-man has no trouble in moving from point to point. Bow lines should always be set to run aft from the point where the line leaves the dredge, and the stern lines forward. A long head-line is requisite, as the dredge gets a better swing; and, in working a wide face, two pennants or free lines are moored in convenient positions, and to these the head-line is attached with an iron shackle, as occasion may require. Deep T-shaped trenches are used for mooring the lines: a log with a chain-sling secured round its centre is thrown into the top trench, and the sling is led out of the other, and attached with a shackle to the dredge-line.

Boulders that will come through the hanging bars of the ladder are raised to deck level, and then taken out of the bucket and thrown on deck. Those that will not pass through the hanging bars may be pushed bodily along the bottom (to the side of the paddock) by the ladder, and there left; or the buckets may be permitted to work at the boulder until one lifts it; it then travels up the ladder until the hanging bars are reached, when a friction clutch on the second-motion shaft slips, the buckets stop, and are pulled out of gear, and the ladder is raised until the stone appears above the water; the dredge is then pulled round, to bring the bows as near the tailings as possible, and, by reversing the buckets, the boulder is thrown back into the paddock out of the way.

With logs and sunken timber, much time is apt to be lost. The position of a log relatively to the face will determine the ease with which it may be removed. A log lying parallel to the face of the paddock is easily stripped and lifted. When embedded at an angle, it has to be got out in sections, the buckets stripping and lifting about 10 ft. at a time. As soon as an end appears above water, a chain sling is thrown round it, and attached to a rope from the derrick, and a constant strain is kept on it; as each portion is stripped and lifted above water-level, it is chopped or sawn off, and swung round by the derrick.

Generally it is preferable to dredge with the buckets trailing on the bottom, so that the latter may be continually scraped and cleaned as the buckets revolve. But when there is a layer of wash on the bottom only a few feet in thickness, below an overburden carrying no value, the wash is stripped and passed through the dredge separately. A cut of 3–6 ft. ahead is made over the wash to take off the overburden, and then the dredge is let back, the ladder is lowered to the bottom, and the wash is raised. Dredges are now constructed to dig 60 ft. below the surface; this means that it is possible to handle wash more than 75 ft. deep (by having a bank more than 15 ft. above water level).

All dredges should be equipped with devices for hoisting and moving the driving, screening, and pumping machinery, but these are often lacking. The belting, for stacking and driving, may cause much loss of time, if of poor quality, or if an attempt is made to apply belts to work for which they were not intended. Friction-clutches, hoppers, and screens of bad design and inaccessibility of parts, especially baffle-plates of hoppers and perforated sheets of screens, are also responsible for many delays. Cleaning-up, too, consumes much time, and an arrangement to permit continuous running during clean-ups is most desirable. Some managers employ larger crews to minimise lost time, though most dredges are still operated with but two men per shift.

Electric power.—The success which has attended the application of electricity to operating bucket dredges is most remarkable, considering the slow speed of the principal machinery, and the extraordinary variation of load. The following examples are all from American practice.

(*a*) 5-cub. ft. buckets; 50,000–75,000 cub. yd. per month; 10–50 ft. deep. A 100-h.p. variable-speed induction motor drives the digging buckets, and the drum for raising and lowering them; a 50-h.p. 2200-volt motor is direct-connected with the shaft of 2 centrifugal pumps supplying water for washing; a 30-h.p. 440-volt motor on the deck of the dredge drives the shaking devices, and operates a large rubber rock-conveyor belt; a 20-h.p. 440-volt variable-speed motor drives the winches with which the dredge is moved. A special sheet-iron lined compartment is built in the outside of the boat for accommodating the transformers, which step the current down from 30,000 volts to 2200 or 440 volts.

(*b*) 7½-cub. ft. buckets. A 150-h.p. motor drives the bucket chain; a 20-h.p. motor drives the deck winch; a 75-h.p. motor controls the centrifugal pump; the tailings stacker is driven by a 20-h.p. motor; and a 10-h.p. motor runs the deck pump.

(*c*) a 100-h.p. motor is used in digging, and a 50-h.p. for the centrifugal pumps. The variable-speed motors of this dredge make it possible to cut at any desired rate, and the swing of a cut is about 90 ft. wide. The bucket line can be run fast or slow, and a single man controls operations.

(*d*) 3000 cub. yd. a day. The buckets are driven by a 100-h.p. motor; a 10-in. centrifugal pump driven by a 75-h.p. motor, and a 6-in. centrifugal pump supply the water for sluicing. Other small motors operate the side line, speeds, etc., using in all about 210 h.p. The whole dredging plant is lighted by 100 incandescent lamps.

(*e*) 5-cub. ft. buckets; 85, at speed of 22 per min., built to cut through hard pan. A 150-h.p. motor drives the bucket chain; a 15-h.p. motor controls the winches and the movement of the boat; a 40-h.p. motor direct-connected to a powerful centrifugal pump furnishes all the water for washing the gravel; a 30-h.p. motor is also connected with the plant for working the sand-pump, which carries the accumulated sand behind the boat to the tailings heap through a long pipe; a 3-h.p. motor is used for operating a deck- and bilge-pump for general washing purposes; the shaking screen is operated by a motor of 15 h.p.; and a 15-h.p. motor drives the conveying belt, which is 30 in. wide and 90 ft. long.

Saving Values.—The first desideratum is an efficient screen, to remove everything but the finest gravel, to thoroughly disintegrate all concretionary matter, and to wash adherent clay (and values) from the surfaces of large stones. Both the shaking and revolving type (trommel) are employed. The former, it is urged, has a larger effective area, is easier to repair, and gives better distribution of the screened material over the tables. But the latter is more effective in disintegrating, in cleaning, and in puddling clay, requires less power, and is cheaper to maintain. The selection is mainly determined by the hardness of the wash, and the presence or absence of clay. For fine gold, $\frac{5}{16}$–$\frac{3}{8}$ in. screen holes suffice. Where fine and coarse gold are mixed, it is suggested that the lower sections of the screen be provided with larger holes, the material thus differently screened being conducted to separate sluices and tables, thereby avoiding the loss of fine gold that would occur in the deeper and stronger stream of water needed for the coarse material. Where there is overburden of any considerable depth, a simple means of blanking the trommel or screen will enable barren material to be stacked well behind the dredge and more rapidly than if handled in the ordinary way. A suitable shaking screen for a 5-cub. ft. bucket-dredge is not less than 750 sq. ft. in area, which means occupying a very large space. Trommels are 3–5 ft. diam., and 20–25 ft. long.

Californian gold-dredges are fitted with tables and with stern sluices. The paving for both is either the ordinary cross-riffle with quicksilver, or coconut matting overlaid with expanded metal $2\frac{1}{2}$ in. mesh. Opinion seems rather in favour of riffles with quicksilver, provided the gold amalgamates freely. But more platinum is saved by matting. Stones, inserted between the riffles, and standing up above their tops, prevent choking by black sand. The diamond-shaped expanded metal, about $\frac{1}{4}$ in. deep over the matting, serves the same purpose.

New Zealand gold saving tables generally consist of a series of strakes each 3 ft. wide. These strakes are first of all covered with calico, then with coconut (coir) matting; on top of this, expanded metal, similar to that used by plasterers, is laid, making an excellent and delicate riffle. Sometimes, where the gold is fine, plush is used on the tables. The fine stuff passes over the tables into the tail shoot, which delivers it over the stern of the dredge, the gold and concentrates being collected on the matting; this is picked up, in sections, without stopping the dredge, as often as necessary. The introduction of a preliminary "boil-box," or agitator (by means of a series of baffles), has been found necessary when the gold is fine and is associated with "black iron sand" (ilmenite, etc.).

There is no doubt that great improvement in gold saving would be attained by placing the tables on a separate pontoon, which would afford a much larger area, and be free from the vibration and oscillation inherent in bucket dredging. Incidentally, the fine tailings would be delivered farther astern.

Tailings.—The principal forms of tailings stacker are bucket-elevators and belt-conveyors. The bucket-elevator works up to an angle of 35° from the horizontal; it is durable, but costly in erection and in motive power. The belt-conveyor is said not to work well above 20°, but it is generally regarded as superior, because of greater economy of operation and less weight for a given length; it may, however, be a source of large expenditure if ill-selected for the wearing qualities of the carrying face. A belt conveying 75 cub. yd. per hr. will travel at about 250 ft. per min. The Robins conveyors most commonly used are 80 ft. long and work at an angle of 18°, permitting tailings to be stacked 35 ft. high; they are made in all widths between 20 and 36 in. A great advantage of the Robins conveyor for this work lies in the fact that power can be applied at the lower end, saving the long and troublesome transmission of power by rope or chain to the head end.

The longest elevator working to full capacity in New Zealand is 112 ft. long, and is stacking tailings 55 ft. above water level, the total depth of face the dredge is treating being 70 ft. (30 ft. above water and 40 ft. below), and the cost of running and maintaining this elevator does not exceed 1*l.* per week.

Two other types of tailings stacker have been adopted to some extent in New Zealand—Roberts's wheel and Peck's centrifugal. The latter consists of a series of cast-steel beaters, driven at a speed of 240 rev. per min., and furnished with renewable liners, which throw the tailings clear of the dredge in an even stream. A machine 2 ft. wide and 3½ ft. diam. deals with 50 cub. ft. per min. It seems to be distinctly superior to the bucket-elevator.

Working Results.—The labour employed on a dredge is small. A full crew comprises 3 winchmen (each working a shift of 8 hr.); 3 engine-drivers (ditto) to attend to machinery and boxes, and fire the boiler; 1 labourer on day shift, supplying fuel for 24 hr., and attending to line shifting; and a dredge master in command who superintends all operations. One engine-driver is a fitter, the second a blacksmith, and the third a carpenter; thus the crew is able to attend to all repairs. Australasian rates of wages to crews are usually as follows—winchmen, 10*s.* per 8 hr.; engine-drivers, 8*s.* 4*d.*–10*s.*; labourers, 7*s.* 6*d.*; engineers, 4*l.* per week; dredge master, 6*l.* per week. The last-named often receives also a bonus on returns; the engineer gets no extra pay for overtime, but all the others do.

Water-supply should always be independent of digging-machinery power In some climates, the sluice-box water is warmed by passing it through the engine condenser. The consumption averages about 2000 gal. water per cub. yd. solid wash raised per min., ranging from 1500 to 3500.

Lost time, from all causes, reckoning the full year at 52 weeks = 7488 hr., varies from 10 to 30 %, leaving 70 to 90 % of actual working time: figures from a number of sources show the following percentages of activity—69·4, 72, 89·9, 85·4, 73·7, 75, 84·6, 72·6, 82·3, 74·4. In New Zealand, with fairly good conditions, 5 days a week is reckoned on.

Capacity of output in practice as compared with theoretical capacity may be anywhere between 30 and 70 %. In all unfavourable ground—whether hard, clayey, bouldery, timbery, or shallow—and always while cleaning bottom, only the lowest figure should be counted on. Theoretical capacities, with a dumping speed of 12–17 buckets per min., are—87–120 cub. yd. per hr. (for "$3\frac{1}{4}$-cub. ft." bucket), 132–190 (for "5-cub. ft."), and 185–265 (for "7-cub. ft.").

Power required to be available at all times is on the average about 7 times as great as that demanded in theory for the ft.-lb. of work to be done. A common initial basis of calculation is 66–75% of $1\frac{1}{2}$–$1\frac{3}{4}$ i.h.p. per cub. yd. of bucket capacity per hr., this allowing a fair margin. Some actual figures are:—

Depth of Wash.	Capacity of Bucket.	Output per Month.	Power Provided.	Power Used
ft.	cub. ft.	cub. yd.	h.p.	h.p.
30	3	40,000–45,000	148	100
30	5	54,000–60,000	138	94
40	4	40,000–50,000	175	100
40	5	54,000–57,000	200	120–150

Working costs (New Zealand) range from 1·15*d.* to 5*d.* per cub. yd., and will average out at about $1\frac{3}{4}$*d.*, including all upkeep, office expenses, etc. Prime costs of dredging plants are 5000–12,000*l.* (but rarely more than 7000*l.*) for a nominal capacity of 800–2400 cub. ft. per diem, with 4–5-cub. ft. buckets. Three examples in New South Wales record 2·4*d.*, 2·4*d.*, and 2·3*d.*, inclusive of everything. Two companies in Victoria publish the following—(*a*) treated 4,653,026 cub. yd., of an average depth of 14·2 ft., costing 1·9*d.* per cub. yd. (wages, 1·2*d.*; fuel, ·4*d.*; repairs and maintenance, ·3*d.*); (*b*) treated 262,960 cub. yd., average depth, 12–13 ft., costing 1·45*d.* (wages, ·777*d.*; firewood, ·392*d.*; repairs and renewals, ·173*d.*; rents, office, law, and general charges, ·108*d.*). Californian figures, with deeper and tougher ground, and somewhat higher wages, rule from $2\frac{1}{2}$*d.* to $4\frac{1}{2}$*d.*, and are mostly between 3*d.* and $3\frac{3}{4}$*d.* Prime costs of plants are 8000–20,000*l.*, to handle 30,000–65,000 cub. yd. per month, digging to 40 ft. and stacking 20 ft. above water. Actual working cost (Hohl) from 5 plants show:—

	d.	*d.*	*d.*	*d.*	*d.*
Labour	·82	·91	·92	1·16	1·03
Power	·53	·60	·58	·80	·88
Repairs	1·43	1·51	1·73	1·49	1·90
General expenses ..	·32	·34	·62	·64	·37
Total	3·10	3·36	3·85	4·09	4·18

Centrifugal-pump Dredges.—These operate by means of a hydraulic jet, directed against the face of the wash in the same way as in hydraulic sluicing. The wash is concentrated into a sump, whence it is lifted by a centrifugal pump into an elevated sluice-box. The plant is mounted on a wooden pontoon or barge, so that, when the face of wash gets too far away, the hole may be flooded and the barge floated to the next point of operation. Work is commenced by making an excavation about 50 ft. square and 6 ft. deep. This is filled with water, and the barge, built on the bank, is launched. The machinery is then placed on board, and the whole is housed in with galvanised iron. The barge is 35–45 ft. long (not counting the 4-ft. tailboard sometimes added), 30 ft. wide and 4 ft. deep; when loaded, it draws 3 ft. of water. The capacity of the engine and boiler will depend on the depth of wash to be worked—generally 130–170 i.h.p.

A typical plant has a cross-compound engine, with 10-in. high-pressure and 20-in. low-pressure cylinder, stroke being 24 in. The boiler, of marine type, 20 ft. long, 7 ft. diam., fitted with 38–60 4-in. tubes, burns about 80 cords firewood per month. Working pressure is 120–125 lb. Failing a natural head of water to sluice with, an artificial head is obtained by a 12-in. centrifugal pump, known as a "pressure," "sluice" or "nozzle pump." Water is conveyed from this, along a spiral-riveted pipe (14 gauge, in lengths of 18 ft.), to the nozzle at the face, which is generally 4 in. diam. Only one jet of water is used, at pressure of 60–70 lb. per sq. in. The water and gravel are elevated from the sump to the sluice boxes by a 10-in. centrifugal pump, known as a "gravel-pump." In addition, there is a washing-down pump (direct-acting plunger). The sluice-boxes have a total length of 80–120 ft., are 4 to $4\frac{1}{2}$ ft. wide, 12 in. deep (except at the head, where they are 18 in.), are made of $\frac{1}{16}$–$\frac{1}{8}$ sheet iron, and are in 10–12-ft. lengths, so as to be readily taken apart and re-erected. The head of these boxes rests on trestles, built on the barge, and passing through the roof of the housing; other trestles (10–12 ft. apart) are erected temporarily on the ground and shifted as the barge changes position. A crane at the end of the barge lifts heavy weights. Most plants have an electric-light installation, driven by a separate engine: incandescent lamps on the barge, and arc lights at the workings.

By the time the machinery is placed on board the barge, another excavation has been made to bedrock, and a level bed prepared for the barge to rest on when working, bed-logs being laid to keep the structure off the moist ground, and allow air to circulate beneath it. When ready, this excavation is flooded and the barge is floated in. The pump then empties the excavation, the barge settling into position. A sump is sunk in the bedrock near the barge; this should not be deeper than 20 ft. (as that is the effective suction limit of the pump). Races are cut in the bedrock from face to sump as the work progresses, on a grade of 1 in 24. The main race is cut up the centre of the paddock, and branch races meet it at suitable angles. The actual sluicing is carried out in the usual manner. Very large stones are forked out of the race, and stacked on either side; and an iron grating near the sump arrests any stone too large for the gravel-pump.

At first, tailings are stacked on the surface; later on, they fill the worked-out paddocks. A low brush-and-log tailings dam behind the barge impounds the tailings, and is gradually raised till the old paddock becomes filled up. Water is drained off through a large wrought iron pipe into the lowest portion of the worked-out ground, and can be used over and over again for sluicing purposes, saving about two-thirds of the total water used. The drain pipe is left, and becomes buried. Water has to be strained before being returned to the pump.

The ground is worked out in paddocks, with an area of 1–2½ acres; under ordinary conditions, it takes a month to work out ¾ acre. When the working face gets so far away from the barge as to necessitate cutting a deeper race to keep a suitable grade (in hard ground, an important item of expense), the pipes connected with the sluicing-pump, and the suction of the gravel-pump, are disconnected, as are also the sluice-boxes on the barge from those on the bank. Meanwhile bedrock is thoroughly cleaned, a new barge-bed is levelled, and a new sump is sunk alongside it. The paddock is then flooded sufficiently to float the barge to its new site, the connections are remade, and the water is pumped out: 1 week in 16 is the time generally allowed for moving.

The power required to work such a plant depends largely on the height the gravel has to be lifted—measured from bottom of sump to head of sluice-box—up to, say, 90 ft. Above 60 ft. (the limit of one gravel-pump for good work), it is better to use two lifts, working the pumps in series. If the ground is shallow, 20 ft. or less, the pump may be placed on piles, and driven by a portable engine from the banks.

The wear and tear of the gravel-pumps is, naturally, very great, for they are capable of passing boulders up to 50 lb.; but the wearing parts are renewable, and the bearings are made sand-proof by means of clean water injected under greater pressure than that at which the pumps are working, so that leakage of clean water into the pump keeps back any sand and grit. The pumps are primed at starting with a steam injector. They are driven by rope gear, the driving pulley being 8 ft. diam. and the pump pulley 2 ft. 7½ in. When running at about 350 rev., the pump will lift 1 cub. yd. gravel and 400 cub. ft. water per min. Since it costs nearly as much to pump water as it does to pump sand and gravel with it, one should try to pump the maximum quantity of wash with the minimum quantity of water. This is best done by increasing the grade of the race, which necessitates moving the barge more frequently. The minimum quantity of water required per cub. yd. of gravel raised by such a pump is about 2500 gal.; but the proper mixture suitable for pumping requires 15 times the bulk of water to gravel. The pumps, besides elevating gravel and sluice-water, act as drainage pumps, by raising the water that may drain into the excavation. The suction pipe of the gravel-pump is of smaller diameter than the delivery, to avoid any chance of a stone that has passed the suction blocking the delivery pipe.

Two types of centrifugal gravel-pump are employed in Australia —the "beater" and the "port-runner" In the former, the runner is made of wrought iron with renewable steel blades, so that wear and tear can be made good quickly and at least expense. The liners of the casing are renewable and adjustable. As the sides of the blades become worn down, causing leakage, the side liners are pushed forward by set screws, and held in position by stud bolts. The lining, in sections, is easily handled and replaced. In the port-runner pump, the runner is shrouded at the sides, but open at the periphery, thus no wear takes place at the sides of the blades, and there is no necessity to take up the sides; but the space between the shrouding and the casing is worn by grit that gets between. The lining, all in one piece, is heavy to handle.

Whilst fully admitting that the raising of gravel by means of centrifugal pumps is quite opposed to true engineering principles, being exceedingly wasteful of power, the fact remains that this form of "dredging" is highly successful under many conditions that entirely prohibit bucket dredging. It scores particularly where the bedrock is hard and irregular and carries value, where much timber is imbedded, and where a body of water is lacking. A very important condition for success is abundant power for furnishing gravel, so as to avoid raising water only. Another desideratum is to build the whole of the saving appliances on fixed trestles ashore.

Many of these plants have been working for years in Victoria. One group of five companies, at Castlemaine, aggregating 834,611 cub. yd. in a year, shows an average working cost of $5\frac{1}{2}d.$ per cub. yd., ranging from $4\cdot14d.$ to $6\cdot76d.$ Another plant, raising 132 cub. yd. per hr. (including stoppages), lifting 60 ft. above pump and with 26 ft. suction, working ground 45 ft. deep (av.), and using 15,000 gal. water per min., has a cost of only $2\frac{1}{2}d.$ per cub. yd.

More remarkable still is the Pioneer mine, Tasmania. Here the banks are 40–112 ft. (av. 66 ft.) high. Pumps raise the gravel a 75 ft. actual lift, and have a maximum suction (vert.) of 18 ft. The face is worked as far as 600 ft. away from the sump, provided the bottom slopes to the sump. Largest pebble is about the size of a man's head, and not many of them; occasional huge boulders are washed round and left. Firewood costs 5*s.* 2*d.* per "ton" of 2500 lb. (or 80 cub. ft.) when bought in 20,000-ton contracts; it is hauled 2 miles. Boilers, at the Author's visit (1906) worked at 160 lb.; feed-water heaters (to 200° F.) and superheating (to 400° F.) were being introduced. Wood consumed = 6·9 lb. per i.h.p. per hr. About 3 t. wood = 1 t. coal (ordinary) = coal at 15*s.* 6*d.*

Maximum efficiency, 3 lb. wood evaporate 1 lb. water. Plant capacity, 1–1½ cub. yd. per min., using 400 h.p.; efficiency, 34½–41½% (av. 37%) on 5000 gal. water and 1 cub. yd. gravel per min. Power costs 18–22*l.* per h.p. per ann. (365 days). Wages are 7–8*s.* per 8 hr. Shifting pontoons, including rebuilding sluice-boxes on

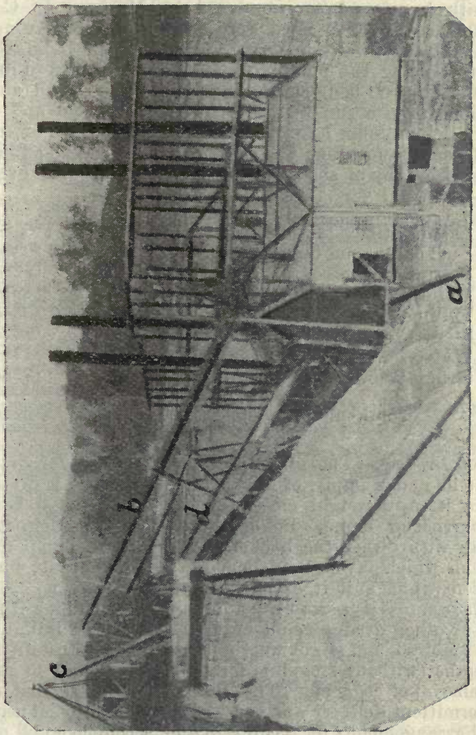


FIG. 121.—CENTRIFUGAL-PUMP DREDGING.

bank, occupies 3–6 weeks. One paddock lasts about 2½ yr., so that about 1½ wk. is lost per ann. in shifting. Some of the wash is cemented: this is holed by a coal auger, blasted (gelignite), spalled, nozzled, and pumped. Bottom is rotten granite. Overburden nil. Bucket dredging is impossible, because the "cement" undermines and caves badly. Sluice-boxes (pole riffles) are about 600 ft. long, 8 ft. wide, 2 ft. deep. Clean-up occupies 2½ days once in 6 wk. Output, 400,000–450,000 cub. yd. (=about 3 acres) per ann. Water ratio, 25 to 1 of gravel, volume being more important than pressure. Pontoon built of Oregon pine. Pump (Otis Co., Melbourne) life, ordinary: runners, 74,340 cub. yd.; liners, 78,000 cub. yd. Pump life, with improvements introduced by Manager E. C. Ryan: 104,730 and 80,570 cub. yd. respectively. Highest runner life, 147,000 cub. yd.; lowest, 50,500. Plant is shown in Fig. 121: pontoon carrying the pumping machinery is shown in its normal position, resting on bedrock, and in full operation; wash (and water) is pumped from a sump or well by suction-pipe *a*, and elevated through delivery-pipe *b* to tin-saving sluice *c*, erected on solid rimrock of basin from which tin-bearing gravel is raised; fuel supply is lowered by way of shoot *d*. Detailed working costs over 18 mo. were:—

	Per cub. yd.
Labour at face and in sluice	1·658*d*.
Fuel (blue gum firewood)	1·038*d*.
Labour on pontoon	·923*d*.
Maintenance of plant	·391*d*.
Tailings dam	·334*d*.
Stores	·262*d*.
Water	·179*d*.
General charges	·756*d*.
Total running costs	5·541*d*.

Grab Dredges.—The grab (or clam-shell) type of dredge consists, as its names imply, of a sort of scoop, which is dropped into (and, by its own weight, closes upon) the material to be dredged by a derrick or crane. Being intermittent in action, it cannot compete with either the bucket dredge or the pump dredge in situations which favour and demand a large output; but, under conditions which suit it, excellent work may be done. Among its advantages are—its portability and handiness (making it available where no other dredge could be used); its small initial cost, low power required, and slight wear and tear; its applicability to varying depths (without stopping for adjustment), and to rapid and easy removal of logs, etc. The chief drawback is that, owing to its intermittent delivery and considerable oscillation, it necessitates special arrangememts for feeding to and accommodation of sluice-boxes. There are many situations where a tough clay, containing much buried timber, and requiring puddling, could be particularly well dealt with by this machine. They have been very successfully used in gold mining in New Zealand and S. America. Various sizes are made—from 3 to 40 cwt. They are practically steam shovels on barges, and can be used equally well mounted on rails.

A much heavier type, up to 5 t., is used in river phosphate mining in Florida. The 8 claws are closed by steel springs having a tension of 14,000 lb., and they surround a central heavy steel drop-chisel for breaking the ground. They work in water up to 50 ft. deep.

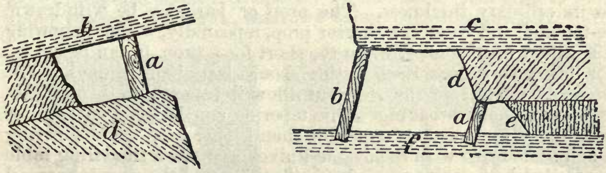
Dipper Dredges.—These are spoon-shaped shovels, dependent from a long boom or derrick. They are useful in river prospecting, but have not come into use for extensive operations, though their lifting capacity has been brought up to 100 t. a day in river phosphate mining in Florida.

BEDS.

Where the mineral deposit occurs in beds or seams lying almost horizontally, often under a heavy overburden, the engineer's chief concern is to provide support for the roof and to facilitate the loading of trucks. The broken mineral cannot gravitate to any point for automatic filling, and, in some cases, suffers deterioration in market value by handling. Various ingenious methods adapted to peculiar circumstances will be described in this section, as well as the conventional long-wall and pillar-and-stall systems associated with coal-mining.

Coal.—The most familiar example of this branch of mining is coal. The low selling-price and fragile nature of the product necessitate simple and inexpensive measures.

Supporting.—By far the most common way of sustaining the roof of comparatively flat beds is by placing posts, or "props," as they are more generally called, in a vertical position between floor and roof, sometimes with a flat slab, called "head-board" or "cap-



FIGS. 122, 123.—WOODEN PROPS.

piece," at top, so as to distribute the load as much as possible. These props are simply sections of straight-grown trees, with the bark removed, and varying in length and diameter acording to thickness of vein and pressure to be resisted. Figs. 122–124 illustrate their application. In Fig. 122, *a* is the prop, *b* roof, *c* rock overlying coal *d*; in Fig. 123, coal *e* is cut away in advance, prop *a* supporting a bed of oil shale *d*, and prop *b* holding sandstone roof *c*, *f* being hard floor; and in Fig. 124, prop *a* with cap sustains top coal *e*, while a "knee-joint" or "cockermeg" *d* temporarily prevents coal *a* from falling while it is being undercut, floor *b* being firm.

In a large coal-mine, the consumption of props is excessive—several hundred a day—so that any means of economising them is worth consideration. W. H. Hepplewhite avoids the breaking of props by rendering their ends the weakest part, so that the rupture shall take place by a gradual process of spreading or "burring," whereby the length of the prop is reduced in accordance with the depression of the roof, and its main part continues to act as a support throughout. The prop is tapered, according to its size and the pressure and average depression of the roof, care being taken to provide plenty of taper for the time it has to last in one position. For instance, a prop 5 ft. long and 6 in. diam. at its thin end is given a taper of 11 in. with a point 3 in. diam. to meet a roof depression of 6 in. about 6 ft. from the face. It is intended for use chiefly with hard floors. Bars or stretchers for supporting the walls may also be tapered, preferably at both ends, so that, instead of breaking across the middle, the tapered portions, being relatively weaker, can "burr" or "fuzz" until the wood has shortened nearly to its ordinary thickness. The prop or bar may be withdrawn, re-tapered, and reset as a shorter prop, repeatedly, without showing a splinter, until finally, when too short for a prop, it can be cut up for lids or sleepers. Each setting should last about 14 days. The props do not pierce the roof, but allow it to settle to its own subsidence without breaking. The tapering can be done by hand or machinery, and need not cost more than ½*d.* per prop. The tapering machine is fitted with removable knives, and has a travelling table controlled by a rack and pinion. To this table the prop is secured by a clamp with right- and left-hand screws. Each machine can taper 30 props per hour, and requires about 4 h.p. Tapered props will last 6 weeks or more, as compared with 2 weeks for ordinary buckled props, and 4 weeks for sheared props. The actual saving in cost exceeds 50 %. With a tender roof and very hard floor, pointed steel props have been successfully applied on similar principles, the pointed end penetrating, and thus affording gradual relief.

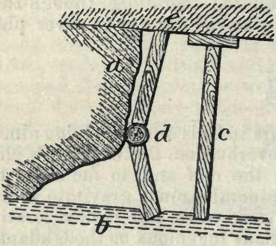


FIG. 124.—WOOD PROP.

At Emly-Moor collieries, H. Baddeley has adopted, with great success, a system of regular timbering (as distinguished from the casual propping usually followed), which is likely to be widely copied where roofs are bad. In machine-cut faces (Fig. 125), two rows of props *a b* are left along the face after the coal has been hewn away (A); a third row *c*, is set behind the machine as soon as the

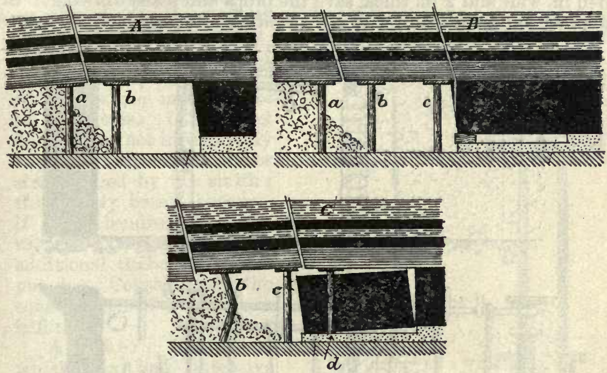


FIG. 125.—TIMBERING AT EMLY-MOOR COLLIERY.

coal has been under-cut (B); and as the coal is filled away, the filler sets another row *d*, about 2 ft. from the last (C). The two back rows *a b* are then drawn out, leaving only the two rows *c d* next to the face about 2 ft. apart. The distance from prop to prop along the face is 2–2½ ft., the rows being perfectly straight. The

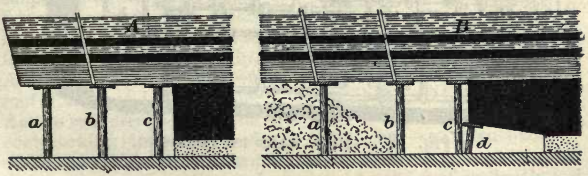


FIG. 126.—TIMBERING AT EMLY-MOOR COLLIERY.

roof now breaks off in a straight line, close behind the back row of props, and there is no trouble with the face falling in. For hand-cutting (Fig. 126) a rows of props *c* 5 ft. apart is set along the face, close to the coal, before the coal is holed (A), and two intermediate props are set between them after the holing is done and before the sprags d are drawn (B). In a seam $1\frac{1}{2}$–2 ft. thick, with softer bind in the roof, the coal is packed solid with holing and ripping dirt, only one row of props is left to support the roof after the coal is gone, but another row is set as the holing proceeds: rows are $3\frac{1}{2}$ ft. apart, and props 3 ft. Caps or "lids" 12–18 in. long, 3–6 in. wide and 2 in. thick, save the props and assist drawing. In another hand-cut seam, with roof of rather soft grey bind, two rows

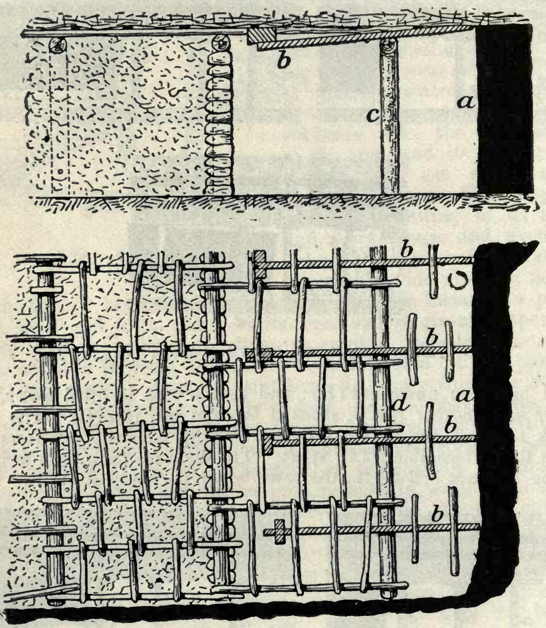


FIG. 127.—IRON BARS IN PROPPING.

of props are set along the face, 3 ft. apart and $2\frac{1}{2}$ ft. prop to prop. A third row is set close to the coal, about 5 ft. prop to prop, before the men begin to hole; when holing is done, intermediate props are set. Hard-wood "lids" are used in this seam, 18–24 in. long, 6 in. wide and $1\frac{1}{2}$ in. thick, and they hold well.

A remedy for falls of roof, which has proved most efficacious at Courrières, mainly consists in supplementing the props and caps by wrought-iron bars, as shown in Fig. 127: *a*, face of coal; *b*, iron bars; *c*, props; *d*, caps. Each miner at the face is furnished with 3 bars, 51 in. long by 1⅝ in. square, with which he is compelled to form a sort of temporary shield in advance of the last row of timber props. When another row of props has been put in, the bars are withdrawn, and then driven on in advance beyond the new set of supports. They are placed 15–20 in. apart, and are fixed securely by wedges. As the work proceeds, the temporary protecting shield is pushed on; it takes very few miuutes to knock out the wedges, drive the bars forward, and wedge them up again. About 6000 bars are in daily use at Courrières. If they get bent, which not infrequently happens, they can easily be straightened by the smith; they rarely break, because, if great bending indicates unusual pressure of roof, additional timber props are put in. Consequently the consumption of iron bars is trifling.

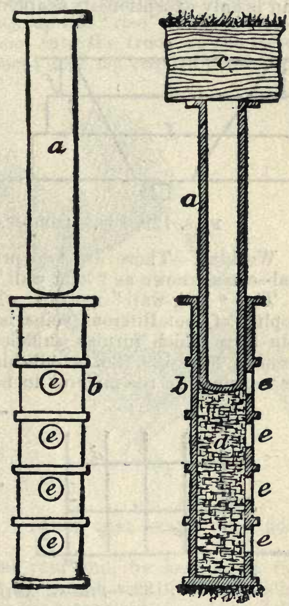
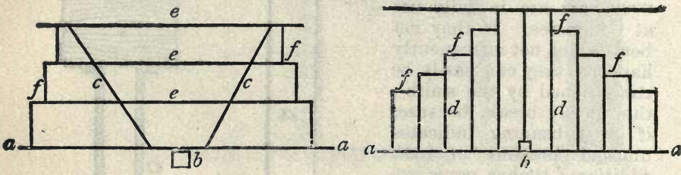


FIG. 128.—BALMER PROP.

Another innovation, which has had extensive trial with gratifying results, aims at replacing timber props entirely by tubular iron. The Balmer "prop" (Fig. 128) consists of two cast-iron cylinders, one of which, *a*, is arranged to telescope inside the other, *b*; the latter is packed with small coal, so as to support the former at any desired height. A cap *c* is placed between roof of seam and top of prop. As the roof creeps down, the coal-packing *d* is compressed and yields slightly, allowing the props to support the roof evenly. As the roof gradually lowers further, the props may be eased by withdrawing some of the coal-packing through one of holes *e* (preferably that next to lower end of *a*). The prop can be readily withdrawn by easing the ram still further in the same way. The following advantages are claimed —(1) They will last many years (practically indestructible), and may be used over and over again; (2) simple in working, and liked by workmen; (3) unaffected by water or atmospheric conditions, which often cause wooden props to rot rapidly; (4) may be used as pillars, shores, or struts in any position, and in connection with girders; (5) during time that roof lowers or floor rises, say 12 in., 6 to 8 wooden props would have to be set to replace buckled or broken timber.

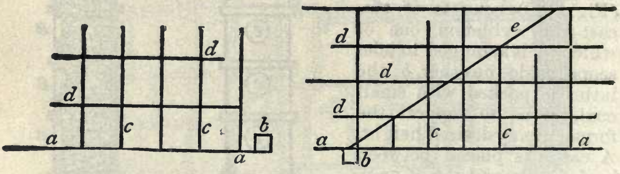
Occasionally use may be made of a crib or pen, as already described on p. 296, waste being disposed within it; or the big pieces of waste may be stacked without any cribbing; but this plan is only exceptionally adapted to coal mining.



FIGS. 129, 130.—LONG-WALL SYSTEM OF MINING.

Working.—There are two principal methods of working flat coal-seams, known as "long-wall" and "pillar and stall."

The "long-wall" method, which closely resembles overhead stoping of metalliferous veins, is applicable to nearly horizontal thin beds which furnish sufficient waste for filling. The main tunnel *a*, Figs. 129, 130, is built high enough to accommodate trucks for transporting the mineral to hoisting shaft *b*, and is driven at



FIGS. 131, 132.—PILLAR AND STALL SYSTEM OF MINING.

the lowest part of the bed, so as to form a natural drainage for mine water. From it, drifts are run into the mineral, either diagonally as at *c*, or transversely as at *d*, and are connected by parallel levels *e*. The direction of the drifts is governed by the rate of dip of the bed, the object being to secure a suitable incline for the trucks which carry out the mineral. The workings are connected all round by cross-cuts as at *f*, to complete the circulation of air for ventilation.

The "pillar and stall" system is adopted where beds are thicker, and do not afford sufficient waste for filling, so that pillars of mineral have to be left standing as a support for the roof. The main tunnel *a*, Figs. 131, 132, is driven as before from shaft *b*, and from it are run drifts *c* at intervals, and, crossing these again, levels *d* parallel with *a*, and occasionally diagonal drifts *e*. The bed is thus divided into regular blocks, portions of which, varying in size, are left to form pillars that support the roof, these being finally withdrawn as far as safety will allow.

In seams having a dip of 40–60°, it is customary to drive the stalls square off from the gangway, up the "rise" of the seam, and to have the coal run down the shoot into the truck at the bottom of it; with this rate of dip, the shoot does not require planking at

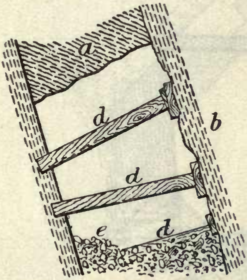


FIG. 133.
STEEP COAL SEAM: BAD ROOF.

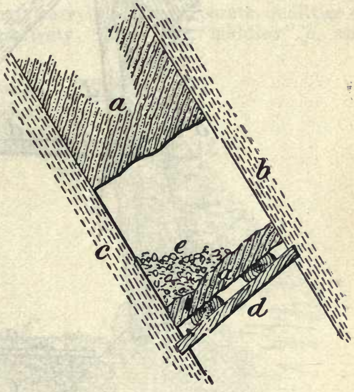


FIG. 134.
STEEP COAL SEAM: GOOD ROOF.

side or bottom to make the coal run, and, by keeping it full, except 3–4 ft. of working room at the breast, very little coal is lost by pulverisation in its descent, as the movement consists in a slow settling in proportion as the coal is allowed to run into the trucks at bottom.

In seams of 30–40° dip, miners are compelled to plank the sides of the shoot to some extent, in order to enable the coal to slide down without assistance. In seams of 25–30°, coal will not descend unless the sides of the shoot are partly planked, and the bottom is covered with sheet iron. In working seams having a dip of 10° or under, stalls are driven diagonally to the direction of the gangway, unless the rate of dip is less than 4°.

The trucks or cars used in Europe are, in nearly every case, smaller than American; the reason, in most cases, is an effort to reduce the enormous first-cost of deep shafts, by having small shaft area, thus leaving but small space for mine cars or cages and pumpway; small cars also suit the large number of boys employed in European mines.

When coal seams lie at a steep angle, extraction of the mineral follows closely on the lines of metalliferous vein mining. Thus,

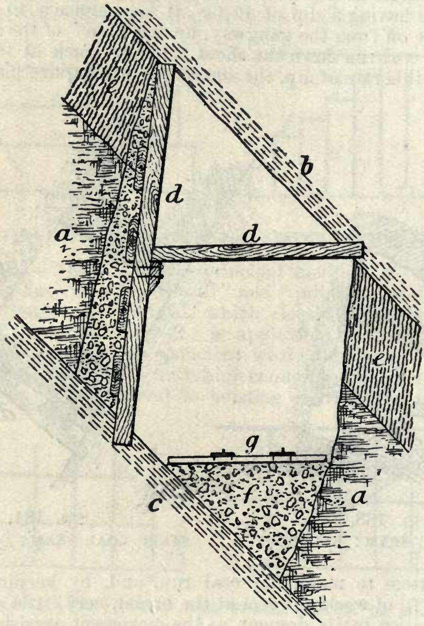


FIG. 135.—TIMBERING ROADWAY IN COAL

in the very steep seam shown in Fig. 133, coal *a* is over 4 ft. thick; roof *b* is full of joints and very treacherous, necessitating the use of many timbers *d*; floor *c* is hard and sound. The waste *e* from floor and roof is allowed to fall and accumulate under the feet of the miners, and by it the timbers are gradually buried.

Fig. 134 delineates a similar steep bed, but in this case the roof is very good, and no timbering is necessary. Both roof *b* and floor *c*, however, are lined with friable rock, some inches in thickness, which falls with coal *a*, and is allowed to collect as at *e*, forming a platform for the miners. The seam is worked in sections, separated by stout timbering *d*.

Fig. 135 illustrates the mode of timbering a roadway. Coal seam *a* is surmounted by a considerable thickness of weak shale *e*, which does not long survive removal of subjacent coal, and is quite distinct from firm and reliable sandstone roof *b*. The practice is, therefore, to let this shale break down and accumulate, as at *f*, on the floor proper *c*, and to lay the tramway *g* upon it. Falls from upper part of seam into roadway are prevented by a lining of posts and slabs *d*, well secured in roof and floor, and further strengthened by cross-posts.

Fig. 136 represents a seam carrying three separate qualities or kinds of coal, called respectively "tops" *a*, "middles" *b*, and

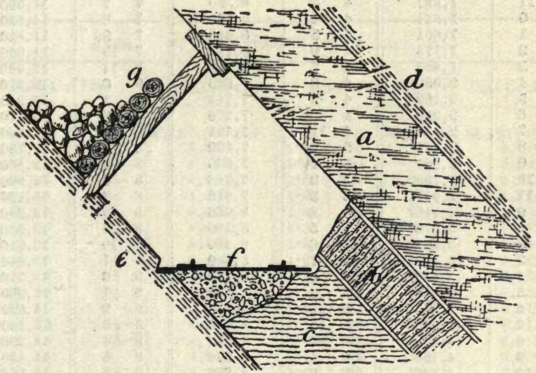


FIG. 136.—SEAM WITH THREE KINDS OF COAL.

"bottoms" *c*. Roof *d* and floor *e* are very sound and firm. Owing to the top coal being very strong, it is possible to work the middles and bottoms separately, leaving the tops for a distinct operation. Tramway *f* is laid on waste, and at *g* is shown a staging of slab-posts, on which coal is collected, and from which loading is done in safety.

Contents of Coal Seams.—A rough and ready way of calculating available contents of coal in a given area of seam is to take a basis of 1 acre 1 in. thick = 100 tons. A more correct way is to ascertain the sp. gr. and multiply by that. In round figures, the sp. gr. taken to two places of decimals will represent the weight of 1 acre

1 in. thick if the decimal point be struck out: thus sp. gr. 1·10 = 110 tons per acre 1 in. thick.

Weight of Coal under 1 *acre of land*, at 19 cwt. per cub. yd., or 78¾ lb. per cub. ft., approximate, and assuming seam to be level.

ft.	in.	tons.	ft.	in.	tons.	ft.	in.	tons.
0	1	127	3	5	5,212	6	9	10,260
0	2	254	3	6	5,346	6	10	10,390
0	3	381	3	7	5,468	6	11	10,520
0	4	508	3	8	5,596	7	0	10,650
0	5	635	3	9	5,724	7	1	10,780
0	6	762	3	10	5,852	7	2	10,910
0	7	889	3	11	5,980	7	3	11,040
0	8	1,016	4	0	6,096	7	4	11,170
0	9	1,143	4	1	6,124	7	5	11,300
0	10	1,270	4	2	6,252	7	6	11,430
0	11	1,397	4	3	6,380	7	7	11,560
1	0	1,524	4	4	6,508	7	8	11,690
1	1	1,651	4	5	6,636	7	9	11,820
1	2	1,778	4	6	6,764	7	10	11,950
1	3	1,905	4	7	6,892	7	11	12,080
1	4	2,032	4	8	7,020	8	0	12,210
1	5	2,159	4	9	7,148	8	1	12,340
1	6	2,296	4	10	7,276	8	2	12,470
1	7	2,417	4	11	7,404	8	3	12,600
1	8	2,534	5	0	7,532	8	4	12,730
1	9	2,661	5	1	7,660	8	5	12,860
1	10	2,790	5	2	7,788	8	6	12,990
1	11	2,917	5	3	7,916	8	7	13,120
2	0	3,048	5	4	8,046	8	8	13,250
2	1	3,176	5	5	8,180	8	9	13,380
2	2	3,304	5	6	8,310	8	10	13,510
2	3	3,432	5	7	8,440	8	11	13,640
2	4	3,560	5	8	8,570	9	0	13,770
2	5	3,687	5	9	8,700	9	1	13,900
2	6	3,815	5	10	8,830	9	2	14,030
2	7	3,943	5	11	8,960	9	3	14,160
2	8	4,070	6	0	9,090	9	4	14,290
2	9	4,198	6	1	9,220	9	5	14,420
2	10	4,326	6	2	9,350	9	6	14,550
2	11	4,454	6	3	9,480	9	7	14,680
3	0	4,576	6	4	9,610	9	8	14,810
3	1	4,700	6	5	9,740	9	9	14,940
3	2	4,828	6	6	9,870	9	10	15,070
3	3	4,956	6	7	10,000	9	11	15,200
3	4	5,084	6	8	10,130	10	0	15,330

Coal Heaps.—To estimate contents of coal heaps, measure the cubic space occupied and multiply the result, stated in cub. yd., by 6; the product = contents in tons.

Coal Seams.—Sp. gr.; weight per acre, per cub. ft., per cub. yd., and per cub. ft. when broken.

Sp. gr.	Natural Weight per acre 1 in. thick.	Natural Weight per cub. ft.	Natural Weight per cub. yd.	Broken Weight per cub. ft.	
				Large.	Small.
	tons	lb.	tons	lb.	lb.
1·10	111½	68¾	·829	42½	37
1·15	116½	72	·867	44½	38¾
1·20	121½	75	·904	46½	40½
1·25	126½	78	·941	48½	42
1·30	131½	81¼	·978	50⅓	43¾
1·35	136¾	84¼	1·016	52½	45¼
1·40	141¾	87½	1·054	54¼	47¼
1·45	146¾	90½	1·091	56	49
1·50	152	93¾	1·129	58	50½
1·55	156¾	96¾	1·167	60	52¼
1·60	162	100	1·200	62	54

Area of Coal Seams when inclined.

Amount of Dip.			Coal Contents.	Amount of Dip.			Coal Contents.
1 in	deg.	in. per yd.	sq. yd. per acre.	1 in	deg.	in. per yd.	sq. yd. per acre.
57¼	1	½	4,841	1·66	31	21½	5,646½
28½	2	1¼	4,843	1·60	32	22½	5,707¼
19	3	1¾	4,846½	1·54	33	23¼	5,771
14¼	4	2½	4,852	1·48	34	24¼	5,838
11½	5	3	4,858½	1·42	35	25¼	5,908½
9¾	6	3¾	4,866½	1·37	36	26	5,982½
8	7	4½	4,876	1·32	37	27	6,060¼
7	8	5	4,887½	1·28	38	28	6,142
6¼	9	5¾	4,900	1·23	39	29	6,228
5½	10	6½	4,914½	1·19	40	30¼	6,318¼
5	11	7	4,930½	1·15	41	31¼	6,413
4¾	12	7½	4,948	1·11	42	32½	6,513
4¼	13	8¼	4,967	1·07	43	33½	6,618
4	14	9	4,988	1·03	44	34¾	6,728¼
3¾	15	9½	5,011	1·00	45	36	6,844¾
3½	16	10¼	5,035	·96	46	37¼	6,967½
3¼	17	11	5,061	·93	47	38½	7,096¾
$3\frac{1}{12}$	18	11¾	5,089	·90	48	40	7,233¼
2·90	19	12¼	5,119	·87	49	41½	7,377¼
2·74	20	13	5,151	·84	50	43	7,529½
2·60	21	13¾	5,184	·81	51	44½	7,691
2·47	22	14½	5,220	·77	52	46	7,861¼
2·35	23	15¼	5,258	·73	53	47¾	8,040
2·24	24	16	5,298	·70	54	49½	8,217½
2·14	25	16¾	5,340	·67	55	51½	8,419¾
2·05	26	17½	5,385	·65	56	53¼	8,639¼
1·96	27	18¼	5,432	·63	57	55½	8,873
1·88	28	19	5,481½	·62	58	57½	9,139
1·80	29	20	5,534	·60	59	60	9,430
1·73	30	20¾	5,589	·58	60	62½	9,759

The contents in cub. yd. may be deduced by multiplying the figure in the fourth column by thickness in yd. or fraction of a yd.

Ex. 1. A seam 1 ft. 6 in. thick lying at an inclination of $16\frac{3}{4}$ in. per yd. will contain 5340 sq. yd. $\times$ ·5 yd. = 2670 cub. yd.

Ex. 2. A seam 1 fathom thick lying at 39° will contain 6228 sq. yd. $\times$ 2 yd. = 12,456 cub. yd.

Coal-cutting Machines.—The object of coal-cutting machines is to cut a groove or channel in the coal, creating a clearance which allows the coal to be broken down (or up), by explosives or by wedging, without an undue amount of small coal being produced. If the groove is parallel to the coal seam, it is called cutting, holing, or kirving; if perpendicular to the seam, shearing or nicking the coal.

Practically only two types of coal cutter are favoured in England —longwall disc, and longwall bar—although some longwall chain, and heading and percussive machines, are also used. The last-named, however, differ from the percussive machine largely adopted in America. The type seen in England is fixed to a rigid vertical post, which takes the shock of the blow. The machine can be moved up and down this post, and can cut in a vertical or in a horizontal plane. As the cut deepens, lengthening-pieces are put in between the piston-rod and cutting-tool, or bit. This bit is something like a crown wheel, whereas the bit of the American machines is fish-tail in form. The vertical post is fixed between roof and floor, either by a screw or by a small hydraulic ram in the column. Only a very strong roof will stand the thrust of the post and the blows of the machine; but given such a roof, the machine is very useful.

The heading machine used in England, and to a slight extent in America, has one or two rotating shafts perpendicular to the coal face to be cut. A sort of cross-head is made to rotate instead of having a reciprocal motion. The cutters are arranged on this cross-head in one of two methods—(*a*) they are fixed across it so that the whole of the coal is bored out of the heading and turned into slack, which is automatically loaded on to a following tub by an archimedean screw; or (*b*) there are only two cutters, but each is 2 ft. long, and fixed at right angles to, and at the end of, the cross-head. When the shaft rotates, the cutters make an annular cut, and a core of coal is left, to be removed by hand when the machine is stopped. A third form of this machine has two parallel rotating shafts, each fitted as described, and geared so that the cross-heads are at right-angles to, and thus clear, each other.

The disc machine consists of a rectangular steel frame, mounted on small flanged wheels, for running on rails. On this frame are mounted a pair of cylinders, which, by a crank-shaft and intermediate shaft carrying a spur-wheel, drive the cutter disc. The wheel is about 5 ft. diam., and the cutters are fixed into its circumference. Just inside the rim of the wheel is a broad circular rack, the teeth running radially. The disc is supported by a broad plate bracket, which is fixed to one side of the rectangular frame. The spur wheel on the intermediate shaft gears with the circular rack of the disc, and drives the latter at 15–50 rev. per min. The machine undercuts 5 ft., and is drawn along the face by a steel rope. Some machines are arranged to cut in either direction, and some are fitted with electric motors instead of pneumatic engines. It takes 20–30 min. to change all the cutters, and 3 or 4 changes are usually necessary during each 8-hr. shift. The cutters or bits of disc and chain machines have chisel edges; those of bar machines have sharp points set at an angle to the shank. Disc machines cannot make a sumping cut; at starting, a place for them has to be holed by hand.

Rotary bar machines consist of a strong tapered bar about 7 ft. long, making a cut 9 in. high at the face and 4 in. at the back. Into this bar are fixed a number of sharp-pointed cutters. The sockets for the cutters are arranged in the form of a helix, so as to cause most of the cuttings to be brought out. The bar can be swung round till it is parallel to the face, or at right angles to it. It can also be slewed slightly in a vertical plane, to go over or under a sulphur ball, or other obstruction. The machine hauls itself along the face by a wire rope, like the disc machine. To start a cut, the cutter bar points straight back; the machine is then started, and the bar is gradually rotated into the coal until at right angles to the face. The bar makes about 400 rev. per min. Bar machines are electrically driven.

Everything allowed for, there is a net saving of 6*d.* per ton in favour of machines, the average cost of labour per ton of machine-cut coal being 2*s.* 3½*d.* The average increase of output per man employed has amounted to 65%. Machines produce on the average 12½% more round coal—i.e. where lump coal got by hand is 60%, machines give 72½%. The cost of a complete plant for, say, 10 disc machines averages 1000*l.* per machine, whether electricity or compressed air be used, though the latter is slightly the cheaper. Individual electric machines cost about 400*l.* each; pneumatic, about 250*l.* each.

There are three types of American machine—pneumatic percussive; chain-breast, driven either by compressed air or electricity; and chain-cutter longwall, driven by electricity.

Pneumatic percussive machines are very similar in principle to a steam hammer, or pneumatic chipping or riveting tool; there is a double-acting cylinder, with piston and heavy piston-rod, at the outer end of which is fixed a bit which cuts the coal. The cylinder has trunnions carrying two wheels 16 in. diam. At the back of the cylinder are two handles for controlling the machine. The dimensions over all are about 6 ft. 9 in. long, 2 ft. 3 in. wide, 1 ft. 9 in. high; weight, about 750 lb. These machines are used on a board 9 × 3 ft., propped up 18 in. at the end away from the face.

One man runs the machine; another clears away the cuttings. The rate of undercutting varies greatly with the quality of coal; 33, 60, 152, and 350 sq. ft. of undercut per hr. were actual results under different conditions. The undercut or holing is very similar in shape to that which obtains with hand work, the height being about 15 in. at the face and 3 in. at the back. The advantage of this shape of undercut is that the coal rolls out well when shot down, whereas, with a chain cutter, which produces a parallel cut about 4 in. high, much harder shots have to be used in order to get the coal out, and this harder firing frequently damages the roof. The skill required to work these machines is considerable; dust and noise are great, but the discharge of fresh air at the face is beneficial, and has a tendency to reduce the temperature.

Chain-breast machines are very different in character, and in great variety. The general design consists mainly of a bed-frame with prime mover attached, and a triangular sliding frame with sprocket driving-wheel at the apex and two idle pulleys at the base angles. The chain with cutters attached fits tightly round these three wheels, the base of the triangle being presented to the face. No rails are needed; they are simply put on skids of timber 3 in. square with half-round iron straps fixed lengthways. Two jacks are used to hold the machine up to the face; a long one coming from the back of the machine to the roof, and a short one running diagonally from the front of the machine to the face, so as to take the thrust due to the reaction of the cutters on the coal. The machine having been fixed in position and started, the sliding frame moves forward perpendicularly to the face, and makes a cut 44 in. wide, 5 ft. 6 in. deep, and 4 in. high. The sliding frame is then run back, the machine is moved about 44 in. to one side, and jacked up; then another cut is taken. The shifting occupies 2 min. The prime mover is either an electric direct-current motor or a pneumatic engine. Dimensions over all are about 11 ft. long, 3 ft. 8 in. wide, 2 ft. 6 in. high; weight, about 3000 lb.

Longwall chain-cutters show great variation. Two leading features are that they employ chain cutters and are driven by electric energy. With this machine, it is sometimes necessary to undercut by hand in the first instance, so as to get it in the right position for travelling along the face; in other cases, the machine makes its own sumping cut, after which, by a slight alteration of the feed chain, it hauls itself along the face, and makes the ordinary longwall cut. Some run on one rail, some on two, and some need no rail. Dimensions over all are about 10 ft. long, 3 ft. wide, 2 ft. 9 in. high; weight, about 3000 lb.

The seams worked are decidedly thick, rarely less than 5 ft. nor more than 12 ft.; hardness varies greatly; depth from surface is never very great, 320 ft. being the maximum; roofs remarkably good, and frequently need scarcely any timbering, or at most a single line of props down the centre of the room. (Ackerman.)

Electrically-driven coal-cutters are favoured in mines which are free from gas. Though no percussive machine has given satisfaction, there are several successful forms with rotary cutters. Continuous-current motors have been mostly used, but alternate current is employed in some cases. An average cutter consumes about 12 h.p. when at work, but the lowest power motor that should ever be employed is one capable of developing 20 h.p. It may safely be said that nine-tenths of the difficulties met with in electric coal-cutting machines are due to insufficient motor power. Their employment in fiery mines is not advisable. There is practically no danger attending the use of a motor; a polyphase induction motor is absolutely sparkless, and even a direct-current motor can be so enclosed as to be quite safe. The danger lies in the leads. At present there is no known method by which sparking can be prevented if the conductors are suddenly severed, especially in a circuit with as much self-induction as must exist in a motor circuit. The higher the voltage used, the greater is the danger incurred. (Prof. H. Louis, Min. Jl.)

Filling.—The filling up of ground from which coal has been extracted, except in the degree provided for by waste broken in the mine itself, is rarely done. Yet it would permit the working of some shallow seams which at present cannot be availed of owing to consequent subsidence of the surface—under townships, railways, canals, etc.; and would facilitate working seams in chosen order, minimise fires and "damp," and improve ventilation. Where conditions favoured, it might be done automatically, at very small cost. Hand-filling is carried out in some cases, costing 1*s*.–1*s*. 6*d*. per ton of packing material, but this only fills up about 60 % of the space, even by most careful packing. The best material is that which will lie closest, viz. sand, and without a due proportion of very fine matters to occupy interstices, filling cannot be really close.

At Lens (Belgium), use is made of shales from the coal washers, which are sieved into two sizes, ·16–·4 in. and ·4–1 in., and then mixed together in varying proportions. This material is taken underground in ordinary steel tubs (of ½ t. coal capacity) ready for distribution, which last is accomplished hydraulically, the water draining off to be pumped to surface, while the filling packs solidly up to the roof. The mixing and filling processes are quite elaborate. The tubs of shale are tipped into a disused inclined plane, sheet-steel floored at an angle of 40°, and controlled by a door, which delivers at will into a large hopper which is also supplied with water. The mixture of shale and water, in a semi-fluid state, is piped (6-in.) to the worked-out ground, which is "paddocked" as it were by planking and brattice-cloth, so that the water drains off, and the packing consolidates. Each day some lower planks can be removed, the packing standing firm when dry. The cost is said to be about 4*d*. per cub. yd. Obviously, it could not be done so cheaply or well in flat seams.

At Ferdinand (Silesia), sand is used in the same way in 7¾-in. pipes; at Myslowitz, similar sand filling costs 6*d*. per cub. yd.; at Concordia, hydraulicing sand direct from a pit, 5*d*.: all these being thick seams at shallow depths. In Westphalia, clinker is employed, costing 1*s*. 11*d*. for hydraulic filling, as against 2*s*. 3½*d*. for hand labour.

Iron.—The vast opencast iron mines of the United States have created their own systems for dealing with colossal tonnages at low costs. Two general methods of open pit-work are adopted—(*a*) to strip a comparatively large area, and, after loosening the ore by blasting, to load it by steam shovels into railway cars alongside; (*b*) the ore is stripped of its surface covering of earth and boulders, loosened by hand or powder, and "milled" down through raises or passes connecting with drifts or levels established 50–60 ft. below the top of the ore, or, in shallow deposits, along their floors; from the raises, the ore is loaded into cars, trammed to shafts, dumped into skips, and hoisted to surface. The first method should be limited to large ore-bodies with approaches not exceeding 3 % grade, as long, deep railway cuttings are expensive, especially if not over ore; still, seven times as much ore is mined on the Mesabi range with steam shovels as by "milling."

As to relative costs, steam-shovel contracts do not exceed 15*d*. per cub. yd. (about 1¼ short ton), and sometimes total costs run as low as 9–10*d*. The ratio of overburden to ore permissible in stripping has advanced from 1 to 1 to about 2 to 1, consequent on increased prices of timber and timbering, and labour conditions. In "milling," costs in average cases will exceed steam-shovel costs by 5–7½*d*. per cub. yd.

In milling, raises are usually placed about 40 ft. apart in the drifts, and are properly timbered to serve as shoots or passes, with pockets and spouts at their base, through which the ore is drawn into tram-cars. When these are mined down, a second series is put up to handle the pyramid of ore left between the first. Loosening ore is usually done by powder: holes are drilled around the sides of the craters by driving down pointed steel rods; the holes are chambered by several sticks of dynamite, and filled with 200–300 lb. powder. Much of the ore is broken directly into shoots when blasted, and the cheapest labour serves to drop the rest; none is lifted by hand. In both steam-shovelling and milling, the loss of ore is practically nil.

The Cleveland iron-ore deposits are mined by a system of temporary pillars. Details and dimensions vary somewhat, but the following is typical. From the bottom of the shaft a "mainway" 5 yd. wide is driven out with only a slight incline. At intervals of 20 yd., drivages (also 5 yd. wide) are put out at right angles to the mainway, known as "bords": and every 30 yd. along the bords, cross drivages are made (4 yd. wide), known as "walls." Thus the bed is cut up into a series of pillars 30 × 20 yd., while considerable ore has been removed in making the galleries. The pillars are now attacked, beginning with those near the boundary and working towards the shaft. A "drift" of 2–4 yd. in width is worked across the pillar from the bord, cutting off one-third. From the drift, this rectangular portion is removed by drivages known as "lifts," two or three in number. While these lifts are being worked away, another drift is driven across the remaining two-thirds of the pillar, in preparation for another set of lifts. Sometimes a corner or portion of the side of a pillar is left standing during working, to keep out the fallen rubbish beyond, or prevent a too sudden fall of roof. During removal of the ironstone, the working-place is timbered, but this is subsequently taken away, and the roof is allowed to fall in. Rotary drills (hand and electrical) are used, and 2-oz. pellets of black powder, giving about 2 t. per hole, and costing $1\frac{1}{2}d.$ per ton of ore.

Petroleum.—In the Far East, where machinery is uncommon, and human labour is abundant and cheap, petroleum-bearing beds are attacked by wells or pits rather than by boring, and, odd as it may seem, the native method is often successful where the introduced method (which means also introduced white labour and its attendant troubles) signally fails. Thus, the Japanese sink wells about $3\frac{1}{2}$ ft. square to a depth of nearly 1000 ft., and could go much deeper if furnished with efficient artificial light. Lyman pronounces the native method superior to drilling with steam-power plant, owing to the great costliness of machinery, heavy fuel bill, and difficulty of transport. A native well 900 ft. deep costs but 200*l.*, or little more than a third of drilling costs; besides which, a dug well can be entered for cleaning or repairing, and it exposes a much larger percolating surface, which may be enormously added to by extending drifts into the oil-yielding stratum. In Burma, the native wells are about 60 ft. deep and 5 ft. square, lined with slabs. The oil is raised in a bucket attached to one end of a rope running over a wheel fixed above the mouth of the well. The other end of the rope is fastened to the waist of a man or woman, who generally has two or more boys or girls to assist in pulling. As soon as the bucket fills, these persons run down a beaten path, and the bucket is thus drawn to the mouth of the well, when it is emptied by another person. The output is certainly very limited, but the working costs are absurdly small.

In European and American practice, recourse is always had to well-sinking or boring machinery—a development of that described on pp. 158–64. The first step is to sink a "conductor" through the surface ground to "bedrock." When the overburden is not more than 10–15 ft. thick, a common well-shaft, 8–10 ft. square, is dug to rock. A wooden conductor, 8 in. square in the clear, is then set up perpendicularly between the rock and the boarded floor of the derrick, the junction between rock and conductor being so made as to keep out gravel and mud. When the depth is too great to admit of digging down to rock, "driving-pipe" is forced down by means of a "mall," working in guides, as in pile-driving, When 200–300 ft. have to be thus driven, as is sometimes the case. a good deal of skill is required. If bedrock is reached in less than 60 ft. from surface, drilling is commenced by "spudding," which consists in alternately raising and dropping the tools by tightening and slackening the cable, which is simply coiled 2 or 3 times round the bull-wheel.

The modern well has an 8-in. wrought-iron drive-pipe, armed at bottom with a steel shoe. This is driven down to bedrock, and an 8-in. (really 7⅝-in.) hole is drilled in water-bearing strata. At this point, the bore is gradually reduced to 5½-in., and there a bevelled shoulder is made; 5½-in. casing, provided at the lower end with a collar to fit the bevelled shoulder, is then inserted, and a sufficiently water-tight joint is thus made. Drilling with 5½-in. bits is then continued until the required depth has been reached. When gas is obtained in quantity to furnish fuel for the boiler, it is conveyed through a 2-in. pipe connected with the casing beneath the derrick floor, and passing into the door of the furnace. A ¼-in. steam-pipe, fitted with elbow and ⅛-in. jet, is inserted in the gas-pipe, close to the fire-box, and a blast of steam is thus caused to issue with the gas. The apparatus acts as an exhauster, drawing the gas from the well, and preventing the flame from running back.

The "water-packer" prevents water that may pass into a well below the casing from gaining access to the oil-sand, and stops the ascent of gas outside the tubing. It is applied round the tubing at any desired point, and its effect is to shut off all communication between the annular space outside the tubing above it and the oil chamber below. The oil and gas are thus confined in the well chamber, and many wells are thus caused to flow that would otherwise require pumping. Under these circumstances, the flow is intermittent, taking place when sufficient gas-pressure has accumulated. There are many forms of water-packer, but one of the simplest consists of a band of rubber which, on compression, is forced against the walls of the well.

If the well does not flow, the oil may be pumped. The working barrel of the pump is placed at the bottom of the well on the end of the tubing, a perforated piece of casing of proper length (the "anchor") being attached to the lower end of the working barrel. To the sucker of the pump are attached the required number of wooden sucker-rods, screwed together, the upper end of the string of rods being connected with the walking-beam. There are valves at the bottom of the working barrel, and in the sucker. The sucker is provided with a series of 3 or 4 leather cups, which are pressed against the working barrel by the weight of the column of oil. Sucker rods are 1½ in. diam. × 24–28 ft. long. When a number of contiguous wells are to be pumped, a "grasshopper" enables it to be done by a single walking-beam.

Most petroleum wells in the United States are "torpedoed" on the completion of the drilling, in order to increase the flow of oil. The torpedo is a charge of nitro-glycerine in a suitable shell, which is lowered to the oil-bearing rock, and there exploded, with the effect of opening fissures into the surrounding rock.

Compressed air at 250–300 lb. per sq. in. is sometimes used for forcing mineral oils to the surface, by medium of a 2-in. air-pipe inside a 5-in. casing. Small wells (producing 2–3 bbl. daily) are blown once per 24 hr.; 20–30 bbl. wells, about 6 times.

An average well costs about 600*l*.

Phosphates.—Much of the phosphate raised in Florida is got by river dredging, and the whole phosphate district is low-lying flat ground, with overburden up to 6 ft. thick. Where possible, the following system of mining is adopted: A main tram-line leading from the washers (which may be miles away) is laid, dividing the field into equal parts. Alternate laterals curve out and run at right angles to the main track as far as the boundaries of the designated field, but conforming to the intermediate ground. The laterals are 600 ft. apart, and the space between any two of them is subdivided by a line ditch parallel to and midway between them. At this ditch two sets of workmen start their lines, in opposite directions, and at right angles to the laterals. This gives each man a space of 300 ft. long and 12 ft. wide to excavate. Over this path he wheels his "stratum" in barrows to his portion of a platform running at the side of the road. Here his work is sharply scrutinised by a foreman before it is loaded on cars for the washer. This material furnishes about one-third its weight of clean washed phosphate. When mining is carried on in wooded land, it is difficult to keep the lines straight: trees are undercut with mattocks, and thrown behind upon the high ground, the rock being picked out. In undrained territory, or old rice fields, where the alluvial character of the soil makes deep ditching impossible, steam pumps are employed.

Pyrites.—The Virginian pyrites deposits consist of a series of lens-shaped bodies of varying size: sometimes several hundred feet long and 80 ft. thick. The general practice is to develop them by inclined shafts. In some mines, stopes are opened up by driving levels along the lens at economic intervals; these are connected by raises, and the ore is broken down by overhand stoping. As the walls are comparatively strong, little timber is used, and the levels are protected by massive pillars with small openings into the stopes, through which the ore is drawn to be loaded into mine cars. But where the walls are very soft, the ore is much faulted, and the dip is variable, the inclined shaft is cut into the slate to some extent, to avoid the sharper bends which the ore-body makes. Levels are run off at intervals of 40–90 ft., according to dip, and pushed to the extreme end of the lens as fast as possible, an occasional raise being made for ventilation, which is widened out into a flask-shaped stope as drifting proceeds. As the stopes are lengthened, "break-throughs" are made into the levels, every 30 ft., from which the mine cars are loaded. When the dip is steep enough to allow the ore to slide, shoots are employed. As the ore is extracted, the roof is held by round pine props or stulls. A thin rib of ore is left at the top of a stope to carry the track above, until that level is stoped out. Draw slate is packed away in the middle of the stope, and held by cribs when necessary. When a level has been thus stoped out, robbing of ribs and stumps begins at the outer end, retreating toward the shaft. When the props hold up the back over too great an area, they are shot out to keep the ground caved nearly even with the retreating party. By this method, little ore, except that shot into the slate packs, is lost in mining. As much slate as possible is picked out and left in the stopes. Machine drills (2½-in.) are used for development, and for stoping where the ore is thick enough. Where the ore is thin, and in robbing, hand drills are used to better advantage. Some 2-in. drills, run by one man, are used in stoping with fair success. As the dip is seldom over 40°, no level-pockets are provided at the shaft, but the mine wagons dump direct into the skip, which is of 3000-lb. capacity. Tramming is done by hand.

The total cost per short ton of mine-dirt hoisted, cleaned, and loaded, where the lenses measure some hundreds of feet in length and breadth, with an average thickness of 5 ft., the mine hoisting about 4000 t. per month from moderate depths, is: Labour, 3*s*. 2½*d*.; timber, ½*d*.; powder, etc., 6*d*.; drill spares and pipe, ½*d*.; fuel, 3*d*.; oil and waste, 1½*d*.; tools and supplies, 1½*d*.; total, 4*s*. 3½*d*.

Salt.—An effective method of raising salt is by solution; a column of descending water is made to raise the brine nearly as high as the differences of sp. gr. between the two liquids will permit. Saturated brine contains 26½% by weight of salt, and has a sp. gr. of 1·204; hence a column of such solution of 997 ft. will support one of pure water having a height of 1200 ft., or a column of fresh water of 1200 ft. will bring brine within 203 ft. of the surface. A hole, say, 12 in. diam. at surface, is commenced, and retained of this size as long as it is safe, on account of its own weight, to let down a wrought-iron tube something under 12 in. diam. This tube is ¼-in. thick, and is used to support the sides of the hole. Boring is continued, and a second length of tube is lowered inside the first, and so on until the bottom of the salt-bed is reached. The outer tube, where it passes through the salt-bed, is pierced with two sets of apertures, the upper edge of the higher set coinciding with the top of the bed, and the other set occupying the lower portion of the tube. Within this tube, a second one is lowered, of outer diam. 2–4 in. less than the int. diam. of the first. This serves for pumping the brine by way of snore holes. The pump itself is ordinary, but at surface is a plunger, which serves to force the brine into an air-vessel for purposes of distribution.

The bucket and clack are some feet below the point at which the brine is raised by the column of fresh water descending in the annulus between the two tubes. The rate at which salt is dissolved depends on extent of surface exposed to action of water. This, at first, is very slow, and the quantity of salt for some months furnished by a well is inconsiderable, and the brine raised is very weak. When the pump is put in motion, it draws at first the stronger brine, which, from its greater sp. gr., occupies the lower portion of the cavity. As it is raised, fresh water flows in through holes in outer tube. The solvent power of newly-admitted water is greater than that of water partially saturated; being also lighter, it occupies the upper stratum of the excavation. The effect of these two circumstances is the removal of much more salt from the upper surface of the bed than at the lower, giving the cavity the form of an inverted cone, and weakening the roof.

Another consideration is that the solvent action of the water may reach far beyond the owner's land boundaries, and adjacent property may be robbed. Finally, there must necessarily be extensive and even dangerous settling of the surface sooner or later, especially if the depth is relatively shallow. The rig adopted in boring brine-wells is practically the same as that used for petroleum and in prospecting (see p. 158).

Sulphur.—A novel method of extracting sulphur from beds at several hundred ft. below the surface, in Louisiana, consists in melting the sulphur by superheated water forced down through iron pipes, and pumping it up in a still molten condition. Wells are bored, as for petroleum, to the bottom of the sulphur bed. Down this well is run a system of pipes, one within the other, extending not quite to the bottom of the well: the outermost pipe, 10 in. diam.; within this, a 6-in., then a 3-in., and finally a 1-in. Water heated to 335° F., is forced down through the annular space between the 10-in. and 6-in. pipes, and issues through a number of perforations in the side of the pipe at a point 2–3 ft. above the bottom of the well. The water, because of its great heat and pressure, forces its way through the seams and crevices of the limestone rock, melting the sulphur, and causing it, because of its superior gravity, to drain to the bottom of the well. Here it enters the pipe through a number of perforations, and passes up through the annular space between the 6-in. and 3-in. pipes. Normally, the two columns of liquid, water and sulphur, would stand in equilibrium at levels whose height would be inversely as their respective sp. gr., the water column twice as high as the liquid sulphur column; so that, when the top of the water column was at the level of the ground, the top of the liquid sulphur column would be halfway between the bottom and top of the pipe. To bring the sulphur to surface, compressed air is forced down through the 1-in. pipe into the liquid sulphur, and the density of the latter is thereby reduced, until it is less than that of the water, and the mixture of sulphur and air rises and flows out in a steady stream at the surface. Some 400–500 tons of sulphur may flow from a well in a single day, this rate being maintained for weeks at a time; and one well has actually furnished over 60,000 tons. The wells are sunk at distances of 50–100 ft. apart. The liquid sulphur is allowed to flow into large open bins about 250 × 350 ft., bulkheaded in with timber, and of sufficient area to permit of the layers forming and cooling first on one side of the bin and then on the other.

Talc.—A remarkable deposit of talc exists in New York State, affording some 50,000 t. per ann. The bed is 12–18 ft. thick, sometimes reaching even 70 ft., and lies at an angle of 30°–60°. A typical mine is opened by an incline, following the foot-wall, to a depth of 300 ft.; for its 16-ft. width it stands well without timbering. The height of the incline is the thickness of the talc, say, 15 ft. In this incline is a single skip track and a stairway for men; at every 30–50 ft. on each side are opened wide-arched tunnels 15 ft. high. After the tunnels have advanced about 25 ft. from the side of the incline, a raise is started to the next level; when holed through, the talc is stoped out clean between walls, underhand, for 40 ft. along the vein. The stope is then squared down to the lower level, and tunnelling is begun under the next pillar, preparatory to another raise and underhand stope beyond. It is not usual to extend the tunnel and its accompanying stopes more than 600 ft. each way from the incline, as tramming costs then become heavy. Usually the bottom level only is in operation; but by leaving a shelf of talc on the foot-wall of the upper level, to support the car-track when the raise comes up, as many levels as may be wished can be kept open with this system, if the face of the upper level is always kept in advance of the one below. The pillars are left 25–30 ft. square, as the cleavage of the talc makes round pillars unsafe. By leaving 1 ft. of talc under the mica-schist layer on the hanging wall, the roof stands well, so that no stulls nor special "roofmen" are necessary. An output of 32 t. daily takes a mine force of 1 shift boss, 1 drill runner, 1 helper and 4 muckers, working the day shift only. The drill works on one side of the incline while the muckers are cleaning out the broken rock (from the previous day's blasting) on the other. It is regular and easy drilling: one shift can break 60–70 t.; its cleavage along bedding-planes causes it to split off in huge slabs. Loading is with 40% dynamite, tamped by a clay roll, 12 in. long, sent down from surface. Care is taken that only clean talc shall be broken down; this is usually an easy matter, in the whole width of the vein, except for occasional horses of tremolite.

Tinstone.—The very extensive stripping operations, involving removal of many feet of basaltic overburden, connected with the Briseis and adjacent mines of alluvial tinstone in Tasmania, are worthy of special mention.

A large heading was run in the top layer of drift above the washdirt, timbered with legs and caps, and laths were driven at the sides and partly on top; a space of 3–4 ft. was left in the centre of the top, across which were placed removable transverse slabs (locally termed "Chinamen"), constituting a pass for the whole length. The material brought down (by blasting, generally) in the heading was passed at once into trucks by withdrawing a few "Chinamen," a trimmer, with heavy rake, controlling. When possible, ground is picked down directly into the trucks. This work, including much blasting, was done for $9\frac{1}{4}d.$ per cub. yd. (wages $7s.$ $6d.$–$8s.$). Trucks held 4 cub. yd. loose material (or $2\frac{1}{4}$–$2\frac{1}{2}$ cub. yd. in the solid) weighing 4–5 tons. These were hauled out by horses, released by the drivers with simple spring bars, the wagons running against "bumpers," and tipping themselves. The smaller material was washed out, along flumes, by a nozzle, the bigger and heavier blocks being rolled down through the top of the heading into wagons. The heading needed to be substantially built, as the face was nearly 150 ft. high, and it was difficult to regulate the falls of boulders with slurry amounting to 1000 cub. yd.

The larger portion of the overburden at this level had not sufficient rock to justify driving a heading; but unless the rock present was speedily removed by some means, the face was soon blocked with it, and the water could do no good. Ordinary mechanical methods—such as a steam shovel—were out of the question, the rocks varying much in hardness and size, and being too scattered to be treated in a large way. Any method of dealing with them had to be subsidiary to the use of water. Hand-trucking was adopted, when the lead was short—small trucks with movable sides and ends, and capable of carrying about $\frac{3}{4}$ cub. yd. in the solid. Small fiddle-stick wagons were subsequently constructed, with hinged backs that could be let down to permit of boulders being rolled in, and holding $1\frac{1}{2}$ cub. yd. of solid material, or nearly 3 t. Two wagons can be conveniently run out by one light horse, switched on to separate roads at the tip, and tipped simultaneously. Where convenient, temporary stages are constructed daily on a level with the tops of the wagons, so that boulders can be rolled in without lifting; such stages take a few minutes to construct—the ends of a spar carrying them being supported on flat-sided boulders, and laths laid on top to form a deck.

The main points for working in this way are—(a) to keep flumes as close up to the face as possible, and at a grade (1 in 20) to give the water every opportunity, otherwise its power is lost amid the boulders that accumulate, and the water drains away, leaving material entangled in the interspaces; (b) to have plenty of branch roads, kept well up, so that the rock is always handy to load; and (c) to have several faces going at once, so that you can be getting a fall at one, washing down fallen stuff at a second, and trucking at others. In this way, with a maximum water-supply of 72,000 gal. per hr., and material limited to 2¼ in. diam. (in the tail race), 500 cub. yd. per day was averaged for some weeks. With an average water-supply of less than half this, the output was 260 cub. yd. per day, at a cost of 7–8*d.* per cub. yd., including all charges; half the material had to be handled and trucked, some of the larger boulders also required shooting, and the lead was about a mile long. When water was available, 3 shifts were worked with the nozzle, the tailings being run over a grating, where all material above 2¼ in. diam. was forked into trucks and hand-trucked to dump. Two shifts were occasionally worked with the 12-yd. trucks, but the heading could only be worked one shift, because if one shift was used to fall material and wash the small stuff out, it had to be left for an entire shift to drain. Nozzles at the stripping face have a head of 320 ft.

At Mount Bischoff (Tas.), the tin alluvial is mixed with boulders and occasional portions of trees. It is worked by open quarrying. Small sloping benches are laid out, and then undercut; when the ground shows signs of caving, the men are withdrawn, and heavy shots are fired to bring down large quantities of material. The dislodged ore is hand-loaded into small cars, and hauled up an incline from the excavation, by a rope drawn by a stationary engine on the outer edge of the ore-body. The cars are detached from the rope, and run into a tippler, whence the ore falls into large bins to be reloaded into railroad cars, and hauled to the mill a mile distant. In the softer ground, a miner can excavate up to 12 t. per diem; in the harder, only 4 t. Wages are 7–8*s.* a day. There is much unnecessary handling of the ore. Official figures as to cost are—Mining, including development and maintenance, 3*s.* 3*d.*; filling, hauling, and emptying trucks, 5*d.*

The Anchor mine, also in Tasmania, is operating open quarries in granite, employing jumper drills (making 22-ft. holes), and various explosives, according to the ground (powder, rackarock, gelignite, and blasting gelatine). Much of the granite broken is not tin-bearing, that peculiarity being confined to the black mica (biotite) variety; consequently much of the rock shot down has to be thrown over the dump. The selected granite is trucked (horse cars) to the breakers (arranged in tiers), and thence to the stamp mill (100 head), where it is crushed to pass a 10-mesh screen, and mechanically dressed to about 69% metal. Though tinstone is very rarely visible in the rock, the contents commonly range about 3–4 lb. per ton. On a year's output of 118,634 tons, the total costs of mining (per long ton) are—Stripping and development, ·54*d.* (·5*d.* being labour); quarrying, 13·56*d.* (11·13*d.* labour); trucking, 1·97*d.* (1·9*d.* labour); total, 1*s.* 4·07*d.*

VEINS.

The extraction of mineral from deposits having the form of veins, and standing at a more or less steep angle, is known as "stoping," and the working place is called a "stope" or step, from the fact that the operation is conducted in such a manner that the solid ground remaining takes the appearance of a flight of steps, as seen either from before or from behind. Stoping commences from the point of junction of a level and a winze or rise, and consists in the removal successively of the separate blocks of ore defined by those workings; but the operation may proceed from below upwards, or from above downwards, or partially in each direction. In the first case, the vein to be removed stands above the men who are drilling, forming an "overhead" or "overhand" stope; in the second case, it is below them, making an "underhand"; and in the third case both conditions are presented, and it becomes a "combination" stope. These embrace the simplest features of ore-winning methods, but there are many variations of practice, dictated by the characteristics of the vein (width, angle, hardness, etc.), and of the adjacent country.

The width of ground removed in stoping is controlled not only by the size of the ore-body but also by the nature of the walls. Many gold-veins are less than 1 ft. thick, but the practical minimum space in which men can work effectively is about 2 ft., and few stopes are less than 3 ft. wide, because it costs more to break narrow ground by which the men are constantly impeded. The flatter the reef, the wider must the stope be. In many cases, one wall is comparatively soft, or a selvage of rotten rock or even clay occurs; this is always availed of, and the "cut" (see p. 210) is made in it, often permitting the veinstone proper to be broken independently, and thus kept free from admixture of worthless material: more particularly is this the case when hand-drilling is adopted.

Timbering Stopes.—As the vein is removed, the walls are deprived of their natural means of support; moreover, at the same time, is almost invariably broken a certain amount of waste rock, "deads," or "mullock," which it is desirable to eliminate from the ore at once, and for which accommodation must be found; and finally, the miners must be afforded protection from falling rock in underhand stoping, and often require a working platform in overhead stoping. All these purposes are served by cross-timbers from wall to wall, called "stulls."

Stulls are invariably round timbers, being sections of straight-grown trees, cut to suitable lengths and deprived of bark. Examples of stulls in position are shown in Fig. 137. When the vein is vertical, the stull lies horizontally as at *a*, *b*, *c*, and each end is inserted in a shallow notch or "hitch" *d* cut in the solid rock.

Poles or slabs *e* are laid lengthwise of the stope to unite the various stull-pieces, and at least 1 ft. or so of waste rock *f* is spread on these, to keep them in place, and break the blow of any falling body. In a vein of considerable width, or where the walls are bad holding ground, the stulls must be reinforced by a "saddle-

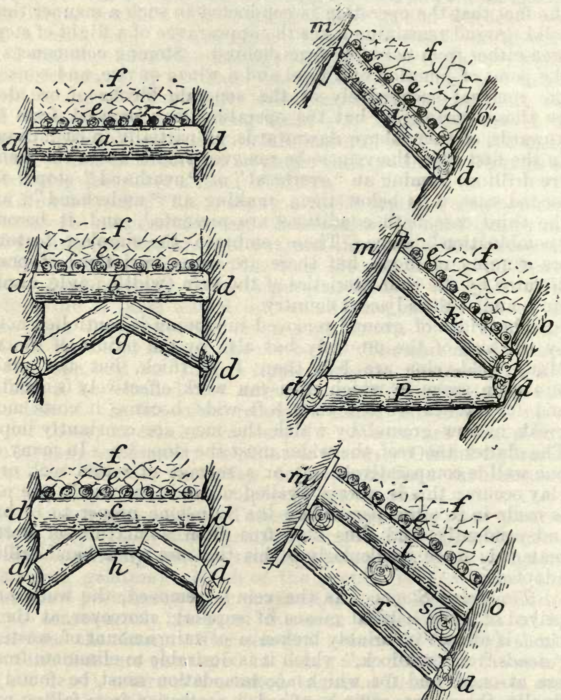
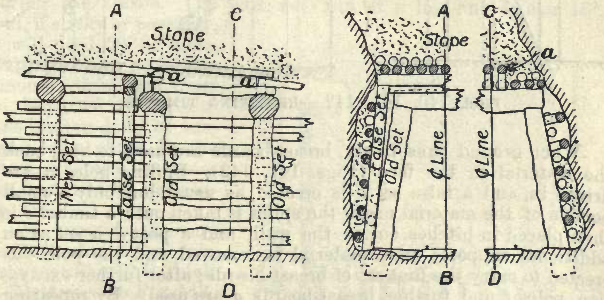


FIG. 137.—TIMBERING STOPES.

back," as *g*, of two pieces, also deeply hitched; or an "arch" *h*, of three pieces: sometimes the saddle-back, with a lagging and rock-covering, is made to dispense with the stull-piece. When the vein is not vertical, the stull-piece *i*, *k*, *l*, is set at an inclination, which usually approximates one-fourth of the angle of dip of the vein, so that the head of the stull-piece shall not be released and let fall by a slight slip of the hanging-wall *m*. A head-board *n* is used, when the hanging wall has a tendency to "wind" off, or is soft, so as to distribute the holding pressure of the stull-piece. If the footwall *o* is yielding, or liable to break away, the stull must be supported at each end by a false stull *p*, hitched deeply into both walls; and, in very wide veins, it may be necessary to adopt double stulls as at *r*, with stout intervening poles *s* running lengthwise, and often in this case the stull-piece proper is not hitched at all.

With ground that breaks heavily, that is, has a tendency to shed pieces weighing a ton or so, it is well to take down at once, by shallow holes and small charges, as much of the wall as will cover the stulls to a depth of 3–4 ft., as this loose rock settles and packs firmly, owing to the repeated vibrations caused by adjacent shot-firing, and then materially aids in relieving the stulls from direct downward pressure.

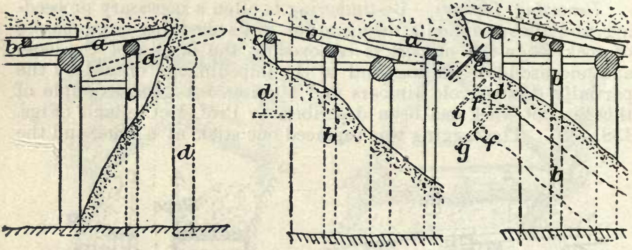
Renewing Timbers.—Re-timbering is often a necessary proceeding, and always difficult, owing to the walls shedding large masses of rock when the support is removed, to the falling of the loose and unconsolidated filling, and to the impediments offered by the partially decayed old timbers and lagging. A good example of this kind of work has been described by Prof. McClelland (Figs. 138, 139). The lagging was replaced one stick at a time, and the



FIGS. 138, 139.—RENEWING TIMBERS.

end of the pole over the old set rested temporarily on short blocks *a*. After the roof had been relagged, the sides were similarly renewed, and then a 6-in. false set was erected, of outside dimensions somewhat greater than the standard set. The false set supported the lagging while the old timbers were being removed, and new sets placed in position; the support was then taken down, to be used again. A timberman and his helper averaged about one set per 8 hr., and the cost per set replaced was approximately as follows: Timberman, 11*s.* 6*d.*; helper, 10*s.* 6*d.*; timber (including framing), 16*s.* 6*d.*; tramming, mucking, etc., 3*s.*; total, 41*s.* 6*d.* The sets being about 4 ft. 8 in. apart, the cost was approximately 9*s.* per ft. of drift re-timbered.

In reopening caved drifts, if the face will stand for a short time unsupported, enough broken ore is removed to make room for a new set, which is then erected. When the ore runs, spiling becomes necessary, as in alluvial mining (pp. 312–8). The timbering is carried forward, and spiling poles *a*, driving out the blocks under the lagging, are forced into the caved material as far as possible (Fig. 140). The poles are kept at the proper inclination by wedges *b*; some of the caved material is removed, the spiles are driven ahead, and a false set *c* is erected to support them; the spiles are driven home, and the new set *d* is erected.



FIGS. 140, 141, 142.—RENEWING TIMBERS.

When ground runs freely, breast-boards are used to hold back the material in the face (Figs. 141, 142). Spiling poles *a* are driven in, and a false set *b* is erected, as usual, but only a small portion of the material under the spiles is taken out; a timber *c* is then placed in hitches cut in the wall, and a post *d* is set as an additional support; more material is removed, and the pole *e* is erected to carry the first set of breast-boards; after further excavation, poles *f* and further breast-boards *g* are used. By repeating these operations, the whole face of the drift is opened up, and a new set of timber is erected close to the breast-boards.

Substitutes for Timber.—Increasing difficulty and cost of procuring suitable mine timbers have in several cases led to other materials being substituted. In collieries, especially, iron and steel supports are coming much into favour (see p. 358), and justify their greater cost. But often much better use might be made of the waste rock for supporting worked ground, if more trouble were taken to select it. Thus, at Ouro Preto gold mine, which has a flat dip (20°), timber is entirely dispensed with in the stopes, advantage being taken of the slab-shaped quartzite blocks of waste for building dry arched galleries and drifts. Where the vein is of such width that no quartzite is broken, it is brought from other parts of the mine. And, in the Baltic copper mine, walls of levels and ore-passes or "mill-holes" are exclusively built of hard lumps of waste rock, resulting in very great economy of timber.

Overhead Stoping.—An overhead stope is opened out, as a rule, from opposite sides of the bottom of a rise, though, when the amount of "backs" is small, the work may commence from the roof of the drive. As the stopes advance, "shoots," "mills" or "passes" are carried up, to receive the broken ore and convey it to the trucks on the level below. Sometimes these shoots are constructed of heavy pieces of waste arranged in dry-masonry fashion to form a circular well; or they may be built of short logs precisely like a crib or box-rise, as described on p. 296; and rarely they are lined with planks—when set at such a low angle that the ore would not otherwise run. The distance apart of these shoots is mainly controlled by convenience for shovelling the ore to them. The extreme capacity of a man in this respect is 12–15 ft., and the economic limit is 8 ft. A common practice is to fix 25–30 ft. as the interval between shoots; but in many cases where all wages are high, and the margin between skilled and unskilled labour is not great, this is more costly than reducing the interval to 8–10 ft., and multiplying the shoots. Ore will not run at a less angle than 45°, and, if sticky or powdery, it will need 50° or more. Failing this, a certain amount of motion must be imparted to it, and this is often done by suspending a chain in the shoot, whereby the mass can be agitated. See also pp. 429–31.

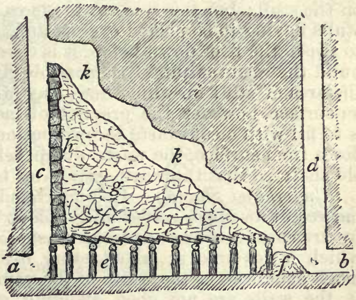


FIG. 143.—OVERHEAD STOPING.

An example of overhead stoping on a vein not exceeding 12 ft. thick is illustrated in Fig. 143. From the drive *a b*, the winzes *c d* have been put up towards a higher level, thus isolating a block of ore for stoping. The drive *a b* is timbered with posts, caps, and lagging *e*, forming at once the roof of the level and the floor of the chamber being excavated. At *f* is a heap of broken ore lying on the floor of the drive, but which should properly be carried in a shoot for automatic delivery to the trucks. The vein affords a large proportion of waste *g*, which is allowed to accumulate on the timbers, and forms the miners' foothold; but if the depth be considerable, series of stulls must be run to break up the weight. Next the rise *c*, the coarse waste has been packed in a wall *h*, to save timber; this, however, is not always feasible or advisable. The miners wield their hammers in the space *k*.

While, as a general rule, the stope is commenced from the point of junction between level and rise, there are cases where it is preferred to secure the safety of the level by leaving a pillar or block of solid ground about 6 ft. deep overhead, instead of employing stulls; the cost of timbering and timber is thus saved, but a certain amount of the standing ore can never be extracted. Thus it becomes a question of relative costs of timber and labour on the one side, and net value of ore lost on the other.

The advantages of overhead stoping are that much less timber is needed for stulls and protection of men, and that the ore falls naturally away from the face and towards the shoots; its disadvantage is that, in hand-drilling, the hammer blows have to be delivered from the least effective position. It is by far the most widely adopted, and is safer where the walls are bad. But wherever waste is broken and stowed in stulls, some ore is certain to get mixed with it, the broken vein in many places falling on the stull. This may be in part obviated by having the stull out of the line of fall from the working face; it is the finer and richer portion of the ore which is the more likely to reach the stull. An effort to minimise this, in the Robinson mine, was made by placing canvas on the stull; but this was cut to pieces, and the plan failed, some stulls having to be picked over again. Other precautions tried are to sweep the floor down before it is covered up by stulling; and to build the waste as quickly as possible right up to the roof, to lessen the area of stull on which fines may fall. With improved sorting on surface, however, the practice of most of the Transvaal mines is to fill with clean waste thrown out on the sorting floor, and sent down from surface, the waste dump being run out over a winze or old shaft, so that the waste gravitates to whatever level needs it.

Underhand Stoping.—In underhand stoping, the work is commenced at an upper corner of the block that has been "developed" by drives and winzes, and the waste is piled on stulls, a line of which is required at about every 6–8 ft. The cost of these is much enhanced as the width of the vein increases, and particularly if the walls are not good standing ground. Consequently the method is best adapted to narrow veins. It is applicable when the ore is friable and rich, because it minimises the loss in breaking, the broken ore and dust falling upon solid vein. It has a further advantage in that all drilling is downwards, securing the maximum effect of the hammer-blows; and the importance of this condition is increased in cases where the coloured labour employed is not proficient in striking upwards. Where the vein is small and hard, underhand stoping will often be 20% cheaper than overhand. The former objection to underhand stoping of small veins which necessitated the breaking of much waste rock, that the waste could not be separated in the stopes, is removed by the sorting arrangements now in vogue, and the economy of underhand stopes frequently outweighs the additional cost of hoisting and sorting.

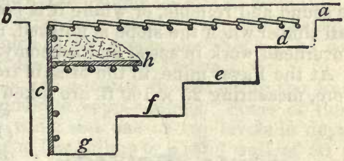


FIG. 144.—UNDERHAND STOPING.

An example of underhand stoping is shown in Fig. 144. From the main adit *a b*, is sunk a winze *c* on the vein. Then, commencing at *d*, miners standing on the ore in the vein excavate it in a series of steps *d e f g*, the ore having to be raised by windlass or other contrivance through openings in the floor timbers of the adit *a b*, whilst the rubbish is thrown back on strong timber shelves or stulls *h* built against the wall of the winze.

A much better plan, however, is to carry the winze completely through from level to level, and stope into it from opposite sides,

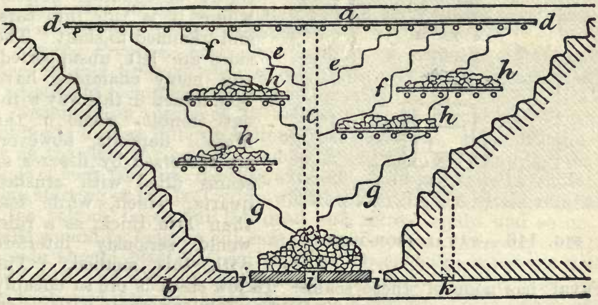


FIG. 145.—UNDERHAND STOPING.

passes being made at intervals of 20–30 ft. (Fig. 145). Between levels *a b* is sunk or raised a winze *c*; a row of stulls *d* is placed as fast as sinking proceeds; or a corresponding block of solid ground is left underneath the level, the stoping commencing from both sides of the top of the winze, and the stopes being carried down successively as shown by the wavy lines *e f g*. Further stulls *h* receive waste rock, and afford protection to the men against possible falls of ground above them. Passes or shoots *i* are prepared somewhat in advance of the stope, by rising from the lower level, as at *k*. Stulls are built, or blocks of solid ground (as between *i i*) are left as a protection. This system, where the walls are sound and reliable, or where it is possible to draw much of the stull timber when the stope is finished, is in many respects superior to overhead work in speed and economy.

At the Fayal mine, in the Mesabi iron region, enormous blocks of ore, measuring 25 × 100 ft. are taken out by underhand stoping, with very little timbering (Fig. 146). First a wide drive *a* is run just under the sand capping *b*, and supported by saddle-back timbering *c*, which becomes the chamber roof. This roof timber is put in by driving from sub-drifts on same level, to avoid hoisting. After the roof is securely supported, the ore is underhand stoped through rises *d*, to level *e* in centre of chamber bottom, where it is run into cars, and trammed to shaft. The sides are left unsupported, and many chambers have been mined in this way without trouble. Some of the Mesabi deposits, however, are traversed by fissures or seams filled with crushed quartz, which, while less than 1 in. thick, as a rule, would seriously interfere with this method. The Fayal has none of these seams. In few cases is ore so cheaply won. Chambers are about 60 ft. deep, and alternate with 24-ft. pillars.

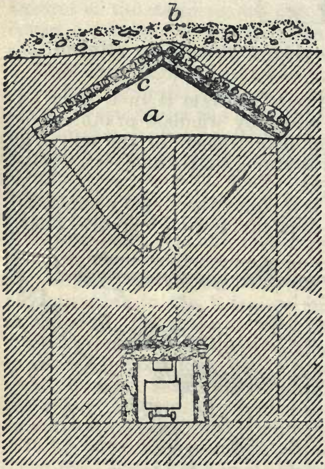


FIG. 146.—FAYAL IRON-MINE.

Combination Stoping.—The combination stope, as its name implies, is in part overhead and in part underhand. It is in favour at many S. African mines, and is very suitable where the "backs" are long. The lower portion of the face is worked as an overhead stope, and the upper portion as an underhand stope, the proportion of each depending upon the dip of the reef: thus with a dip of about 55°, the underhand stope is in greater proportion; and with a dip of under 30°, the overhand stope is in greater proportion. With high dips, the underhand portion gives the advantages of underhand stoping, while the overhead portion affords greater facilities for packing waste; with low dips, the underhand portion permits a good deal of the ore to be thrown up to the level above, instead of being passed down the greater length to be trammed away from the lower level. With long backs, it is often advisable to leave a pillar in the centre of the stope, and this can be conveniently and inexpensively done under this system.

Practically similar means are adopted in extracting the huge pockets of ore in the Namaqualand copper-mines. Some of these stopes are 300 ft. long, 50 ft. wide, and one or two levels in depth, without interruption; while a few pillars or a solid wall of 20 ft. may remain between one stope and the next. As all timber has to be imported from Europe, the strength and firmness of the ground are incalculable advantages in the mining of these immense bunches of ore. Fortunately, all that is necessary, for the most part, is to preserve a more or less regular arched form in the back of the overhead stopes. In these, the general method of procedure is as follows—The ore as it is broken is left to pile up and form a floor for the drill men, until the stope has been worked three-fourths, or more, of the distance to the level above. The remainder of the block is then broken underhand with long holes. The old stopes are filled in as much as possible with waste rock, but too little of this is broken to fill more than a portion of them, so that many large worked-out chambers remain empty. (Chalmers.)

Stoping Practice. (See also Drilling, pp. 210–19.)—Different mines possess different working conditions, and result in variations of practice to secure most economical results.

Generally, in a stope of 35° or less, it is good practice (Wager Bradford) to open out the bottom from the winze, undercutting as far as the first box-hole on each side, and gradually forming the whole of each face into benches, extending from winze to box-hole; then, starting at the bottom, to work back to the last bench, return to the bottom, break out the dead end, retreat again, and so on. Drills should be kept on adjoining benches to maintain sequence; no man should blast his own front and back holes bored on the same shift, and where a bench does not break properly, it must be put in shape before the work proceeds. The ground always breaks away from the machine and toward the stope-boxes, thus materially aiding in shovelling; the benches are never so buried with broken rock, as when they are carried down, and a "leader" is thus more easily followed; there is less tendency to bore holes into the hanging wall, with consequent increase in waste; and machines lifted from the bench they have drilled to-day, are safe on the one they must drill on to-morrow, and, if time admits, may be rigged up and left standing for the next shift, thus saving time. The system lends itself to the shaping of pillars on the lower side, and, by starting at the right distance above the pillar and benching down to it, the face of the stope, when the pillar is holed, is in such condition that the benches above and below readily merge into each other. Finally, it requires less room. This refers especially to S. Africa, but has wider applications.

In lodes consisting of alternating bands of rich stone and waste rock, as often occurs, for example, in the South Reef series of the Rand, the practice of stoping the whole 8½ ft. in one operation and relying on sorting to remove the waste is open to the objection that 30% of fine waste will go with the milling dirt. Assuming the section to be—4 in. waste, 18 in. reef, 16 in. waste, 3 in. reef (say 45 in.), 43½ in. waste, 1½ in. reef, 16 in. waste: Carter suggests "resuing," i.e. removing the top 45 in. first, over a large area, then taking up the barren section (say 40 in.), and finally getting the rich 1½ in. vein almost clean; and in lifting the barren seam, he would hole parallel with the band and not across it.

Some practical experiments carried out by F. C. Roberts, on narrow, rich and steep veins, by full stoping and resuing respectively, showed the following contrasts:—

(*a*) Reef 6 in. wide, value 50 dwt. Results:—

Full Stoping.	Resuing.
100 t. milled contain only 10·41 dwt., yielding 7·8 dwt., value 150*l*., cost 155*l*., loss 5*l*. (104 t. mined, 4 t. sorted).	100 t. milled contain 50 dwt., yielding 37·5 dwt., value 720*l*., cost 426*l*., profit 294*l*. (500 t. stripped, 104 t. waste handled).

(*b*) Reef 12 in. wide, value 30 dwt. Results:—

100 t. milled contain only 12·4 dwt., yielding 9·3 dwt., value 180*l*., cost 154*l*., profit 26*l*. (103 t. mined, 3 t. sorted).	100 t. milled contain 30 dwt., yielding 22·5 dwt., value 432*l*., cost 292*l*., profit 140*l*. (250 t. stripped, 23 t. handled).

With very strong ground, such as obtains in the Boundary District, B.C., immense stopes over 100 ft. square permit as much as 50 t. per machine per shift to be broken. The men stope from broken ore instead of scaffolding, and the only timber used is for shoots, which can be removed. It is practically "glory-hole" (open-cast, filling into shoots) practice carried underground.

Somewhat the same system is being widely introduced into W. Australia. The first heading (6 × 8 ft.) of the drive is carried 4–6 ft. above the level proper; then other machines following 10–20 ft. behind take out the leading stope, lift the bottom, and cut back to one wall (preferably the foot). No passes or shoots are used, and no timber, unless compulsory. The ore falls on the level timbers, and remains there, being drawn off so that the top of it may be 4–8 ft. from the back of the stope. Bars are rigged on it and against the back. The stope is carried forward and up in breasts and backs 7–10 ft., and is taken off on either side of the rise or winze between the two levels which provides ventilation. When all the reef has been mined, the stope will be full of broken ore, representing 60–70% of the total broken, about one-third being drawn as the work progresses, to provide room for the men. The shoots are large and ample, about 12 ft. apart, depending on width of lode. This method is very safe; there are only 6–8 ft. of hanging-wall between broken and unbroken ore; and men easily reach the back of the stope to work down any loose rock, and can at all times sound the wall and back. Air is compelled to travel along the back and face, just where the men are working; dust and smoke are thus rapidly carried away. The men can work and move about freely and safely; staging is done away with; and bars are as firmly rigged as in a drive, and are easily handled and moved, saving much time in rigging. Filling is not necessary while work is in progress, and should there be large exposures of hanging-wall, while drawing off the remaining 60% of broken ore, little danger or damage results. As the drawing off advances, the necessary filling follows close behind. With dangerous walls and heavy ground, the broken ore acts as filling and support. Only one connection is necessary between levels. With steep lodes (say 55°), requiring no shovelling, it works admirably.

In some of the big iron mines of Minnesota, diamond-drill holes 20–33 ft. deep are employed. In a stope 40 ft. wide, 2 holes 24–30 ft. deep, one pointing a little to the hanging and the other towards the footwall, are charged with 30–50 lb. of 50% dynamite, and bring down 500–1000 tons at a blast. When using ordinary drills, 8–15-ft. holes are general; these are "squibbed" or chambered with dynamite, and fired with black powder. Passes at every 100 ft. in depth are found to give maximum economy, it being cheaper to run extra levels than to have deeper shoots.

Slicing.—With very massive ore-bodies, e.g. those over 12 ft. thick, it is necessary to deal with the width of the vein in successive sections. This is illustrated in Fig. 147. A main gangway *a*, is first driven along either wall *b c* of the vein *r*, and is substantially timbered. From it a series of breasts or crosscuts are driven at right-angles through the vein till they reach the opposite wall *c*. These breasts are 6 ft. high and 6–12 ft. wide, and are so worked as always to have firm ground on both sides of them, either solid ore or rubbish *d* stowed back in a former breast. As one level is worked out, new gangways *o* are driven overhead, and the cross-cutting is repeated, with timbering *p* where necessary, and always providing that no two breasts in the same vertical line shall be worked simultaneously.

Something similar is done in some of the wide (and liable to crush) gold-quartz reefs of Victoria (Fig. 148). Main levels are first driven from the crosscut along the footwall; they should be of ample dimensions—not less than a cap of 4 ft. in the clear, with 7-ft. legs spread to 7 ft. in the bottom. A convenient distance having been driven, a winze is commenced a few feet from the cross-cut, divided into two compartments, each 3 ft. clear, one as a travelling way or for timber, the other as a mullock-shoot. When this has holed through, stoping can be commenced over the back of

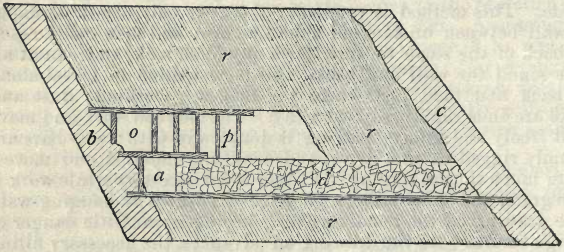


FIG. 147.—SLICING.

the level. A convenient arrangement is to first run a slice *a* from winze *c* to the hanging *d*, and commence blocking both ways from this; leading stope *b* can then be opened out along the footwall; size will depend to some extent on the nature of the ground, but, in "fair standing ground," a 10-ft. cap is generally used with 7-ft.

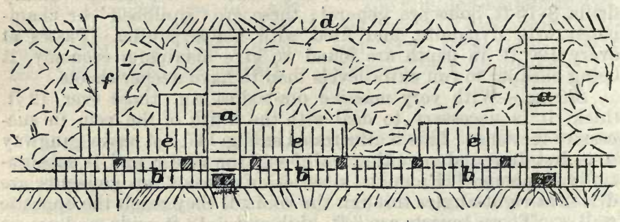


FIG. 148.—SLICING, VICTORIAN GOLD MINES.

legs. When this stope has been advanced, say 9 or 10 sets, another slice *e* is commenced alongside or "on the wing." By the time this is well under way, the leading stope has been advanced to 50 ft., at which it stops. Mullock is then sent down, and the stope is closely filled, packed up well under the caps and back laths. Whilst this filling is going on, stone-shoots or passes are logged up to the back; these (communicating with the drives below) are generally 3 ft. in the clear, and put in every 30 ft. Sometimes stone-shoots and manways are separate, a much safer plan. The second slice *e* is worked out to 50 ft., and filled, the lode being blocked out in this manner right across to the hanging-wall. To serve the stopes along the hanging, crosscuts *f* are usually run out from the main drives at every 40–45 ft., stone-passes and manways being logged up at the end of each crosscut. In some cases it is more economical to run a second main drive on the hanging side. Mullock passes should be sunk every 100 ft.; if much further apart, mullocking becomes slow and tedious. Distance between winzes regulates length of stopes. When nearing the floor of a level, the last 7–8 ft. of stone should not be taken out until the drive above ceases to be required for working purposes. It is sometimes the practice, before mullocking up, to lay poles on the floor of the first set of stopes, to prevent runs of loose mullock when coming up underneath; but, as a rule, the mullock or filling is compressed, and almost as solid as new ground. (J. V. Lake.)

Blocking.—The "block" method largely used at Broken Hill silver mines, N.S.W., is a form of slicing. It consists in running a drive in the centre of the ore-body, from three sides of which, at intervals, crosscuts are taken to the walls. The crosscuts are then enlarged to form 9 × 9-ft. rooms. When all ore is taken out, waste is filled in, and adjoining ore on either side of the packed waste is extracted in blocks 9 ft. square. The ore above packed blocks is extracted in the same way. This system permits extraction of all ore, even in the most dangerous ground, and requires very little timber. It can be varied to meet almost any condition. In the Central mine, an ore-body 300–400 ft. wide is worked by the block system, but differing somewhat. Main gangways are driven in the hanging, and crosscuts are run from them, across the lode to the footwall, at every 100 ft. Blocks 50 ft. long are then worked off from the foot, leaving pillars of equal size standing; i.e. alternate blocks are extracted on first working; worked ground is then filled with waste, kept in position by square-set timber.

The hard sulphide ore of the lower levels of Mount Morgan gold-mine, Queensland, are similarly blocked, but local hardwood timber is preferred to the Oregon pine so popular at Broken Hill.

At the Kedabeg copper mines, the ore-body is divided into horizontal layers 12 ft. thick by driving levels, each of which is divided by drifts into rectangular pillars, from which the ore is taken in vertical slices. In the first place, along the outer side of the pillar, up to the end, a stall is driven and timbered, after which, working homewards, the ore above the caps is removed. The space left vacant is filled by the caving-in of the dead rock above the ore. Working begins in the upper level and is carried downwards, and extraction can only be begun in the next lower level when the top level is entirely worked out. This involves a very large quantity of timber, which is mostly abandoned and lost; hence, only the caps are of oak, the legs being beechwood, which is less costly. (Köller.)

The practice at Rezende, Rhodesia, is as follows. (Woodburn.) A main roadway is driven in the country, about 20 ft. from the footwall, and crosscuts are put out every 50 ft. to the hanging. If the reef does not exceed 15 ft., the full width is taken by a single drive 6 ft. high between the crosscuts; if it exceeds 15 ft., either (*a*) a wide drive is made along the footwall and filled up, then the portion on the hanging side is excavated in similar manner and filled up; or (*b*) the reef is taken out by crosscuts 10 ft. wide, each filled before another is begun. The hanging is always very treacherous, and this allows only a small portion to be exposed at one time. As the reef varies in width from 6 to fully 30 ft., and frequently splits into three or more portions, a drive along one branch might find it pinching out altogether, and a crosscut would be necessary to intersect another branch, resulting in a very crooked level, unsuitable for a main roadway. By making the main level in the soft ground some distance from the foot, and crosscutting at regular intervals to the hanging, it was easy to attack the reef in small portions, no matter how wide or split-up it happened to be, and to fill up the excavations entirely with loose ground from surface, leaving nothing open but a pass for the ore at each crosscut. The first cut along the reef is made about 6 ft. high, and timbered by caps 12–15 ft. long, supported on at least 3 legs; the hanging is supported by lagging, and, where the reef is vertical, the foot is secured in the same way. Planks are placed on the caps underneath the quartz, so that the timber lining resembles that of an ordinary roadway. The top lagging and caps only are recovered; but if the legs were set small end downwards, and a lever used to lift them, many might be regained though completely buried. The ore-passes are box-rises or pig-sties 5 ft. square inside, divided into 2 compartments (1 being ladder-way); 1½–2 in. plank lining in shoot. As each 6 ft. slice is excavated, the quartz falls on to the rubbish below, which is packed together very tightly, and makes a firm surface to work on. Before each cut is filled in, the top inch of the filling below is scraped and sent down as ore, to prevent fine gold from being lost. Where the reef divides into several branches, each is worked separately, and the solid ground between is left standing; one is taken out and filled in before another is started. If workable reef is left under a slice, corrugated iron sheets are sometimes laid along the bottom, so that when the stopes approach from below, these sheets can be caught up by the props.

Practically the same system is in vogue at the Van mine, Wales, where the slices are 30 ft. wide (wall to wall).

Pillaring.—Where the ore-body will not repay timbering or filling, sufficient pillars of ore must be left to hold the ground.

Prominent examples of this system are the Treadwell group, Alaska. The main drive is carried along the foot, and main crosscuts are driven. At intervals of 25 ft., raises are put up on alternate sides of main crosscuts and drifts; they are 15 ft. high, and shoots are put up while the drift is run, and given a slope of 60°, so that ore will run freely in them. Generally the drifts and shoot-raises are in ore; but in the Mexican, owing to flatness of the vein, they are run in the footwall slate, and the shoot-raises are put up to the ore at an average height of 20 ft. above the track. At the same time as the main drift and the shoot-raises are being run, an intermediate drift is carried directly above the main, and separated from its back by a pillar of rock 10 ft. thick; it is the same size as the lower one, and connects with the top of each shoot-raise. At the ends of the main crosscuts, and at intervals of 200–500 ft. along the deposit, the levels are connected by winzes, used as manways and for ventilation. The intermediate drift furnishes a large area for the machine-drills, in cutting out or undercutting the ground-floor for the stopes. When it has advanced about 50 ft., cutting out the stope is started. This consists of mining out a chamber 7 ft. high, 150–300 ft. long, and varying with the width of the ore-body. It is economical to cut the floor, so that it slopes from the parallel lines of shoots at an angle of about 30°. This does away with a large amount of shovelling, and the ore thus left is ultimately obtained through the stopes from the next lower level. The roof of the stope is arched, serving the double purpose of supporting the back, and offering a better surface for attack by the drills. The ore is shot down in large, thin slabs, so that the shock of falling, combined with that of the blasting, breaks it up as much as possible. No timber is used: sufficient broken ore is left in the stopes to form a solid working floor. The levels are protected by horizontal pillars 20–30 ft. thick; formerly these were left at each level, but now only those at every other level are left: even with this saving, fully 20% of the ore remains in the mine as pillars and ribs to support the ground and prevent caving.

Much the same practice prevails in the Alabama iron mines, which are opened by inclines or "slopes," beginning at the bottom of open-casts and approximately following the dip. From each side of a main slope, headings are driven on a 3% grade. When all the ore is worked, the slope-track is cut in the underlying formation; but occasionally, when the lower stratum is soft, the slope is driven the whole thickness of the iron, only the upper portion being mined, thus throwing the heading-track some distance above the slope-track. Headings at every 50–55 ft. along the slope subdivide the deposit into lifts or levels, which are worked independently. The width of headings is 15 ft., maintained for 100–150 ft., then suddenly increased to 25–30 ft. for room-headings. The portions remaining, after driving 25-ft. headings, are called pillars; these are robbed after headings have been driven as far as advisable. At 75 ft. from the slope (on both sides, and parallel with it), are manways continued for the whole length, connecting all headings, and, on one side at least, reaching surface; they are 5 ft. wide, 5–7 ft. high, and usually timbered.

Beyond the manways, at 50–75 ft., where the headings abruptly increase in width, are "break-throughs," or "upsets"—passages cut through the pillars, and 50–150 ft. apart. Their object is threefold—for running air- and water-pipes, for ventilation, and for operations preliminary to pillar-robbing. After removing as much ore as can safely be done, pillar-robbing is begun, by cutting off transverse slices, slopeward. Pillars are robbed in the upper levels first, to protect men, tracks, and cars from falling ground. As a rule, the pitch of the beds suffices for gravity transference of broken ore to heading-tracks below; if not, branch tracks are run diagonally up the slope to the pillar face. (W. R. Crane.)

Some of the Lake Superior copper mines follow the same fashion, more or less—incline shafts from outcrop, overhand stoping, pillars, and some filling. Distance between shafts is mainly controlled by ore-handling methods underground, notably tramming, and ranges from 570 to 2370 ft. Levels are at 100–125 ft., and measure only 7 × 6 ft. for first 25–40 ft. from shaft, afterwards expanding to 8 ft. × width of lode. Stoping operations are divided into drift-stoping, cutting-out stoping, and raise-stoping. In drift-stoping, a passage 25 ft. wide (6 ft. drift and 19 ft. stope) is run parallel with a level, the drift being paid for at per lin. ft., and the stope at per fath. broken. Cutting-out stoping is mainly ore-breaking—portions of stope face 7–10 ft. wide × 100 ft. long are removed in succession parallel with level. Raise-stoping is begun on the footwall side of a drift or drift-stope and carried directly up to a break-through into next level. System is aimed at in leaving pillars, but position, kind, and size may all vary with conditions of lode (width and pitch) and hanging-wall. Shaft pillars are invariable. When a shaft is sunk in a lode, pillars consist of the portions of lode left between shaft and workings on both sides, 25–50 ft. wide, making a total width of 50–100 ft. of pillar for protection of the shaft. Shafts sunk in the footwall have the same width of pillar, but the shaft being outside the lode leaves one continuous pillar, 50–100 ft. wide. An "arch pillar" is located a short distance up the stope from a level, or may form one side of an arch of a drift. When some distance from levels (half-way, or more, up the stopes), they are called "wall pillars." Arch and wall pillars are left at 50–75 ft. along the stopes, and are roughly 6–16 ft. square. At top of stopes, there is left, on completion of cutting-out stoping, a long, thin pillar pierced by a number of break-throughs, forming the floor of the level above, and known as "floor," or "chain" pillar; width, 8–10 ft. up to 20 ft. Sometimes, instead of chain pillars, 8-ft. arch pillars are left directly above drifts forming levels, and continuous, except for shoot openings or "mill-holes"; and again, such a pillar may be built of waste. At the end of every two or three stopes, each 100 ft. long, pillars ("dead ends") are left, extending from level to level transversely with the stopes; they are a combination of arch, wall and floor pillars, all run together, forming a long pillar which aids materially in securing the workings.

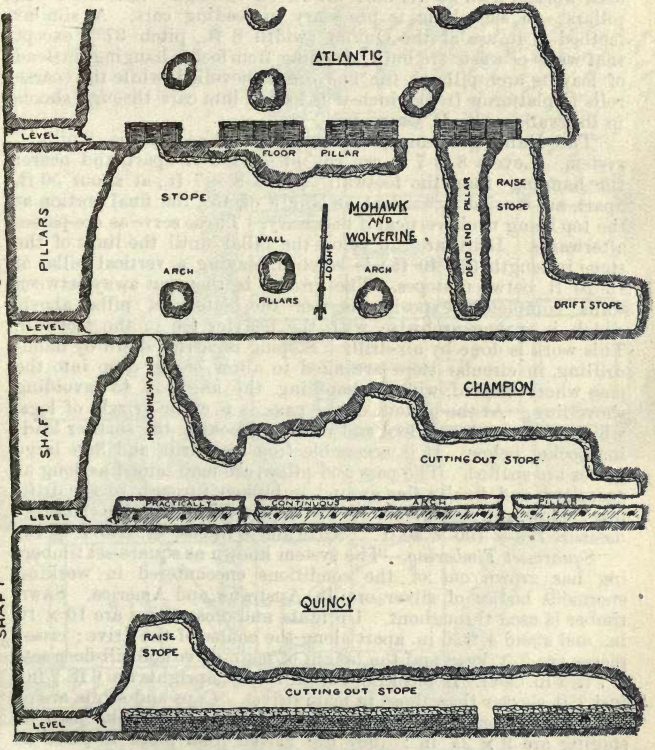


FIG. 149.—PILLARING, LAKE COPPER MINES.

Typical methods of arranging stopes are shown in Fig. 149. In the Atlantic (width 15 ft., pitch 54°) stulls covered with lagging check runs of broken rock, serve as footing for men and drills, and permit storage of considerable waste; ore is shovelled into cars.

At the Mohawk and Wolverine (width 8–20 ft., pitch 38°), broken ore runs down the steeper portions, but elsewhere needs shovelling. In the Champion (width 15–25 ft., pitch 60°), a continuous arch pillar holds broken ore in bottom of stope, to which point it has been worked from above, until drawn off through "mill-holes" in pillars; no shovelling is necessary in loading cars. A similar method is in use at the Quincy (width 8 ft., pitch 37°), except that walls of waste are built, reaching from foot to hanging, instead of leaving arch pillars; the fine ore is shovelled, while the coarse rolls to platforms from which it is loaded into cars through shoots in the walls. (W. R. Crane.)

The Haile gold-mine, Carolina, is worked by the "pillar" system. Levels 8 × 7 ft. are run at 70–100 ft. apart, and nearer the hanging- than the footwall. Rises 8 × 7 ft., at about 50 ft. apart, are carried forward at an angle of 45°, the final portion at the top being made vertical if necessary. These serve as ore-passes afterwards. Drifts are run below the pillar until the limit of the stope in length (30–40 ft.) is reached, leaving a vertical pillar of 15–20 ft. between stopes. The ground is then cut away between walls, completely exposing as roof the bottom of pillar above, which is sprung archwise, with the heavier toe in the footwall. This work is done by air-drills. Stoping is carried down by hand-drilling, in circular steps arranged to allow ore to drop into the pass where blasted, without handling, the angle of 45° avoiding shovelling. At the bottom of the pass, is a crude grizzly of logs, which holds back boulders, and prevents choking the smaller loading-pocket below. It is accessible from the drift, and here large pieces are spalled. The pass and pillar are maintained as long as necessary, and finally the ground is broken through to the drift-level below, and shovelled. No timber is used, though the chambers measure 100 × 100 × 40 ft. (Nitze and Wilkens, Tr. Am. I. M.E.)

Square-set Timbering.—The system known as square-set timbering has grown out of the conditions encountered in working enormous bodies of silver ore in Australia and America. Sawn timber is used throughout. Uprights and cross-pieces are 10 × 10 in., and stand 4 ft. 6 in. apart along the course of the drive; cross-pieces are 5 ft. long, and the height of main-drive and sill-floor sets is 7 ft. 2 in. clear. In blocking-out the stopes, uprights are 6 ft. 2 in., just 1 ft. shorter than those in main drives. Caps and struts are of same dimensions and timber as the sill floor. Planks used as staging are 9 × 2½ in.; they are moved from place to place as required, and upon them the men stand when working in the stopes and faces. A stope resembles a huge chamber fitted with scaffolding from floor to ceiling. The atmosphere is cool and pure, and there is no dust. Stage is added to stage, as the stoping requires, and ladders lead from one floor to another. Accessibility of the face is a great advantage. If, whilst driving, a patch of low-grade ore is met with, it can be enriched by taking a higher class from another face, and so on. If signs of weakening become apparent in the timbers, 4 or more uprights are lined with planks, and waste material is shot in from above, whereby a solid support is at once formed; if crushing is noticed, it is possible to get into the stope,

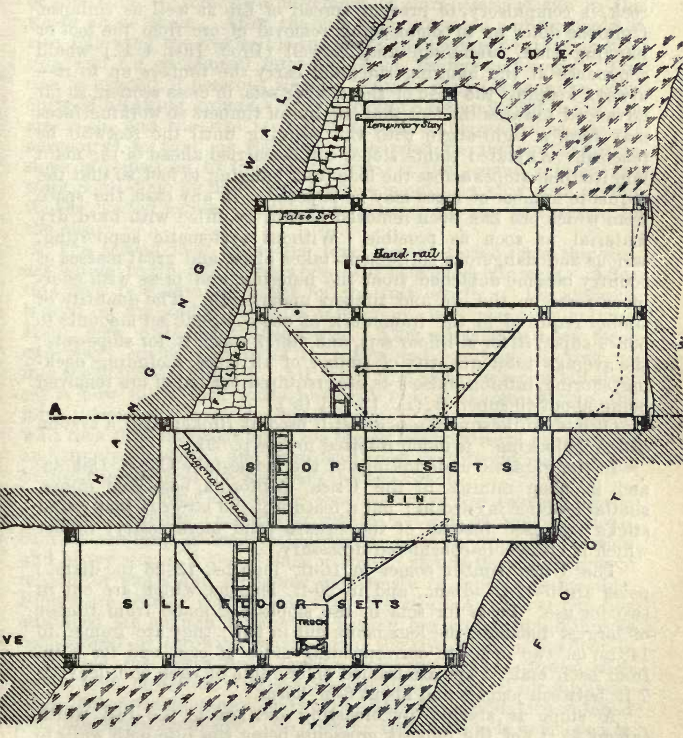


FIG. 150.—SQUARE-SET TIMBERING, BROKEN HILL.

break down ore, and at once relieve the weight. A prominent example (Fig. 150) of the system is the Broken Hill mine, Australia, where ore-bodies range from 15 to 316 ft. wide, and average about 105 ft. Where the pressure is light, single timbering suffices; under heavy pressures, false or double sets, with diagonal struts, are necessary; and in extreme cases, solid timber bulkheads have been built. The timber preferred is Oregon pine. But no timbering can always withstand the pressure arising from the excavation of such enormous ore-bodies; and filling the stopes at certain points from hanging to foot, with hard dry material such as slag or waste rock, is compulsory, to prevent spread of fire as well as collapse. Opinions differ as to commencing removal of ore from the foot or hanging side. Jamieson and Howell (Proc. Inst. C.E.) would commence at the hanging side, and carry the timbers up to it—always keeping the base of the timber sets, in cross section, so far advanced towards the foot that the line of timbers on working faces may form a right-angle with the hanging until the footwall be reached. At stated points, stopes can be carried ahead of the main longitudinal stopes across the lode from hanging to foot, so that the requisite number of faces may be exposed. In any case, the space from which ore has been removed should be filled with hard dry material as soon as possible. Without systematic supporting, serious and dangerous settlement takes place, and great masses of country become detached from the hanging, and press with enormous force on the ore and timbers underneath. The quantity of timber required in the framework of one complete set amounts to 533·4 super. ft. for sill-floor sets, and 405·7 super. ft. for stope-sets; the average total quantity of timber of all sizes, including decking, shoring, lathing, false sets, etc., required per ton of ore removed being about 40 super. ft. (i.e. 12 × 1 in.).

Of late, while square sets are still used at Broken Hill, a system called "blocking" is much resorted to (see p. 391).

In the gigantic undertakings of the Homestake Co., S. Dakota, and in deep mining in the Utica, California, both gold mines, similar practice is current; but a feature of the latter is that round sticks are used instead of the square (and more costly) timber which is elsewhere considered necessary.

The Utica timber comes in 16-ft. lengths, 12–26 in. diam.; poles are 6–12 in. diam., and in 20-ft. lengths, which are cut in two for use. Posts for sets in the stopes are made from timber of largest diameter, the logs being cut in two; they are framed to 14 in. on two opposite corresponding sides of each end for 3 in. from each end. Caps are framed on one side of each end, leaving 7 ft. between joggles and about 6-in. horns.

A stope is started by breasting out ore to the full width (about 35 ft.) of the deposit, crosscuts being run into both walls to be sure that all ore has been taken; the opening is then timbered with 8-ft. stope-sets. If the ground is solid, no timbering is done until the whole mass of rock covering area of stope has been removed. Posts are then set in solid rock, with 6-in. spreaders and 12-in. round brace-sprags between posts at bottom. A floor is then laid over the spreaders. If the rock is loose or soft, one set is put in at a time, as fast as room is made. In soft ground, heavy sills are laid to give a solid foundation for posts. Sills are not an advantage in working up under an old stope. Good floors laid across spreaders, even though they have been in place so long as to be badly decayed, are more serviceable than sills. Sills are seldom in place when reached, and have to be caught up securely, or are liable, by their own movement, to start a serious run in the waste above them. After the sill floor is opened and timbered, a rise following the footwall is run up to the level above for ventilation, and for economical introduction of timber and waste. The rise is located, if possible, where a seam of gouge lessens the difficulty of breaking ground. If the rock is hard and solid, machines are used, and the rise is timbered with full-sized stope-sets (if necessary), so that, as the stope is carried up, the timber of the stope is joined on to that of the rise; if the ground is loose, the work is all done by hand, and the rise is built up solid with round timber, halved together at the ends, making the rise 4 ft. square in the clear.

The second floor above the level is now started from the rise, ore falling to the sill floors, where it is shovelled into cars. After this floor is excavated, a set for the full length of the stope is lagged on top and sides with half-round slabs, made by halving 12-in. logs, 8 ft. long, lengthwise. This set is kept open for a gangway; along the line of it, passes and manways, leading up into the stope, are started. All remaining space to top of sill-floor set is now filled with waste, obtained from 3 sources within the mine: from vein rock, by hand sorting; from cross-cuts run in wall rock from different floors for the double purpose of prospecting and supplying waste; and from dead work in different parts, being brought in through the rise from the level above.

From this time the passes and manways are carried up by means of cribbing (p. 409) to within one set of back of stope. Similar cribbing is used to help support heavy ground. In wide and heavy stopes, a row of such cribbing extends the full length of middle of stope, and is stowed with waste as rapidly as possible. Floors are started successively from the rise. As soon as one has advanced far enough to prevent mixing ore and waste, filling is commenced by throwing waste down the rise. When the floor is completed, the lower floor, remote from the rise, is stowed with waste from crosscuts. To supply sufficient material for this purpose, the crosscuts, which start with small dimensions, are widened out into large chambers. No opening in the stope is kept open to a greater vertical height than 16 ft. When a level is being worked, a large mass of vein is left in place, opposite the shaft, until the last thing before the level is abandoned. When the vein is badly crushed and broken up, poling is run out over a cap or sprag—depending on direction of work: whether with or across the vein—to support the ground until timber is in place.

The system has most of the merits of the square-set, while being cheaper and more flexible. The framing is far more simple, and much of it is done in the mine, while waiting for partially-framed timber to come from surface. Caps are only framed on one side before being sent below, for several reasons. As logs are not always exactly 16 ft. long, nor cut square on the end, it is necessary to leave a space of about 2 in. between the ends of caps to allow for this irregularity. This is brought about by framing the ends of posts to 14 in., and caps so that they measure 7 ft. between joggles, which, if the log was exactly 16 ft., would leave 6-in. horns. If side-pressure is great, this space may be reduced the full 2-in., thus lessening the hitch formed by the two adjacent caps (if framed) to 12 instead of 14 in. The ends of the posts, not being exactly square, would raise one cap higher than another. Caps, being made from logs, have the natural taper of the tree, and ends are not of equal dimensions, the difference in diameter being often as much as 6 in. As the posts in the stope do not settle equally, use is made of this irregularity to keep the line of caps horizontal: the end of cap of largest diameter is placed on lowest post. When the next set above is put in, the joggle over the higher post is cut deeper than the one over the lower post of the under set. Thus the size to which the end of cap is framed depends on circumstances, and can only be done to advantage in the stope.

The general dip of the ore-body is nearly vertical, but in places where it bulges, the wall rock sometimes has a pitch of 45°. Timbering is adapted to this by first leaving a horn on the cap extending over the post nearest the footwall. When this horn must be so long as to greatly increase the weight of cap, a short butt cap is used instead: framed as an ordinary cap on one end; on the other, bevelled to fit the wall rock. Length of butt cap is increased as the wall recedes, until the distance between post and wall rock becomes great enough for a full set. A hitch is then cut in the footwall for another post. While caps on the footwall side have been lengthened, those on the hanging have been correspondingly shortened. Full sets are also omitted on one as they are added on the other, and thus adaptation is made to a vein having considerable width, and but small dip. The system proves satisfactory, despite constant settling of the ground, and a low-grade ore-body which demands economical working.

The conditions at Chillagoe, N. Queensland, demand a style of square-set timbering which can utilise stunted and twisted local timber without a saw-mill, as follows. The sets are made up of the usual members—posts are cut as at A (Fig. 151), caps and stretchers as at B, from rough-hewn logs having a clear length of 6 ft. 6 in., and diam. of 10 in. (min.). The heavier logs are selected for posts. All are cut to shape and dimensions by using a mitre-box and angle and square templates, and such rough logs are both cheaper and stronger than sawn timber, weight for weight. At

Chillagoe, logs cost 3–4*d.* and converting 2*d.* per ft. run. (T. J. Greenway.)

At Rossland, B.C., also, round logs are similarly used, costing 4*d.* per ft. run; and the total cost per set fixed is 21*s.* 8*d.* when hand-shaped, and 1*s.* less if machine-shaped.

Where the chief crush is downwards, sets should be framed with the horns of the posts butting; where sideways, caps should

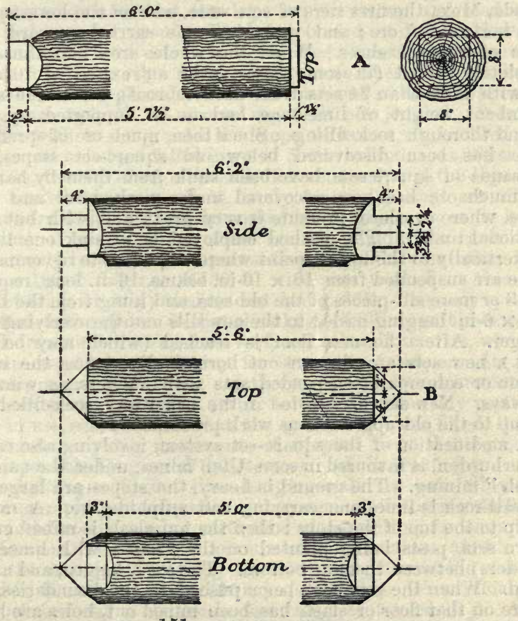


FIG. 151.—SHAPING ROUND LOGS.

butt. This latter condition exists at Bingham, Utah, and the system adopted there, and since copied elsewhere, is as follows. Caps are 10 × 10 in., and posts are 10 in. wide in the direction of the "girts" (ties, or braces), and 9 in. wide cap-ways, thus saving 1 in. in cross-section. Posts have a bottom and a top end, being "bald" at bottom and having only a 1-in. horn on top. Hence the top mortice made up by assembling caps and girts differs from the bottom mortice, and there are top and bottom sides to caps and girts. Naturally, it is necessary to have a tenon at top end of post on which to rest caps and girts. The bottom, resting on caps and girts, does not need to be framed; but if top and bottom ends of posts were similarly framed with horns, it would avoid special framing of a cap only on top side of girt: the latter would then be a plain 6 × 10 in. timber, having to resist only side movement of caps. (C. T. Rice.)

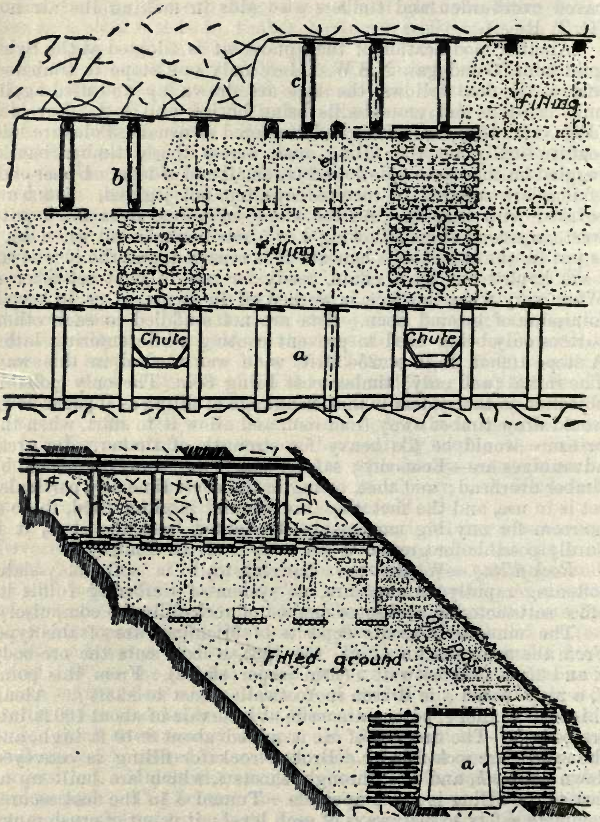
In the large soft bodies of lead ore (in limestone) of the Sierra Mojada, Mex., the first tiers of sets were laid on the lowest profitable horizon of ore; and timbering was carried upward in a series of vertical slices. When the whole area had thus been completely mined (in some cases, over an extent of 10 acres, and with more than 24 sets from floor to roof), the whole superincumbent weight of limestone had to be supported by careful and thorough rock-filling. Since then, much ore of profitable grade has been discovered below old square-set stopes, and thousands of square sets have been sunk from them by hangers, and much ore has been recovered under worked-out and filled stopes, where overhead pressure is very great, and with but little additional timber. The method employed is to sink one line of sets vertically to the lowest point where stoping is to re-commence. These are suspended from 10 × 10-in. beams, 16 ft. long, reaching over 3 or more sill-pieces of the old sets, and hung from the beams by 3 × 6-in. lagging nailed to the new sills and the overlying 16-ft. stringer. After the new floor is reached (which may be 6 or 60 ft.), new sets are thrown out horizontally, using the initial column or columns of suspended sets with hangers, as winzes or manways. New sets are started in the usual way, rock-filled, and tied up to the old upper sets as work proceeds.

A modification of the square-set system, involving also caving of overburden, is favoured in some Utah mines, under the name of "slash" mining. The ground is heavy, the stopes are large, and the wall-rock is limestone carrying soft sulphide ore. A raise is put up to the top of the stope; then the top slash is mined out by square sets, posts being planted on the ground with braces (or spreaders) between them at bottom. The set is lagged, and a floor is laid. When the sets have been pushed to the boundaries, and the ore on that floor or slash has been mined out, holes are bored into the posts and loaded with dynamite; on blasting, the overburden caves down upon the floor. If the ground becomes heavy, the floors can be mined in sections, each caved as soon as its ore is mined out. Miners then drop down a set in the raise, and begin another slash. The floor of the slash above is caught up by the square sets, and waste is kept from mixing with ore. No timber is recovered and no back of ore is caved. All the ore is recovered, small timbers can be used, and the method is safe; but a bottom of ore must be taken out on each floor, and costs of breaking are high. Raises must be driven in solid rock instead of carried up in the sets as in ordinary square-set mining. The stopes are very hot, and the air is poor, because there are no raises to levels above to promote ventilation. Decaying timbers, which follow down with caved overburden, soon foul what little air gets in, and crushing of caved overburden and timbers also aids in making the air hot. (C. T. Rice.)

Another modification of the square set is adopted at the Sybil gold mine, Gundagai, N.S.W. Here only one stope is timbered, and, as the next follows, the legs are drawn up and used again, only bed-logs and ground-sills being buried and lost. Levels *a* (Figs. 152, 153) are driven and timbered as usual. Poles are laid longitudinally on the filling, and, across these, timber baulks joggled on the upper side to receive small end of legs. Upper ends of legs are squared to receive caps, but not joggled. Sets *b* are covered with lagging. As the stope advances, it is filled with waste, a second stope following 4 or 5 sets behind; but when set *c* is put in, the cap of set *e* beneath it is removed, and the legs of set *e* are hauled out by a chain, ready for use in next set after *c*. When very wide, the face is worked off in strips, so as to keep a minimum of ground open. Sets are not studdled to each other, battens only being used to prevent canting while entering laths. A stope 100 ft. high × 25–30 ft. wide was worked in this way. The initial (and only) timber cost being 35*l*. The only possible objection to the system is that shrinkage of filling as it packs down would draw timber away from roof, and allow it to start, when the pressure would be too heavy for strength of timber. Its great advantages are—Economy; safety, men being always protected by timber overhead; and that, owing to the short time any particular set is in use, and the fact that the ground is close filled, there is no room for any big movement—if adopted from the start, it is hardly possible for ground to get away. (J. R. Godfrey.)

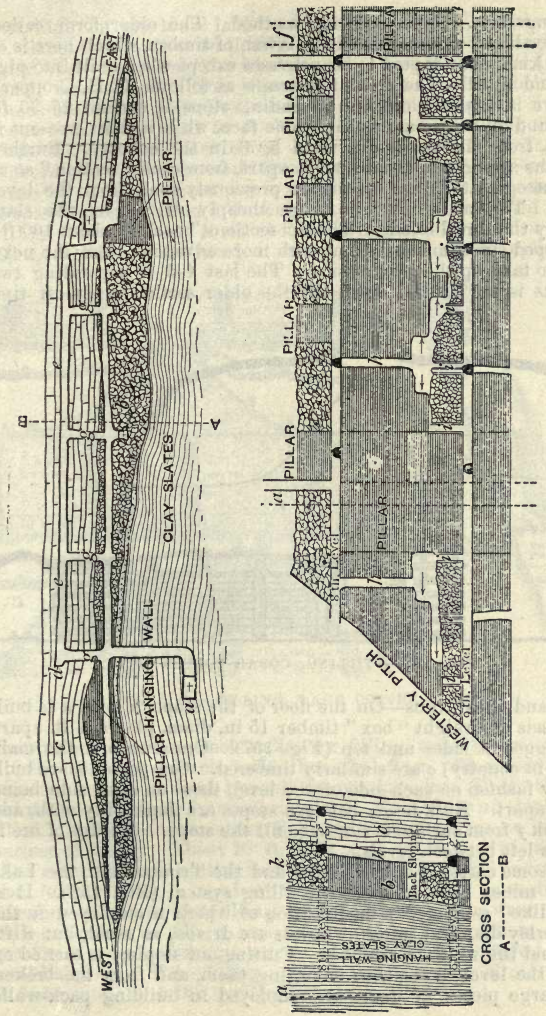
Rock-filling.—Where the hanging-wall is soft clay-slate, softening rapidly on exposure, no system of timbering fulfils its office satisfactorily, and some method of rock-filling is compulsory.

The immense hematite deposits of Michigan are of this type. From the main shaft *a* (Figs. 154–156), a drift cuts the ore-body *b*, and 25 ft. into footwall *c* (firm jasper slates). From this point *d*, a rock-tunnel *e* is driven in footwall *c* east to shaft *f*. Along this rock-tunnel *e*, ports *g* are made at intervals of about 100 ft. into ore-body *b*. The first cut of ore is mined about 8–10 ft. high, and the spaces are rock-filled *k*. Broken rock for filling is conveyed down winzes *h*, and ore through shoots *i*, which are built up as fast as the filling is made upwards. Tunnel *e* in the foot secures complete safety to mainways of each level; it is out of crush range in any event. Winzes *h* are also ventilators. Ore-body *b* is attacked on double face at each port *g* from main rock-level or tunnel *e* in the foot. Mining by sections upwards is simply a repetition of the first 8–10 ft., rock-filling following mining as rapidly as room will permit. Where previously timber and timbering cost 1*s.* 7½*d.* per ton of ore mined, rock-filling costs 7*d.* a ton, besides affording a permanent support, not liable to decay, and not requiring renewal.



FIGS. 152, 153.—SQUARE SETS, SYBIL MINE, N.S.W.

The Great Cobar copper mine, N. S. Wales, where bunches of ore 300 ft. long, 50 ft. wide, and of unknown depth, occur in well-



FIGS. 154, 155, 156.—ROCK FILLING, MICHIGAN IRON MINES.

standing slate, adopts a filling method. The older form, called "barricading," was unnecessarily lavish of timber, which here is of a poor kind, not adapted to bear stresses except when built into pig-sties and filled. The present system is as follows (C. H. Cropper). A drive is made in ore, and a leading stope is carried 25–30 ft. high, and arch-wise across the whole face, while a short crosscut is driven from the shaft to a main level in the country. Further crosscuts are driven, about 100 ft. apart, from each "section" so as to come opposite 6 × 4 ft. winzes previously sunk from the level above. The main level is more cheaply driven in the slate country than in the lode. Three "sections," each of about 100 ft., are stoped simultaneously, but each more advanced than the next, so as to take up the progression. The last 4–5 ft. separating two sections is not broken down till the older section is almost tim-

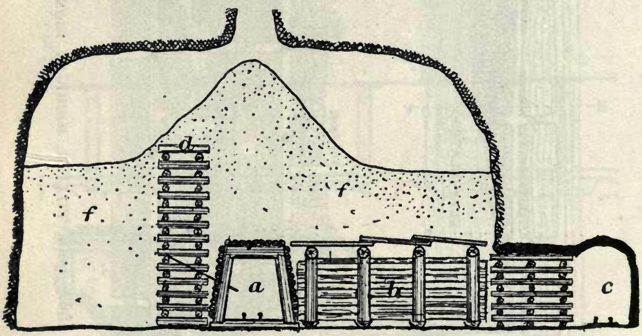


FIG. 157.—FILLING, COBAR COPPER MINE.

bered and filled, thus—On the floor of the stope, a level *a* is built up of sets of straight "box" timber 15 in. diam. placed 4 ft. apart, and lagged at sides and top (Fig. 157). Crosscuts *b* to the main level (in country) *c* are similarly timbered. Ore passes *d* are built pig-sty fashion on each side of the level, those on same side being 24 ft. apart. Bulk-heads between stopes are done away with, and mullock *f* from surface is run in to fill the stope. A ledge of ore is always left between levels.

In some cases, as at the Baltic and the Trimountain, the Lake copper mines have adopted the filling system (Fig. 158). It is not unlike "rabitage," a modification of which is employed in the Kimberley diamond mines. Levels are driven, as usual, but 8 ft. high and the width of the lode. Cutting-out stoping is carried on along the level drifts, thus enlarging them, and from the broken rock large pieces of waste are employed in building pack-walls

8 ft. high, on which are laid "wall-pieces," or 14-ft. timbers 18–30 in. diam., from wall to wall, followed by plank or pole lagging to carry waste for filling the whole space. Levels are thus permanently established. From the footwall side, "milling-holes" or passes are built upward as waste is provided; they are circular, about 5 ft. diam., and 50 ft. deep. Cutting-out stoping ensues. Then the drills are mounted on broken rock piles 15–25 ft. high. Pickers and trammers work from the rear face of pile—stowing waste in the stopes, and passing ore down mill-holes to cars; 25–45% as broken is waste. Cutting-out stoping is continued up to within about 20 ft. of level above, leaving a 20-ft. floor or chain pillar, which is penetrated at intervals by break-throughs. As

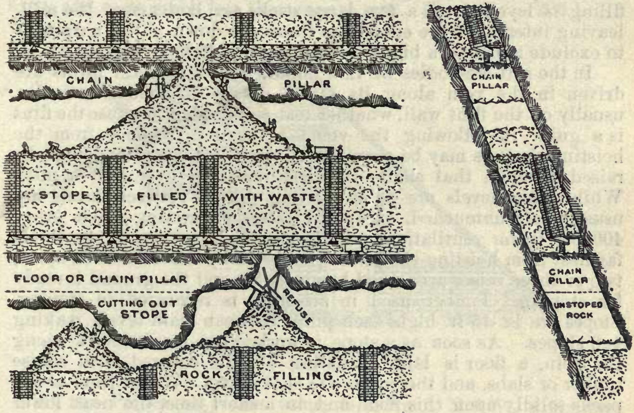


FIG. 158.—FILLING, LAKE COPPER MINES.

the waste broken does not suffice for filling, some is drawn from upper into lower levels, and thus the floor pillars are safely removed. This is done by making openings into filled stopes at points directly above break-throughs in floor pillar of stope to be filled. Drills are then mounted under ends of pillars, on inclined surface of filling. About 10 ft. wide × 15–20 ft. long of pillars is thus removed, as in cutting-out stoping; then the drill is reversed, and further holes bring down the ends of pillars, enlarging the opening through which filling flows. Ore as broken from pillars falls on surface of filling, and gravitates down to pickers. Costs per ton of ore mined are quoted at—mining, 17·9*d.*; picking, 6·9*d.*; wall-building, ·81*d.*; filling, ·17*d.*; tramming, 8·7*d.*; hoisting, 2·54*d.*; total, 3*s.* 1·02*d.* (W. R. Crane.)

When the ore is hard enough to stand over the width of the vein, it is taken down in an overhead stope running from hanging to foot for any desirable length, and for a height of say 12 ft. A timber-drift is then built along floor of stope, and the balance of the stope is packed with waste sent down from surface through winzes previously sunk or raised at intervals of about 50 ft. A "mill" or shoot at every 50 ft. runs the ore down; it is about 4 ft. sq. inside, and is built of round, rough hardwood sticks, spaces filled with pieces of plank, and the inside lined with hardwood planks spiked to side timbers, and easily replaced when worn. The packing is levelled as close to back of stope as convenient, and is planked over to keep ore from mixing with filling. The "mill" is carried up before filling as high as this is to go; and when the filling is levelled off, a few large sticks are laid across the mill, leaving intervals large enough to throw ore down, but so narrow as to exclude a man or a block of ore that would choke the outlet.

In the soft ore-bodies at Low Moor, Virginia, main levels are driven in the vein along its strike, 60–80 ft. apart vertically, usually on the flint wall, whether foot or hanging (because the flint is a guide in following the vein), to such a distance from the hoisting-shaft as may be required to reach all ore intended to be raised through that shaft—in some instances over half a mile. While main levels are being driven, pillars between levels are usually left untouched, except by raises connecting levels every 400–600 ft. for ventilation; when completed, portions of two levels farthest from hoisting-shafts are connected by raises 60–75 ft. apart, two or three raises are joined by air-drifts, and the ground is ready for stoping. Timber used in air-drifts is recovered in stoping. Stopes are 12–15 ft. high, each pillar between main levels making 4–6 stopes. As soon as a stope is worked out for 40–60 ft. along the vein, a floor is laid, consisting of sills covered with refuse timber or slabs, and the props are shot down. Waste from above packs solidly upon this floor, and, in a short time, the next lower stope can be worked, using the floor previously laid as a roof to hold waste from the ore. A stope 40–60 ft. long, measured along the vein, is begun in the drifts, by first mining ore above the drift-timbers till the floor of the next stope above is reached, and setting props. The face of the ore for the length of the stope is then mined back to the opposite wall. Ore is dumped into shoots, drawn from them into cars on main levels, and hauled by mules to surface or to hoisting-shaft.

When the vein is 12 ft. or less thick, so that a single prop will reach from wall to wall, a stope-drift is driven a short distance above the main level, parallel with it, and connected by shoots at intervals of about 50 ft. Ore is then stoped from this drift to the next upper main level, props being placed from wall to wall; these are eventually shot down, and the waste is caught by a horizontal floor, serving as a roof for the next lower stope, as before. In this way, the timber for a number of floors is saved ; but the modification is only of advantage, when the vein is narrow enough to permit a single prop to reach from wall to wall, and where the hanging is fairly good. All the ore is mined, no filling is required, and the work is comparatively safe. In some instances, means must be taken to exclude surface water from the breaks which run up to daylight when the country sinks to fill cavities.

Pig-stying.—On Charters Towers goldfield, Queensland, where the country is granite, and the quartz veins often run 20–30 ft. wide, the "pig-sty" method of timbering is employed. The "sties" are built up, like square sets, in the form of pillars until the hanging-wall is reached. Waste is filled into the sties, and stamped hard. This enables cheap timber of poor quality to be used, and costs much less than square sets. (See also p. 296.)

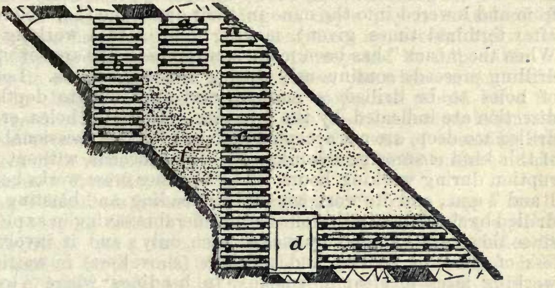


FIG. 159.—PIG-STYING.

At the Prince of Wales mine, Gundagai, N.S.W., a similar system (Fig. 159) is in vogue. Sties are started from hitches cut in the solid foot, and continued right through to the hanging, carrying the back as they go; they are built systematically in rows. The reef is 30 ft. wide, between rotten talcose slate walls, very heavy and treacherous. Stack *a* is built off filling ; *b*, from a hitch, is holding the back while stoping; *c* is finished to the hanging; *d*, level; *e*, bulkhead; *f*, ore-pass.

Payment for Stoping.—Practice with regard to payment for stoping ore varies very widely. It may be all on contract, at per truck, per ton, per cub. yd., per sq. fath., or per lin. ft. ; or it may be partly on contract (at per ft. drilled, or per hole of specified depth) and partly on wages (setting out, loading, and firing holes); or even all on wages, though that is rare. Supplies may be all furnished by the employer, or all by the contractor, or partially by each.

Many Rand mines contract at per sq. fath., the surveyor computing area of stoped ground: prices average about 66*s.* 8*d.*, contractors paying 3*s.* a day each for their Kaffirs, and for stores consumed. A contractor (miner) who drills and fires 40–45 ft. of holes per diem, with 2 machines, breaking 40 sq. fath. per month, will earn about 50*l.* per month. A generally understood day's work is 4 holes 5½ ft. deep per machine. This may be finished by 2.30 p.m., and the interval till firing time (5 p.m.) is idled—an obvious loss to both employer and employed, and an outcome of trades-unionism. (T. L. Carter.)

A good example of payment per ft. of hole drilled is that followed at the Center Star mine, Rossland. Underground work is carried on by two 8-hour shifts, arranged as follows: morning shift, 7 a.m. to 4 p.m., with an interval of 1 hr. for dinner; afternoon shift, 4 p.m. to 1 a.m., with 1 hr. for supper. Men are raised from and lowered into the mine in their own time (i.e. before and after terminal times given), making 8 hr. actual working time. When the "back" has been made secure, machines are set up, and drilling proceeds continuously during two 8-hr. shifts. Location of holes to be drilled is marked, and approximate depth and direction are indicated, by the foreman. Misplaced holes, or those drilled too deep, are not accepted and paid for; an occasional check of this kind ensures good work. Drilling proceeds without interruption during working hours. The blasting crew works between 1 and 7 a.m., and its work consists in loading and blasting holes drilled by the miners: this effects considerable saving in explosives, since these are handled by picked men only; and it involves no loss of time by miners and muckers (shovellers) in waiting for working faces to clear of smoke. In headings where a certain number of holes have to be drilled before the whole set or "round" can be blasted, a difficulty in making sure that contractors finish before blasting time (that they may not have to lose working time during blasting, and thus may be kept continuously employed), is met, either (*a*) by increasing or decreasing the depth of holes to be drilled; or (*b*) by having one or two spare headings or stoping breasts in which contractors can utilise their extra time —the latter is to be preferred, enabling depth of drill-holes to be determined on other grounds than that of time required to drill them. Within limits, the number of machines in any one stope can be varied at will; and there is no necessity to keep separate the work done by each set of contractors. The same set of contractors can be employed in different headings or stopes without any confusion in measuring work. But the loss of possible drilling time (only 2 shifts per diem instead of 3) may be objectionable in some cases. As compared with wages, the contract (hole) system showed an advantage of nearly 50% in cost for stoping; and 25% in cost and over 50% in speed for driving and crosscutting. (C. A. Davis.)

There is no mine where the physical conditions do not admit of some form of payment by result: the political conditions are another matter.

Machine v. *Hand Stoping.*—The relative advantages of machine and hand stoping are much debated. Every mine manager must judge for himself—there is no law. The problem is complicated by wages rates, costs of power and supplies, labour available, output, supervision, fitting-shop facilities, character of ore-body, lay-out of mine, and many minor considerations. Under some circumstances, machines are immensely superior; under others, hand labour is equally so.

Thus, at the Alaska Treadwell, where 75% of the ore comes from open-cast pits, a $3\frac{1}{4}$-in. drill on tripod averages 36·35 ft. per 10-hr. shift (holes 12 ft. deep), and breaks 70 tons of ore; when the pits were smaller, a tonnage of 150–200 was reached.

On the Rand, machine stoping on day's pay is considerably cheaper than hand stoping. Now that there are several light and handy stoping drills on the market, it may not be impossible to entirely do away with hand stoping; then one white miner could be put in charge of 10 machines in a stope, and would have no difficulty in efficiently supervising the 10–15 coloured labourers required to run them. By this means, the cost of breaking ore could easily be reduced to 3*s.* per ton as against 5*s.* 6*d.* under present conditions. (Ross Browne.)

The Geldenhuis Estate has been (Sept. '07) running 4 small (120 lb.) stoping drills, by 1 skilled miner and 10 Kaffirs, against hand drilling. As a rule, the 4 machines drill 40 3-ft. holes in the morning shift, and 32 3-ft. holes in the afternoon shift, making 72 3-ft. holes in 16 hr., equal to 9 holes per machine per shift. As the average work for one native is one 3-ft. hole per shift, it would require 72 natives to do the work of 4 machines. Comparative costs per shift are:—

	Machine. £	*s.*	*d.*	Hand. £	*s.*	*d.*
Air (including compressor charges and depreciation), 4 machines at 3*s.* 3·1*d.* per shift		13	0·40	..		
White labour—1 skilled miner	1	0	0	1	0	0
Native labour—						
10 natives at 3*s.* per shift..	1	10	0	..		
36 hammer boys and 1 boss boy, at 3*s.* per shift	..			5	11	0
Oil—4 machines, at 1·41*d.*			5·64	..		
Hoses—4 machines, at 3·51*d.*		1	2·04	..		
Maintenance, at 51*s.* 4*d.* per machine per month, and depreciation (cost of drill, 28*l.*; life, say 4 years)		4	10·16	..		
Totals	£3	9	6·24	6	11	0

The Rand rock is much too hard for economical hand drilling, even by Kaffirs. Compared with California, Rand mines employed (in 1904) about 3 times the number of men (white and coloured) per ton of ore milled. This is not fair comparison, since it includes men engaged in sorting and in workshops: in California, there is no sorting, and much of the repair work is sent to custom shops in neighbouring towns. But making due allowance, Rand mines employ 2·5 times the number of men per ton of ore. Since resorting to hand stoping with Chinese labour, they employ 4 times the number, which means great excess of labour or great inefficiency.

Mining Costs.

In comparing costs of mining, essential conditions to be taken into account are—Size and character of the ore deposit. Method of mining, proportion of ore-body extracted, system of breaking, lay-out of mine, method of handling ore, drainage, etc. Depth and longitudinal extent of workings: cost of hoisting and pumping, increase in depth; cost of tramming increases with longitudinal or lateral direction. Character and amount of development work. Rates of wages. Costs of all supplies, including power. Tonnages mined, sorted, and waste. And, last but not least, mining laws and administration. A number of examples are given below, quoted from various sources, and all reduced to the short ton.

Alaska.—Treadwell mine: largely open cast, no timbering, no pumping, immense tonnage, 1 drill breaks 35 t. per 10 hr., wages 12*s*. 6*d*. and keep—3*s*. 9*d*. per ton.

Mexican mine: 4*s*. 9*d*. Ready bullion, 4*s*. 11*d*.

California.—Central Eureka: 43,545 tons cost 8*s*. 7*d*.

Melones: on Mother Lode, in slate and greenstone, stopes 4–30 ft. wide, wages 12*s*. 6*d*., output 7000–8000 t. per month —mine labour and supplies, 1*s*. 9*d*.; tramming, 3½*d*., total, 2*s*. 0½*d*. (Langford.)

Utica: on Mother Lode; 2 10-hr. shifts of 50 men furnish 300 t. daily, costing, for labour alone—24 miners (12*s*. 6*d*.), 1*s*.; 24 helpers (10*s*. 5*d*.), 10*d*.; 30 shovellers (10*s*. 5*d*.), 1*s*. 0½*d*.; 12 trammers (10*s*. 5*d*.), 5*d*.; 10 timbermen (12*s*. 6*d*.), 5*d*.; total, 3*s*. 8½*d*. (Collier.)

Yellow Aster: 500 t. per diem, wages 12–14*s*.—mining, 3*s*. 1*d*.; timbering, 1*s*.; explosives, etc., 9*d*.; total, 4*s*. 10*d*. (Barton.)

Colorado.—Camp Bird: stopes 6–7 ft. wide, adit working, very little timbering, only 40% of ore broken taken to mill, wages 12*s*. 6*d*.–19*s*.

Cripple Creek: moderate depth, variable water, much timbering, wages 14*s*.—on 230,000 t. raised, costs are 15*s*. 5*d*.–17*s*. 5*d*. (including 2*s*. 10*d*.–5*s*. 4*d*. for development). (Finlay.)

Idaho.—Bunker Hill and Sullivan: large bodies, adit mining, water power, abundant timber, wages 14*s.* 6*d.*, electric haulage at $3\frac{1}{2}$*d.* per t.—on 260,000 t., costs are 8*s.* $7\frac{1}{2}$*d.*, wages accounting for 5*s.* $4\frac{1}{2}$*d.*

Lake Superior.—Copper mines on very large scale, worked by inclines, very little timbering.

Atlantic: on 450,000 t. milled, total mining costs are 3*s.* 9*d.* -4*s.* 6*d.*; stoping alone, 16*s.* 7*d.*-17*s.* 4*d.* per fath.—10-12*d.* per t.

Baltic: on 490,000 t. milled, total mining costs are 4*s.* 8*d.*

Osceola: on 900,000 t. hoisted, total mining costs, 4*s.* 11*d.*

Tamarack: on 630,000-660,000 t. milled, total mining costs, 9*s.* 7*d.*-10*s.* 2*d.*

Wolverine: on 190,000 t. milled, total mining costs, 5*s.* 6*d.* per t. hoisted.

Missouri.—Lead deposits in limestone, 300-500 ft. deep, no timbering, 200-2000 gal. water per min., stopes 80 × 60 ft., wages 8*s.* 4*d.*, coal 9*s.* 2*d.* per t.—on 1000 t. daily, total mining costs about 4*s.* 6*d.*

Zinc mines (Joplin), flat beds in limestone at 150-250 ft., practically no timbering or pumping, lode extremely hard chert requiring heaviest machine drills, wages 8*s.* 4*d.*-9*s.* 4*d.*, outputs 75-100 t. per 10 hr.—total mining costs, 2*s.* 4*d.*-3*s.* 11*d.*

Montana.—Copper mines on veins 100 ft. wide, 10-20 ft. stopes, outputs of 100,000-1,400,000 t. per ann.

Anaconda: 630,000 t., total mining costs 16*s.* 4*d.*, 10*s.* 4*d.* being labour.

Syndicate: 820,000 t., total mining costs 14*s.* 4*d.*, 9*s.* 4*d.* being labour.

South Dakota.—Homestake gold mines group, immense bodies of mineralised schist 300-500 ft. wide, partially open-cast working, mine depth 1100 ft., heavy timbering, wages 14*s.* 6*d.*, output $1\frac{1}{4}$ million tons—total mining costs 8*s.* 6*d.*-9*s.*

Tennessee.—Huge lenses of cupriferous schist, up to 150 ft, wide, shallow mining, no timbering—on output of 250,000 t., mining costs are 3*s.* 6*d.*

Utah.—Bingham: masses of pyrites in limestone, worked by adit, square-set timbering—on output of 170,000-190,000 t. per ann., mining costs are 7*s.* 1*d.*-7*s.* 7*d.*

Mercur: auriferous deposits in limestone, flat and shallow, adit mining and caving—on output of 350,000 t., mining costs 5*s.* 5*d.*-5*s.* 10*d.*

Virginia.—Iron pyrites for sulphuric acid making, beds 5 ft. thick and hundreds of ft. long—on output of 4000 t. per mo —total mining costs, 4*s.* (Painter.)

British Columbia.—Center Star: gold quartz—total mining costs, 8*s.* 7*d.*

Ymir: on 71,000 t., total mining costs, 8*s.* 6*d.*

Transvaal.—Auriferous banket, incline shafts, down to 5000 ft., no timbering, little pumping, outputs of 150,000–300,000 t. per ann.—average costs for nine important deep mines are: Stoping, 6*s.* 2½*d.*; developing, 1*s.* 4½*d.*; shovelling, 1*s.* 10*d.*; tramming, 1*s.* 2*d.*; general mine maintenance, 10*d.*; hoisting, 1*s.* 2½*d.*; pumping, 6*d.*; total, 13*s.* 1½*d.*

Rhodesia.—Rezende: wide quartz vein, waste filling—costs, stoping and filling, 9*s.* 2*d.*; tramming, 5½*d.*; hoisting and pumping, 1*s.* 4½*d.*; total, 11*s.* (Woodburn.)

Australia.—Lucknow, N.S.W.: small erratic ore-bodies, very hard country, costs—

	1899 per ton.		1898 per ton.		1897 per ton.		1896 per ton.	
	s.	*d.*	*s.*	*d.*	*s.*	*d.*	*s.*	*d.*
Wages	20	6	1[illegible]	10	28	8	28	6¾
Explosives	1	9¾	1	10½	2	0	2	1¼
Fuel	3	9¾	1	11¾	2	4¾	1	10¾
Stores	0	6¾	0	8½	1	0¾	0	10½
Timber	0	9	0	7	0	5½	0	6¾
Total	27	5¼	24	11¾	34	7	34	0

The wide difference in wages before and after 1897 punctuates the benefits derived from a labour strike; fuel (for air-compressing and hoisting) consumption varies with proportion of machine-drilling in stopes, this being sometimes necessary to keep mill supplied; stores embrace mainly steel and candles. (Author.)

Hard quartz lode 6–12 ft. wide, fairly soft walls, 400 ft. deep, heavy pumping, wages 7*s.*–9*s.*, firewood 9*s.* per cord—total mining costs 10*s.* 9*d.*, viz. mining, 6*s.* 1*d.*; tramming, 10*d.*; hoisting and pumping, 3*s.* 10*d.* (Griffiths.)

West Australia: wide ore-bodies, inefficient labour, wages 3*l.* 10*s.*–4*l.* a week, power 3*l.* 10*s.*–3*l.* 15*s.* per h.p. per mo., all water purchased, rock filling, outputs 4000–14,000 t. per mo.—costs in principal mines range from 7*s.* 3*d.* to 10*s.* 10*d.*, plus about 4*s.* per t. for development. (Hoover.)

Golden Horseshoe (1902): on 46,391 t. oxidised ore—stoping, 4*s.* 5*d.*; timbering, 3*s.* 7*d.*; mullocking, 1*s.* 1*d.*; trucking, 10*d.*; hoisting, 1*s.*; sundries, 3*s.* 5*d.*; total, 14*s.* 4*d.*; and on 83,790 t. sulphide ore—stoping, 3*s.* 5*d.*; timbering, 1*s.* 1*d.*; mullocking, 10*d.*; trucking, 6*d.*; hoisting, 1*s.* 6*d.*; explosives, 10*d.*; sundries, 4*s.*; total, 12*s.* 2*d.*; plus development, 7*s.* 5*d.*; grand total, 20*s.* 5*d.* (Official.)

Korea.—Gold lodes: stoping width 4 ft., moderate deadwork, native labour (good)—total mining costs, 4*s.*–5*s.* per ton. (Speak.)

Malaya.—Raub gold mines: irregular lenses, heavy timbering and pumping, poor wood fuel, but partial (now entire) electric power at 14*s.* per h.p. per mo., coloured labour at an average of 1*s.* 10*d.* per 8 hr.—on an output of 44,000 t. (1904) total mining costs, stoping and trucking, 3*s.* 10½*d.*; timbering, 7¼*d.*; filling, 8½*d.*; development, 8*d.*; Chinese mine bosses, 1*d.*; pumpmen, 1¼*d.*; tool sharpening, 3¼*d.*; fuel, 5½*d.*; engine-drivers and stokers, 2½*d.*; plat and bracemen, 1¾*d.*; total, 7*s.* 1½*d.* In February 1905, total working costs, including European salaries, milling, and all charges except gold export duty, were covered by 2·27 dwt. of bullion per ton, or 2·036 dwt. fine gold, being 50% less than Homestake costs, 60% less than Mercur, 275% less than Rand and Mount Morgan, and 370% less than Champion Reef, India. (Author.)

HAULING AND HOISTING.

THIS subject admits of convenient subdivision under four headings —Surface Transport, Underground Haulage, Hoisting, and Headgears.

SURFACE TRANSPORT.

The conditions in each particular case are subject to such wide variation that no one system can be declared superior. Distance, quantity, speed, gradient (whether up or down), water supply, cost and quality of fuel, wages, horse-keep, continuous or intermittent service, and climate—all exert an influence on choice of system. (See also pp. 3, 10–13.)

Gravity.—Where gravity can be applied, no motive power is cheaper. It may take the form of a decline, such as a timber-slide or ore-pass; or of a traction system by endless rope, using either a terrestrial or an aërial track; or of a flowing stream. This last has been used not only for floating material, but also for transporting ore through launders from mine to mill.

A form known as "raw-hiding" is commonly practised in British Columbia: it is suited to any mountainous country covered by deep snow, and, under such conditions, is preferable to any other system of hauling. An ordinary bullock hide is used; this is spread out with the hair side on the snow, the sacks of ore are laid upon it, the legs are drawn over the top, and the hide is pulled together by lacing from side to side; then the tail and head are laced together, and a log chain is passed around the outside to serve as a brake. A horse may be hitched to the head end. The hair protects the hide itself in slipping down the trails, and one will last as long as the snow is hard; but of course, when it commences to melt, damp and friction soon wear holes. Trails used by rawhide trains are selected so that the grade will not be too steep for horses to climb with the empty hides, or to cause the loaded ones to run away, but sufficiently inclined to compel the horse to pull a little, about enough to keep the tugs taut. One man looks after 3 or 4 horses and their loads.

Pumping and piping are carried to great perfection in the U.S. for petroleum transportation; the cost per bbl. is about 1*d.* per 100 miles. The pipe is specially made of wrought iron, lap-welded, tested to a pressure of 1500 lb. per sq. in., working pressure being 900–1200, or even 1500 lb. The pipe is in lengths of 18 ft., provided at each end with coarse and sharp taper threads, 9 to the inch, and the lengths are connected with long sleeve couplings, also screwed taper. The line is usually laid 2–3 ft. below surface; in some places it is exposed, and at intervals bends are provided to allow for contraction and expansion. At the pumping stations there are storage tanks of light boiler-plate, usually 90 ft. diam. by 30 ft. high, the oil being pumped from the tanks at one station to those at the next, though sometimes loops are laid round the stations, and oil has thus been pumped a distance of 110 miles with one engine. The characteristics of these pumps are independent plungers with exterior packing, valve-boxes subdivided into small chambers, and leather-lined metallic valves with low lift and large surfaces. The engines vary in size from 200 to 800 h.p. The pumps are so constructed that before one plunger has completed its stroke another has taken up the work. The column of oil is thus kept continuously in motion, and violent concussions caused by oil being allowed to come to rest between strokes are avoided.

Oil is frequently pumped, in hot weather, when it is most fluid, a distance of 80 miles. At high pressure, leaks occasionally occur, and workmen sometimes have their hands cut to the bone by a fine stream of oil issuing from a minute orifice, when engaged in stopping leaks.

A very interesting feature of pipe-line transport is the arrangement for cleaning pipes, and removing obstructions caused by sediment. The apparatus (termed a "go-devil") consists in many cases of a brush of steel wire of conical form, fitted, at the base or rear end of the cone, with a leather valve in 4 sections, strengthened with brass plates, and also furnished with long steel wire guides. This instrument is impelled by the stream of oil, and travels at the rate of about 3 miles an hour. Its progress can be traced by the scraping sound which it makes, and it is followed from one pumping station to another by relays of men on foot. It must never be allowed to get out of hearing, otherwise, in the event of its progress being arrested by an obstruction, it may be necessary to take up a considerable length of piping to ascertain its position.

Tramming.—For short distances and intermittent work, especially when wages are low, human labour may be most suitable; and for somewhat longer distances and less easy gradients, animal power commands attention in countries where horses, and particularly mules, and their forage, are cheap. A tramway, even if only wooden rails be used, is infinitely more cheaply worked than a road. Hand-tramming in Sarawak (Chinese labour), in ½ t. trucks up to ½ mile, costs 2*s.* 4*d.* per ton-mile, as against steam loco. traction up to 3 miles at 2*d.* per ton-mile.

The ideal wagon is one that is light for the load it carries, can be the most easily loaded, and the most easily emptied, and

2 E

handled with ease. What is known as the "fiddlestick" wagon has many advantages. It can be made low in the body, and with a door that admits of loading at a very small elevation. It can be either of timber or iron; if the former, it is lighter and cheaper, and is easily made and repaired. A capacity of about 2 tons is a good average size.

Loading into trucks may be done by steam shovel. A 2 cub. yd. shovel will lift and load 30 cub. yd. per hr. of gravel or small-broken rock, and 25 cub. yd. or less if the rock breaks big; while on fine and friable material the duty may reach 60 cub. yd. per hr.

An extremely economical, almost entirely automatic, system of handling is in vogue in the Cleveland iron-stone mines. A full truck, on arriving at pit-head, is pushed out by a man on to an incline, down which it passes to a tippler; there it is emptied of its ore, which falls down a grizzly into a railway truck. On leaving the tippler, the empty truck continues down an incline, and, passing automatic points, reaches a siding which is inclined upwards. At the end of the siding, a powerful spring sends the truck back again down a siding and past automatic points, in a reverse direction; passing the points, it continues down a short incline until it meets a creeper chain, which takes it up on to a level, at the end of which it meets a stop. Three or four trucks may remain on this level until required. On releasing the stop, the truck passes down an incline, past automatic points and a catch, and travels some way up a second incline. On losing momentum, it returns, and is stopped by the catch, where it waits until required. The automatic points last mentioned are on the opposite side of the pit-head to those previously mentioned, so that the truck is now going in the same direction as when it started, and is returning to the cage. On releasing the catch, the truck passes down an incline, and past points which are arranged to send it to the compartment of the shaft where the empty cage awaits it. The tippler has cog-wheels instead of friction-wheels for turning, on account of the heavy weights; it takes one truck at a time. As the full truck pushes the empty one out, the latter catches a lever, which puts the tippler into gear. The tippler makes a revolution, and, in coming back to its place, puts itself out of gear again, while the truck just emptied remains in, ready to be pushed out by the next full one. The tippler is capable of dealing with 300 t. per hr. The only manual labour required, beyond that involved in pulling levers to release catches, etc., is that of pushing the trucks out of the cage to start them on their journey.

Elevating.—Elevators of various kinds are much used in mills for raising crushed material. They are principally of the belt and cup or the bucket-wheel type. The belt may be of webbing, or rubber, or chain; the cups or buckets are of malleable iron, and their front edge should possess double thickness. The bucket-wheel is quite an institution for raising wet tailings, and several examples are shown in the author's 'Gold Milling'; the cups or segments are of wood. Pumps, both of the centrifugal and the plunger pattern, are similarly employed, and give entire satisfaction with certain precautions. (See Hydraulic elevators, pp. 330–1; Dredging, pp. 347–8.)

In the design and construction of belt elevators, authorities differ materially as to angle, feed (whether direct or from a boot), speed, spacing, discharge, etc. Ordinarily, and unless the material to be elevated is hot or otherwise detrimental, rubber (not less than 6-ply for 8-in. to 10-in. belt) is best for the belting. Some recommend a 10-in. belt of 7-ply, 12-in. of 9-ply, 14-in. of 9 to 11-ply, and 16-in. of 11 to 13-ply. Such heavy belts as the last are of doubtful utility with the small head- and foot-wheels commonly provided, though it is rational to increase thickness of belt as width increases, since the capacity of buckets and strain on belt increase at a much greater ratio. An improved belting is surfaced with pure rubber, adding 1–2*s*. per sq. ft. to cost of belt. Buckets are made of malleable iron (seamless, strong and smooth, their round corners aiding free delivery), spaced at 12–20 in.—18 in. is probably best. Capacity is a function of volume, speed, spacing, and size. Assuming buckets to be running $\frac{1}{3}$ full, spaced 18 in. apart, and belt speed 300 ft. per min., capacities of sizes commonly used (in terms of ore weighing 125 lb. per cub. ft.) are:—

Width of Belt.	Dimensions of Buckets.	Capacity of each Bucket.	Capacity of Elevator.
in.	in	cub. in.	t. per hr.
12	10 × 6 × 5	160	22·5
10	8 × 5 × 4	108	15·0
8	6 × 4 × 3·5	50	7·0

The head-wheel should be of diameter to afford proper friction for the belt—30–36 in.; the foot-wheel should not be so small as to cause too sharp a curve in the belt—20–30 in., 24 in. in common practice: the smaller the pulleys, the worse for the belt. Belts should always be 2 in. wider than buckets. To impart a speed of 300 ft. per min., the head-wheel must run at 38·2 rev. per min. if 30 in. diam., and 31·8 rev. if 36 in. diam.; at 300 ft. per min., the centrifugal force will ensure satisfactory discharge, whether elevator be perpendicular or sloping. The former permits in many cases a simpler arrangement, is less costly, and less expensive in repairs, but gives trouble in maintaining proper tension of the belt; its stretch must be taken up by a tightener, while an elevator inclined at 10°–15° is always self-tightening. Feeding ore into the forward slope of the "boot," to be scooped up by the circulating buckets, saves height (sometimes important), and, even with $2\frac{1}{2}$-in.

material, buckets should not be torn off. A useful expedient to prevent spilled ore from falling between belt and foot-wheel, is to insert in the housing one or two sloping shelves *inside* the belt, arranged to deliver into boxes outside. Bearings must be well protected from grit and dust. Operating costs are for power and repairs. Power will be about 1 h.p. per 10 t. per hr. to 40 ft. Repairs may be ½*d*. per ton.

Horse and Van.—

Cost of Running 2 *2-horse Vans* 30 *miles daily for* 300 *days.*

Capital :—

	£	*s.*	*d.*
Two vans at 45*l*. each	90	0	0
8 horses at 40*l*. each	320	0	0
2 sets of harness at 10*l*.	20	0	0
Sundries	5	0	0
Total	£435	0	0

Working :—

Interest on capital at 5 %	21	15	0
Drivers, 2 at 28*s*. per week	145	12	0
Forage, 8 horses at 10*s*. 6*d*. per week	218	8	0
Stabling, 2 vans, 8 horses, 40*s*. per week ..	104	0	0
Insurance	20	0	0
Shoeing and vet., 8 horses at 8*l*. per annum ..	64	0	0
Renewals, 2 vans at 5*l*. per annum	10	0	0
Depreciation (horses) at 20 %	64	0	0
Depreciation (vans, etc.), 15 % on 115*l*.	17	5	0
Annual repaint, 2 vans at 10*l*.	20	0	0
Cleaning 2 vans and looking after 8 horses, 2 ostlers wages at 25*s*. per week each ..	130	0	0
Total	£815	0	0

Or about 1*s*. 10*d*. per mile-day-ton.

Petrol Motors.—

Cost of Running 1*-ton Petrol Van* 60 *miles daily for* 300 *days.*

Capital :—

	£	*s.*	*d.*
One 1-ton petrol van	415	0	0
Sundries	5	0	0
Total	£420	0	0

Working:—	£	s.	d.
Interest on capital at 5 %	21	0	0
Depreciation at 20 %	84	0	0
Insurance	10	0	0
Storage at 5s. per week	13	0	0
Annual repaint as agreement with mfr.	10	0	0
Cleaning at 10s. per week	26	0	0
Driver at 30s. per week	78	0	0
Petrol, 15 miles to the gallon, at 7d. per gal. for 18,000 miles	35	0	0
Oils—gear, lubricating and lamp	5	0	0
Tyres, as per agreement with mfr.	75	0	0
Renewals (tyres not included)	20	0	0
Total	£377	0	0

Or about 10*d.* per mile-day-ton.

The fuel consumption of petrol motor road lorries in practice is is about ⅞ pint per b.h.p.-hr.

The petrol-driven engine might be replaced by a producer-gas engine. With petrol at 6*d.* per gal. and coal at 1*l.* per ton, 1 b.h.p.-hr. costs ⅞*d.* with petrol, and ⅛*d.* with coal converted into producer gas.

Steam Lorries.—The following actual costs on several years' run are from English experience:—

(*a*) Cost of 6 days' average working of 5-ton dray:—

	£	s.	d.
Wages: 1 driver at 5s.	1	10	0
1 assistant at 4s.	1	4	0
Fuel: 24 cwt. coke at 16s. per ton		19	2½
Lubricants: 2 pints cylinder oil at 4s. per gal.		1	0
4 pints engine oil at 1s. 9d. ..			10½
1½ pint bath oil at 1s. 6d. per gal.			3½
	£3	15	4½

Averaged 315·5 ton-miles in 6 days = 2·8*d.* per ton-mile.

(*b*) Cost of 5 days' average working of 4-ton dray:—

	£	s.	d.
Wages: 1 driver at 5s.	1	10	0
1 assistant at 4s.	1	4	0
Fuel: 14 cwt. coke at 16s. per ton		11	2½
Lubricants: 2 pints cylinder oil at 4s. per gal.		1	0
4 pints engine oil at 1s. 9d. per gal.			10½
2 pints bath oil at 1s. 6d. per gal.			4½
2 lb. grease at 20s. per cwt. ..			5
	£3	7	10½

Averaged 284 ton-miles in 5 days = 2·8*d.* per ton-mile.

(*c*) Cost of 6 days' average working of 2½-ton dray :—

	£	s.	d.
Wages: 1 driver at 5*s*.	1	10	0
1 assistant at 4*s*.	1	4	0
Fuel: 21 cwt. coke at 16*s*. per ton		16	9½
Lubricants: 1 pint cylinder oil at 4*s*. per gal.			6
1 pint engine oil at 1*s*. 9*d*. per gal.			2½
3 gal. bath oil at 1*s*. 6*d*. per gal.		4	6
	£3	16	0

Averaged 262·5 ton-miles in 6 days = 3·4*d*. per ton-mile.

From figures covering 42 horses for a period of 4 years, it was established that the 5- and 4-ton steamers each did the work of 7 horses, and the 2½-ton that of nearly 6 horses.

(*d*) Cost of 6 days' average working of 5-ton wagon :—

	£	s.	d.
Wages: Driver	1	15	0
Assistant	1	4	0
Fuel: Coke		15	0
Lubricants: Grease and oil		6	6
	£4	0	6

Cost per ton-mile not given, but wagon replaced 8 horses.

(*e*) Cost of 6 days' average working of 5-ton lorry :—

	£	s.	d.
Wages: Driver	1	12	0
Assistant	1	0	0
Fuel: Coke at 8*s*. 9*d*. per ton		7	3
Firewood, 36–42 lb.			3
Lubricants: 1 gal. crank and cylinder oil ..		2	0
Tallow and waste			3
Water		2	4
Stores, engine-packing, etc.			3
Repairs		3	6
	£3	8	7

Averaged 3·3*d*. per ton-mile.

Depreciation or sinking fund, including all repairs, may be estimated at 25–35 %, the higher figure allowing additional 10 % for greater wear and tear on the bad roads common in mining districts. Fuel consumption is about 3 lb. coke per b.h.p.-hr. (Min. Jl.)

Trackless Trolleys.—Where a supply of electric current is available, the trackless trolley which is in common use in Germany, involving an overhead cable but no tram-line, may prove very serviceable and economical. A speed of $8\frac{1}{2}$ miles per hour is common, and the consumption of power is about 25 % more than on rails.

Cableways.—The cableway, or aërial wire-rope tramway, is especially adapted to the conditions commonly encountered in mining districts. By it the steepest gradients can be surmounted, and rivers, buildings, and other obstructions easily passed over. Loading and unloading can go on during most weathers, as working is not interfered with by rain and but rarely by snow. Comparatively inexpensive foundations are required, and the number of men employed is reduced to a minimum. As lines cannot be worked satisfactorily round *curves*, care must be taken, when setting out, to have loading and unloading stations situated on a straight line from one to the other. If this is not possible, one or more " angle " stations must be employed; these increase working costs. There are two systems—single rope and double rope.

Single-rope lines have an endless running rope, which at the same time does duty for carrying and hauling. Buckets are either rigidly attached, or are suspended by a " saddle " which grips by friction. These lines are cheap and simple, and are suitable for light loads and easy gradients. A rough-and-ready application of this principle is used on scattered nickel-quarries in New Caledonia (F. D. Power). The ore is placed in 20-oz. jute bags, which are only filled to 70 lb., so that they hang loose about their contents, and are easily slung in a looped wire for suspension from the rope, besides being less liable to burst when falling off at the lower terminal. The rope is simply anchored at top and bottom, and is tightened by a rude horizontal capstan worked by hand-spikes (Fig. 160): *a*, rope; *b*, tightening drum (a round log or wooden cylinder); *c*, hand-spike; *d*, leaning-posts sunk well into the ground, and keeping drum in place; *e*, sills holding drum clear of the ground; *f*, pole securing hand-spike. The wire rope is generally $\frac{1}{2}$ in. diam., and the grade is about 6 %, but lessened near the lower terminal to reduce speed of descent. Slings are wire strands from old ropes. The suspender, on steep grades, is a wooden C (*g*) about 3 in. thick, which must be kept greased; on lesser grades, it is an iron J hook (*h*) attached to a $2\frac{1}{2}$-in. pulley. The bags are despatched singly, unless head-winds

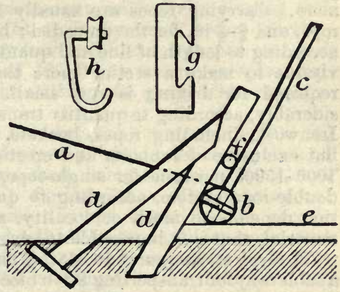


FIG. 160.—PRIMITIVE CABLE-WAY.

make it necessary to secure greater impetus and offer less obstruction, when 2 to 6 are sent off together. A wooden suspender is sometimes hung in front of an iron one as a brake. At the lower terminal, a heap of dried grass serves as a buffer. Derailing is effected by throwing a cloth in front of the suspender. Spans of $\frac{3}{4}$ mile are not unknown.

The double-rope system employs an independent, heavy, fixed rope for carrying, and a light running rope for hauling the load; this is commonly in 5–8 cwt. parcels, and in special cases up to 10–20 cwt. The steepest gradients can be worked with safety by double lines; buckets are attached to the hauling rope by special grips, and travel at about 300 ft. per min. Double lines can transport 60–80 t. per hr. and over very long distances. Supports are of either wood or iron, and are usually 2-legged; 4-legged are only employed where strains are very great. Spans up to 1500 ft. can be adopted, but are not recommended unless absolutely required. For fairly even country, supports are generally placed 100–200 ft. apart, according to quantity transported. Ordinary wooden supports cost 4–5*l.* apiece; iron, 8–10*l.*; high standards, proportionately more. Carrying ropes are usually 1–1$\frac{1}{2}$ in. diam. for the loaded rope, and $\frac{3}{4}$–$\frac{7}{8}$ in. for the unloaded; hauling ropes are $\frac{3}{8}$–$\frac{3}{4}$ in. diam. according to length of line and quantity transported. It is not advisable to make a section more than 3–3$\frac{1}{2}$ miles long. Power required for driving is very small. Cost, of course, varies considerably, according to quantity transported, and nature of country. Ironwork, including ropes, buckets, and terminal gear complete, but exclusive of supports and erection, would cost approximately 1000–1500*l.* per mile for single-rope, and 1200–2000*l.* per mile for double-rope system, according to quantity carried. Cost of erection depends so much on locality, and cost there of labour and material, that it is impossible to give any close figure.

A large stone-quarry in Maine has applied an electric hoist to a cable-way for transporting 8-ton blocks. The cable is 1$\frac{3}{4}$-in. smooth locked-coil rope, and the carriage has 3 wheels designed to distribute the load so as to preserve the cable. Miller button-stop carriers are used; they have the advantages of lightness, greater spacing, diminished stress on the cable, and less wear and tear. The electric hoist consists of hoisting-drum and narrow-faced winch traversing-drum of equal diameter, connected direct to two 1200-volt motors.

On most open-cast mines and quarries, and even in connection with hydraulicing and dredging, a simple form of cable-way, easily and cheaply erected, and movable from place to place, is the most economical and effective medium for moving large quantities of dirt or heavy boulders.

A very satisfactory self-acting rope-way, on Roe's principle, is used at St. David's gold mine, N. Wales. It has a nominal capacity of 24 t. per hr., a speed of 120 yd. per min., and 6-cwt. buckets.

The line is 1100 yd. long, with a fall between terminals of 210 ft., giving a mean grade of 1 in 15 with load. Velocity of travel is kept uniform by an automatic regulator.

A good example of double-rope haulage is that at Block 10 mine, Broken Hill. Its total length is 2000 ft., in 5 spans, longest 850 ft. Skips holding ½ t. of ore are attached to the hauling rope by automatic grips, the operation of which is as follows: The buckets, on being filled at the ore-bins, are pushed along an overhead steel rail, until they reach a cradle, through which the endless rope travels, and are there automatically attached (by a weighted lever, which is thrown over). The movable jaw of the grip travels first on a high-pitched thread to ensure rapid closing, and, as it grips on the cable, it is tightened on a much lower-pitched screw, to prevent injury to the rope. The buckets, hung eccentrically from a frame attached to the traveller, are kept upright by a catch. On reaching the unloading station, skips leave the fixed rope, and travel along an overhead rail, but are still attached to the hauling rope, and are unloaded automatically by a device which trips the catch and causes the bucket to capsize at any required bin. After dumping, the skips are automatically uncoupled and again coupled, being set upright again by hand, before starting on their return journey. They travel at about 5 ft. per sec., and are spaced at intervals of about 120 ft., to ensure even loading, the spacing being marked by the striking of a gong. The capacity is 75 t. per hr.

For hoisting and conveying to a distance not exceeding 200 ft., the Dawson self-dumping carrier (see p. 314) is the best system of handling gravel from a shallow shaft, being strong, compact, and simple in operation. (Fig. 161.) Springs are avoided, as being unsafe under constant jar. Three distinct operations are accomplished, two by the carrier, and a third by an auxiliary rope used in dumping the bucket. The first consists in engaging the bucket as it arrives from the shaft, and carrying it to the dump-box, the second, in returning the bucket to the shaft, and dropping it to the bottom. As the carrier returns down the ⅞ in. cable *a*, hook *b* (to which the bucket is attached) occupies the position indicated by the solid line; cam *c* lies horizontally, and is held firmly in position by dog *d*, the weight of the bucket pressing the notch of the cam against the point of the dog. When the carrier reaches the shaft head, the corner *e* of the dog strikes block *f*, and frees the point of the dog from the notch of the cam. The front of the cam now occupies the position indicated by the dotted line, and, as the bucket sinks into the shaft, is pressed against the block, since the carrier tends to move backward. Thus, as the bucket sinks with the hook on the ½ in. cable *g*, the carrier is held firm. As the bucket rises from the shaft, the strain is toward the engine, and the lower end of the dog still prevents the apparatus from moving. When cam *c*, by the upward movement of the bucket, reaches the horizontal position indicated by the solid lines, the point of dog *d* jumps into the notch, at the same time allowing the carrier to move up cable *a*, and also once more securing the cam in its horizontal position. Block *f* is held in position by a suspended log *h*, which ensures its engaging with the dog. Much the same contrivance is used in some stone-quarries in Scotland, and has been copied in one or two Malayan tin diggings; it is there known as a "Blondin."

Conveying.—The "conveyor," of whatever form, is adapted only to limited distances. It exists in many types.

"Screw"-conveyors consist of a semicylindrical iron or steel trough, in which turns an endless screw or spiral 8–12 in. diam. at 50–75 rev. per min. Capacity, varying with pitch and diam. of screw, speed of rotation, weight of material, and angle and height

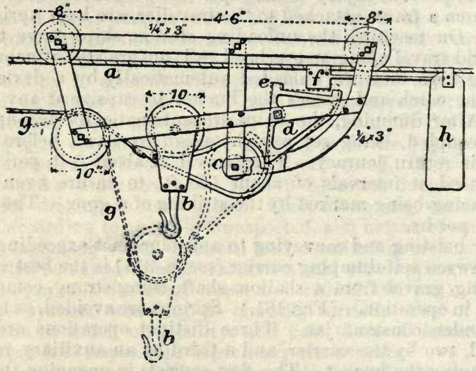


FIG. 161.—DAWSON SELF-DUMPING CARRIER.

of elevation, is approximately 2–12 t. per hr. of material weighing 100 lb. per cub. ft. Power requirements are high, say 5 h.p. for 10 t. per hr. in a 100 ft. × 9-in. screw, owing to excessive friction. It is cheaply installed, say 60*l.* for such an example; but wear and tear are very great. On dry finely-crushed material, they are often useful, but quite impossible for wet, sticky, or coarse stuff.

Screw-conveyors in cylinder form are sometimes applied to transporting hot material and cooling it at same time by outward application of cold water or inward draught of cold air.

"Scraper" conveyors operate by a series of "flights" or plates travelling in a square, semi-cylindrical or pointed trough. They admit of feed and discharge at any point. The capacity of a 10–12 in. diam. scraper, flights 16–20 in. apart, travelling 100 ft. per min., is about 10 t. per hr. of 150 lb. per cub. ft. ore for 100 ft., requiring at least 5 h.p.; installation costing about 90*l*. Many are in operation moving coal and ore, but are very prone to break down. The Chinese chain pump, made exclusively of wood, is on the same principle, and most effectively unwaters shallow mines.

The reciprocating conveyor is really a multiple shovel working mechanically in a shoot, the flights being hinged, so as to hang free and be drawn over the surface of the load at each back stroke. Only material readily penetrated by the upward travelling flight is adapted to this style; moreover, it is very wasteful of power. A machine to transport 10 t. per hr. 100 ft. would demand a 15-h.p. motor, and would cost 125–250*l*.

The continuous bucket or pan conveyor is of many types and capacities, ranging from a style suited to feeding ore from breaker-bins to mortar-boxes, to installations for loading ships with coal or ore. They are very strong and durable, tolerant of great strains and arduous conditions, capable of much elasticity in feed and delivery, and require very little repair or renewal; but they are costly to instal, and, having in themselves much dead weight, demand considerable power.

The belt conveyor is quite the most widely applicable and useful of all—least expensive in first cost, renewals, and power consumption; most adaptable to conditions of load, incline, and discharge; has great capacity; and permits simultaneous picking of ore in transit. It consists essentially of a belt or band, supported on idlers, and running over pulleys, by one of which it is driven, an arrangement at the other end taking up slack and keeping the belt tight. The simplest form is a flat belt, made very wide to prevent material from spilling off; but generally the belt is troughed by side rollers. The most durable belt is made largely of rubber, but for hot substances it is replaced by cotton duck. Capacity depends on width of belt, depth of troughing, speed of travel, and weight of ore carried. In a troughed belt, the load will cover ½ the width, and the depth in centre will be ¼ width. A speed of 300–400 ft. per min. is usual, rising occasionally even to 900, but at great wear and tear. A 12-in. troughed belt run at 300 ft. per min. will deliver 10 t. per hr. 100 ft. with 3–3½ h.p., and the installation will cost about 120*l*. Repairs are ordinarily about 12½ % per ann.

The Homestake Co. uses 3 Robins belt conveyors, each 167 ft. long. They receive ore through shoots in the bin bottoms, leading points being close together; they then diverge from this centre, and deliver into a 100-stamp mill. Each belt carries about 160 t. per 24 hr.; run at moderate speed, and driven from a single shaft at the discharge end, they require about 3 h.p. each. While ore is in transit, small pieces of drill steel and wood chips are picked out; by keeping these from the mortars, 10–20 tons more per day are crushed. The 3 belts are fed by one boy.

At the Van Ryn mine, Transvaal, one belt carried 142,800 lb.

of rock per hr. 199 ft. horizontally and raised it 48·5 ft. Power, including motor losses = 8·11 h.p.; efficiency, estimated on vertical lift = 43·12 %; empty belt took 3·69 h.p. Another belt carried 177,856 lb. of rock per hr., 497·5 ft. horizontally and raised it 25·5 ft. Power, including motor losses = 8·47 h.p.; empty belt took 2·94 h.p.; efficiency 40 %, allowing for driving of travelling "tipper."

At the Knight's Deep, a conveyor belt was installed for handling sands, tailings and waste rock from the sorting house, at an angle of 20°; when operating on rock alone, this inclination was found to be too steep, an occasional lump breaking away and rolling. But schemes are on foot which contemplate using conveyor belts up to angles of 45°, a second belt similar in all respects to the conveyor, and travelling at the same speed, making contact with it, and causing the lumps of ore to be firmly held between them. The breaking load of a belt has been calculated at 7200 lb., as against chain at 6600 lb.

Costs.—The cost of surface transport per ton-mile is so largely dependent on local conditions that the variations are very wide.

Wheelbarrow work, based on a man's capacity to wheel 1500 lb. 1 mile a day, may be computed at about 3*s.* 9*d.*, with wages at 2*s.* 6*d.* per diem.

Man haulage, with cars on level rail-track, costs about 4*d.* in Germany, but reaches all told, on the Rand, to 6*s.*, of which labour alone is about 2*s.* 6*d.*

Horse and cart, on a basis of 20 ton-miles a day, 1 man per horse at 2*s.* 6*d.*, and horse-keep, at 2*s.* 6*d.* per diem, figures out at about 3*d.*

Horse or mule on railed track, in England, with wages at 7*s.* and feed at 1*s.* 3*d.* a day, comes to about 1*d.*; mule tramway in Germany is about 2½*d.*; in Pennsylvania (wages 6*s.* 6*d.*), 3*d.*; and in S. Africa, about 7*d.*

Cableway haulage ranges between 1½*d.* and 4*d.* Actual examples at Bingham, Utah, after 3 years' operating, showed, for wages alone, 1½*d.*–3*d.*, and for all charges, 1·9–3·4*d.* per ton-mile.

Steam locomotive traction at a Belgian colliery is stated at ½–1*d.*

Compressed-air motors operate at about 1*d.* to 2*d.*

Electrical haulage is reported at prices varying from ¾*d.* to 2*d.*

Endless rope haulage of tailings at the Meyer and Charlton, steam driven, handling 6500 t. per mo., vertical lift 47½ ft., length of haulage 1242 ft., cost 6·22*d.* per ton, ½ being wages and ⅓ maintenance. At the New Goch, on 4500–5000 t. per mo., lift 54 ft., haul 1726 ft., costs were 7·93*d.* per ton. (Denny.)

Belt conveying: Roodepoort United, 6500 t. per mo., length 240 ft., lift 50–60 ft., speed 276 ft. per min., width 18 in., first cost installed 2537*l.*, power used 11 h.p.; working costs per ton 1·627*d.*, including ·602*d.* labour, ·361*d.* power, ·664*d.* maintenance and renewals. Van Ryn No. 1, 10,000 t. per mo., 497 ft. haul, 23 ft.

lift, 120 ft. per min., 18 in. wide, first cost 3250*l.* (including "tripper"), power 5 h.p.; working costs per ton, 2·229*d.*, including ·343*d.* labour, ·063*d.* power, 1·823*d.* maintenance and renewals. Van Ryn No. 2, 12,000 t. per mo., 199 ft. haul, 48 ft. lift, 340 ft. per min., 24 in. wide, first cost 2050*l.*, power 5½ h.p.; working costs per ton ·578*d.*, including ·109*d.* labour, ·07*d.* power, ·399*d.* maintenance. (Denny.)

UNDERGROUND HAULAGE.

Handling Ore in Stopes.—When the angle of dip of the lode is about 60°, no difficulty is experienced in removing ore from stopes: the only labour necessary is that required to clear it from working faces to enable drilling operations to be resumed, which does not occupy more than about ½ hr.; it falls by gravity to boxes at foot of the stope, and the cost of handling is practically nil. But between this angle and, say, 10°, as met with in many of the Transvaal mines, is a very wide range; the relation between angle of dip and cost of handling becomes apparent, the average cost of conveying from working faces to boxes being generally about 1*s.* 6*d.* per ton. With a stope width of only 3–7 ft., there are many difficulties in the way of mechanical handling, but a number of expedients have been tried. In many of the foremost American mines, even with large outputs, shovelling is still relied on, costing frequently as much as 1*s.* 8*d.* a ton, and amounting to 20% of the total expense of mining.

Shaking shoots have met with greater favour than any other form; they possess simplicity of construction, consisting merely of a length of plate bent in a trough, and suspended from the hanging wall by chain slings. The shoot is made in sections of 15 ft., which can readily be removed for blasting, and as readily re-erected. The chief difficulty is getting the holes drilled in line and properly spaced. When the hanging wall is of irregular contour, difficulty is experienced in getting the holes properly pitched, with the result that the suspension chains tend to pull against each other instead of producing a swinging motion, and this is most marked in narrow stopes, where the chains are comparatively short, and a few inches difference in the hanging bolt centres seriously affects the swing of the shoots. Also, each time it becomes necessary to move the shoot in the stope, fresh holes must be drilled in the hanging wall, and often new hooks supplied.

Tuchtan's improvement on the shaking shoot, used at the Village Main Reef, is suspended from a light angle-iron frame, and made in 8-ft. sections, being thus easily handled by 2 "boys." It is held at the top end of each frame by a ⅝-in. round-iron U, being fitted with a series of lugs which allow for inequalities in the floor. The U hanger of the top section is fitted with a lever arm which is depressed by a "boy," and gives a swinging motion;

a spring bumper also gives a sudden jerk to the shoot, and considerably increases its efficiency, especially in flat stopes. This conveyor works quite well on perfectly level ground, and a distinguishing feature is its adaptability to narrow stopes, as it does not stand more than 1 in. above ground level. Obviously, it is extremely portable, and needs no hole-boring. It is said that, on trial, 1 " boy " did the work of 6.

Stationary shoots are similar, but need no chains, being simply laid on the foot-wall. In stopes having a dip of 30° and upward, no difficulty is experienced with ore sticking, provided it is dry, and the shoot is free from wet fines, which materially affect the coefficient of friction between ore and shoot. They are made ⅛-in. thick, and in 10-ft. sections, so as to be conveniently handled; each length is lapped over that in advance, and held by bolts

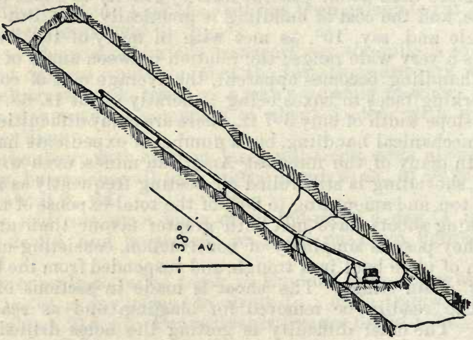


FIG. 162.—STATIONARY SHOOT.

passing through both. This type is quite efficient in suitable stopes, and is in general use in the Lake copper mines. That at the Mohawk (Fig. 162), for example, is of ¼-in. sheet iron, semicircular in form (2 ft. wide × 16 ft. long), and supported on wooden horses and props, or, where near the floor of the stope, simply blocked up. The lower end is set at such a height that ore thrown into the shoot will discharge into a car standing beneath it on the level track. It is extended, section by section, as height increases, the ends simply overlapping. Any desirable slope may be given to the shoot, even different grades to different sections, as is usual with the last one or two sections at the foot of the stope, in order that the velocity of downward-moving ore may be checked somewhat before entering the car. A shoot may be set up after the face of a stope has become cumbered with rock, in which case several shovellers are placed along the line; but it is better practice to keep the floor clean, advancing the shoot as the face advances, the shovellers being at the upper end. Further, the slope may be varied slightly by extending the shoot diagonally across the stope, whereby it may be made to serve a considerable area, being shifted from side to side.

Truck haulage from stopes is only practicable on flat lodes, when the stope is not less than 4½ ft. wide, fully 18 in. being necessary above the top of the truck for loading purposes. Trucks are run on ordinary rails (12–14 lb.), and are operated by a winch at top or foot of stope, or run in balance with a brake-wheel. Either way, the chief advantage lies in the absence of boxes at foot of stope, as cars are run in on flat-sheets or turn-tables, unhitched, taken to main bins, relieved of their contents, and returned; also, no time is occupied in erecting or dismantling gear, and rails suffer no damage if left in stopes during blasting.

A modified cableway, known as the Henderson-Tucker conveyor, arranged for easy erection and removal to accommodate blasting, has been used for some time at the Geldenhuis Estate, where some of the stopes are very flat (5–10°). A 3-cub. ft. skip is carried on a ⅝-in. wire rope strained between two points, one fixed over the box-hole, the other secured to an ordinary machine-bar, which may be moved as required. The loaded skip descends by gravity, and is hauled back to the loading point by a small winch clamped to the bar (operated by air, or electric motor, or hand); arrived at loading point, it is held by a brake on the winch. This gear is equally efficient in long stopes and short, but when the span exceeds 100 ft., the rope must be supported against sag, say by means of jumper arms clamped to an ordinary machine-bar, at 60 ft. intervals. Tipping is done automatically by a lug riveted to the skip body, which comes in contact with a piece of piping arranged diagonally to the rope at the required dumping place. Tension is put on the rope by tighteners. The gear may be dismantled in 15–20 min. and erected in 30 min., irrespective of length of stopes; its removal from one point in the stope to another occupies only 10–15 min. During blasting, it is only necessary to remove the skip and slacken the rope. It was claimed that mainly by this appliance the output per "boy" per mo. was increased from 24·27 to 33·12 tons.

The table on p. 432 shows some wide divergences in practice. Thus, Village Main Reef stopes at 29° are only 150 ft. long, while Glen Deep at 28° are 230 ft., and the cost in former case is 14·7*d.* as against 18·5*d.* Some of the Treasury stopes, only 45 in. wide, actually necessitate lifting the ore.

Rails and Flat-Sheets.—Underground tracks in metal mines range from 14-in. gauge with 12-lb. rails and small trucks up to 24-in. gauge and steel rails weighing 16 lb. per yd. for heavy loads. Even still stouter rails (say 24-lb.) are economical for mines of

Cost of Ore-Handling in Stopes: Transvaal. (Williams.)

Mine.	Length of Stope.	Angle of Stope	Cost of Conveying from Face to Box at foot of Stope.	Method.	Ore Removed from Face to Box at Foot of Stope per Boy per Shift.
	Ft.	°	*s.* *d.*		Tons.
Robinson	120	35	1 5·2	Mech. and hand	2·3
Randfontein ..	150	60	..	Hand	..
Geldenhuis Estate	150	21	0 5·1	Henderson-Tucker	5·6
Village Main Reef	153	29	1 2·7	Hand and Tuchtan	2·4
City and Suburban	225	30	..	Hand	5·4
Treasury	225	15	2 6	Mech. and hand	1·5
Robinson Deep ..	225	..	1 8	Shaking shoot and hand	2·0
Simmer and Jack East	225	..	1 3·4	" "	2·1
Glen Deep	230	30-10	1 6·5	Mech. and hand	2·9

large output: the Quincy mine, Lake Superior, has 35-lb. rails in the levels (3-t. trucks), and 50-lb. in the incline shaft (8-t. skips); and the New Goch has 80-lb. rails in the incline for 6-t. skips. They are laid on sleepers. These may be made of wood or of steel, the former making a firmer road. Light rails may be fastened down with iron dogs, and no hole is then necessary in the flange of the rail; heavy ones need coach-screws passed through the flange, and fish-plates for the end joints. Steel sleepers are provided with slits and with bolts and washers. Common dimensions for rails on sleepers 26 in. apart are given below:—

Tramway Rails.

Load.	Weight per yd.	Rail Height.	Width of Face.
cwt.	lb.	in.	in.
6	6	$1\frac{1}{2}$	$\frac{3}{8}$
10	8	2	$\frac{3}{8}$
14	10	$2\frac{1}{8}$	$\frac{1}{2}$
18	14	$2\frac{1}{4}$	$\frac{5}{8}$
24	16	$2\frac{3}{4}$	$\frac{5}{8}$
30	20	$2\frac{3}{4}$	$\frac{3}{4}$

Where two lines of rail make a junction, the track is broken, and flat-sheets are interposed (Fig. 163). Hard-wood planks *a*, 12 × 2 in., are first laid down, and to them is spiked a ⅜-in. steel flat-sheet *b*, abutting against the sleepers *c*. From the end of each rail, for a length of about 6 in., the flange is cut off, and the rail is curved, and hammered down to a feather edge, as at *d*. The main track *e* is joined by the secondary track *f*. When another secondary track comes in from the opposite side, an endeavour should always be made to afford it separate accommodation, so as to avoid four lines of rails converging upon the same flat-sheet, otherwise it is impossible to retain any solid rail, and, if reasonable speed is maintained in trucking, there will be frequent delays and spills through the wheels missing the rails. In the system shown, the continuous rail *g* serves as a guide, and prevents over-running with the full truck.

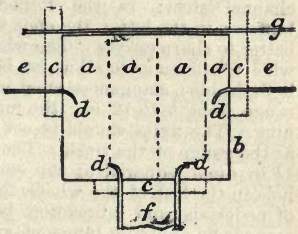


FIG. 163.—JUNCTION OF TRACKS.

The rails of the Angelo Deep west incline shaft are laid on concrete 15–18 in. wide, raised from bed-rock just enough to give the necessary elevation and solidity; this is found to be cheaper than timber, and quite satisfactory.

In some of the Lake copper mines, the rail-bed proper ("stringer") of the inclines is about 13½ in. wide by 14 in. high; supporting it is a mass of concrete 16–18 in. wide, extending to rock. The depth of this supporting mass varies, with the irregularities of rock face to which it is attached, from 4 in. to 2½ ft. The stringer and support are structurally one, being moulded together. The stringer portion is reinforced by 1⅛-in. steel cable passing longitudinally through it. The rail is attached by bolts, spaced 3 ft. apart; they hold by clips which grasp the rail-flange. The bolts pass through the stringer into a rectangular opening about 3 × 4 in. in section, which affords access to lower end of bolt. Concrete mixture is 1 part Portland cement, 3 of sand, and 5 of crushed rock, tightly rammed. In first cost and in maintenance it is cheaper than timber, but the wear and tear of rails and wheels are notably greater, owing to rigidity of bed.

Trucks.—Most underground tramming or trucking is done by human labour, and the capacity of the truck is limited to 10–16 cub. ft. or 20–25 cwt., and even less where youths are employed. For very large outputs, 2- or 3-ton trucks hauled by mechanical means effect great economy. The body of the truck may be a rectangular box, or more or less U-shaped in cross-section. The box has greatest capacity for over-all size, but can only be tipped endwise, and must have a swinging door held by a long rod turned at one end. The pointed U admits of easy side-tipping, and needs no door. Sometimes bodies are made of 1½-in. deals, lined with thin sheet iron; but for hard service and heavy loads, ⅛-in. sheet steel is preferable. The carriage may be of wood, or of light channel steel: in the former, the longitudinal timbers form buffers; in the latter, the ends are rounded, and adapt themselves better to sharp curves. The wheels should have deep flanges and wide treads, so as to allow for badly-laid track. A durable metal is desirable; manganese steel answers well. In diameter, wheels range from 6 to 12 in.; the maximum size gives the easiest running. The axles should be set the same distance between centres as the gauge of the track. They are most often simply a steel bar 1½ in. diam., squared at the ends, and driven into a corresponding hole in the hub of the wheel, the body resting on them by means of pedestals, and lubrication being performed in a most casual manner by applying lumps of grease. An improvement is to make the xale round, and pass it through the wheel, with a shoulder inside and a linch-pin outside, so that both wheel and axle revolve independently, especially if the track is irregular. A grease-cup is sometimes formed in the pedestal. When the output is great, it pays to have a self-lubricating axle, such as that used at the Anaconda mine, Fig. 164; axle *a* is made in halves, wheels *b* being pressed on cast-iron sleeves *c*, which serve as supports for axle, and as oil reservoirs. Each sleeve is bored to fit the axle, and is counter-cored to provide ample oil-space *d*. The sleeve enters about 1½-in. into the hub of each wheel, with a sufficiently close fit to prevent loss of oil. Axles divided in the centre allow for variations of travel in passing around curves; they are made of cold-rolled steel, 1¾ in. diam., and slightly tapered at one end before the wheels are pressed on; the other end of half-axle is provided with a groove, in which is placed a small fork-shaped brass casting *e*, straddling the axle and serving to hold it in position. At centre, the sleeve is enlarged, and provided with an opening sufficient to let the two forks drop in; a pressed-steel cover *f* over this opening retains them in position, and keeps out dust and dirt. Each sleeve is provided with lugs for attaching wooden or iron bodies. Rowbotham's self-oiling wheels and axles, made by Hadfields, Sheffield, are simpler, cheaper, and quite as efficient. Manganese-steel wheels have the advantages of lightness and practical immunity from breakage, but the disadvantage of being excessively

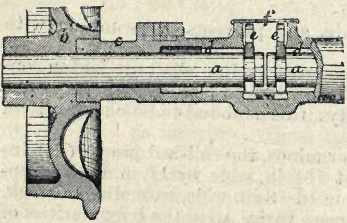


FIG. 164.—TRUCK WHEEL.

hard on the boxes, by reason of the hard hub grinding its way into the face of the box. This can be minimised by using 2 thin soft-iron washers between the hub and the box, which wear comparatively little, and can be readily replaced at nominal cost.

Atlantic mine trucks are 8 ft. long, 2 ft. high, and 28 in. wide inside, holding 1·7 t., and requiring 2 men. The bottom stands only 8 in. above the track, so that shovelling is easy; both ends are open, and the wheel-base is so proportioned to the overhang that tipping is easy at either end, while the sloping surface of the bottom is adapted to sliding heavy rocks in. Trammers pile big pieces at each end, so as to make a rough sort of retaining wall, and shovel small stuff inside. It used to be the practice to fill a box-truck from a platform or sollar 28 in. above the track, but most of the rock "breaks big," and trammers cannot control the lumps without great loss of time and smashing of trucks. Bottom plates are ½ in.; others, ⅝ in., reinforced by angle-iron.

Another recent pattern is 7 ft. long, 2 ft. 1⅝ in. high, and 2 ft. 7½ in. wide, holding 2½ t., though the load usually handled does not exceed 2 t. Wheel diameter is only 12 in., the bottom coming close to the ground. One end is open, and when the car is tilted, rests on the ground, the bottom forming an inclined plane, up which large rocks are worked until loaded.

With one fast and one loose wheel, the axle turning in a waste packed box, the loose wheel rotates with the axle, except when turning curves, concentrating wear on a cheap cast-iron seat in the axle-box, which is readily removable. Boxes can be made solid and very simple—an important consideration where trucks must be dumped, and sustain a heavy blow in the process. The wheels are readily removable, and the car need be raised but little to pull out an axle. This, with manganese-steel wheels and steel boxes, makes a nearly ideal running-gear for mine cars. (Norris.)

The frictional resistance of trucks on ordinary mine tracks varies from nearly 6% in starting on a curve with plain wheels, to less than 1½% for long trips with self-oiling wheels moving at 12 miles per hr. Traction resistance is commonly assumed at 20 lb. per ton, but, on a dirty mine track, it is rarely less than 40 lb., and may easily reach 60.

Bins and Loading Stations.—In metalliferous mining on any but an insignificant scale, bins for stowing broken ore below ground, and controlling its supply to the shafts, are a necessity, so that getting and raising may be carried on independently of each other. While each ore-pass or shoot is in a sense a bin, and suffices where operations are limited, trucks filled from it must be raised without much delay, because the same trucks do duty both in the levels and in the shafts. When large outputs are dealt with, a much larger truck or skip is used for hoisting than could be conveniently handled, especially by manual labour (which is most general), in the levels; hence large bins are constructed in communication with the shafts.

These are filled by ordinary hand-trucks, and discharged automatically into hoisting-trucks.

The bin itself usually consists principally of an excavation in solid rock, timber and iron lining being resorted to only for the lip or lower portion. It is a good plan to cover the mouth of the bin with a heavy iron grating affording 9–10 in. square spaces, so that any larger pieces may be arrested there and spalled before going

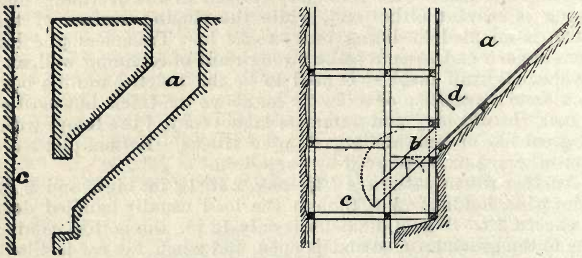


FIG. 165.—VERTICAL-SHAFT BINS.

farther. Anything much above that size is apt to cause inconvenience by blocking outlets and interfering with operation of doors. Commonly 2–3 in. plank and $\frac{1}{8}$–$\frac{1}{4}$ in. steel sheeting are used for lining lower ends of bins.

Examples of bins for vertical shafts are shown in Fig. 165. In one, solid ground is left between bin and shaft, except at outlet of the former; in the other, the side of bin next shaft has to be close-

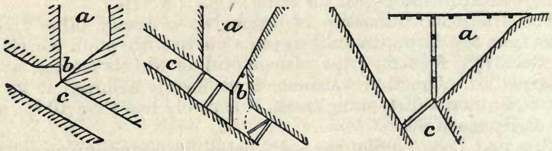


FIG.166.—INCLINE-SHAFT BINS.

lagged and lined: *a*, bin; *b*, loading-shoot; *c*, shaft. Bins for incline shafts are illustrated in Fig. 166: *a*, bin; *b*, loading-shoot; *c*, shaft. The three types require progressively increasing quantities of timber. Discharge may either be up or down shaft, the former (in low-angle shafts) requiring the skip to be cut down somewhat at top, and thus reducing its capacity. An outlet in the floor of the bin is sometimes adopted, but this is most ill-advised, because of the load being always on the door. When the capacity of the bin is very great, this inconvenience will be experienced even with side doors, and then it is necessary to provide a set-off,

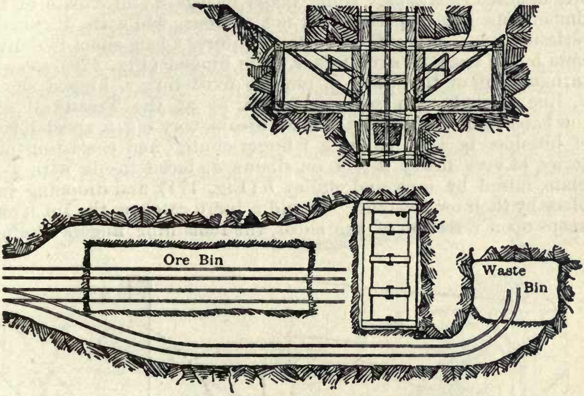


FIG. 167.—TREADWELL SHAFT BIN.

as shown by ledge *d* in Fig. 165. The Treadwell bin (Fig. 167) accommodates 500 t. of ore on one side of the shaft, and 200 t. of waste on the other. Every bin should have a grizzly, to prevent the admission of inconveniently large lumps, say exceeding 9 in. cube; if made of 30-lb. rails, spalling may be done on them.

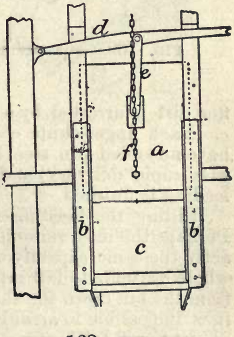


FIG. 168.—BIN-DOOR.

Bin-doors are made in several patterns, but principally they consist of ½-in. iron or ¼-in. steel plate moved up and down between narrow angle-iron rims fixed to upright timbers of bin-frame; the door is actuated by a lever bearing upon one of the uprights, and either having a short link or a slot to receive a pin on the door. Sometimes the door is counterpoised by a suspended weight, as Fig. 168: *a*, door, 24 in. wide; *b*, rim plates, 3 in. wide; *c*, exit from bin, 21 in. wide; *d*, lever; *e*, link; *f*, chain to counterpoise. Occasionally the door is arranged to slide laterally, which affords better control over a rush of material. Rarely a rack and pinion are substituted for the long lever, and are more satisfactory, occupying less room and being handier of manipulation, but costing a little more. A novel method (Fig. 169) is used in the Fayal iron-mines, where enormous quantities are handled, advantage being taken of legs *a* and caps *b* of the timber sets. Its superiority is not obvious, while its drawbacks certainly are. When the situation requires a long shoot to deliver from bin to truck or skip, this is often hinged (Fig. 170), so as to turn up out of the way: *a*, bin; *b*, fixed lip; *c*, hinged shoot; *d*, line of shaft. In some instances, as at the Treadwell and the Liberty Bell, where the rock breaks very big, a special type of bin-door is used, called a "finger-chute," and consisting of a series of very heavy 6 × 4 in. elbows *a*, faced inside with ¼-in. plate, raised by rope and pulley *b* (Fig. 171), and dropping into place by their own weight. Should a lump catch on the lip, it only keeps open a section of the shoot, the remaining fingers closing;

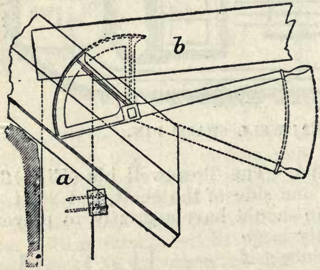


FIG. 169.—BIN-DOOR, FAYAL.

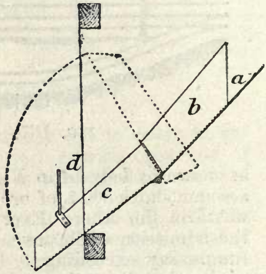


FIG. 170.—HINGED LIP TO BIN.

fine dirt is arrested by a tail-board dropped behind the angle-irons *c*. Each finger-chute costs about 15*l*. Those used in the shaft have an added ½-in. steel lip (hinged), lying at 36° when in use (to give rapid delivery) and turning up into a vertical position when loading is finished.

Filling the hoisting-skip from the bin is done in several ways. Perhaps the most reliable is to first fill a small spill-shoot, of exactly the same capacity as the skip, from the bin, and allow the whole contents to fall into the skip; this prevents any serious run from the bin down the shaft, owing to jamming of a door. Where it is impossible to arrange for a spill-shoot, a lip must be added to prevent rock falling on the incline track. At the Liberty Bell, a steel pocket is used, holding 4 mine-truck loads, which exactly fill 1 skip. The skip generally rests on the track rails simply, while being loaded, but is kept in place by a heavy wooden gate hinged to a cross-timber along the hanging-wall of the shaft, and lowered under the skip by a $\frac{5}{8}$-in. wire rope, running over a block operated by a lever at the station. Skips standing on a track of slight inclination, even as high as 38°, do not readily fill when loaded from the back, owing to the slope of the bottom of the skip being practically equal to the angle of repose of the ore: loading from the side largely overcomes this difficulty. At the Mohawk mine, the bottom of the skip is permanently set at a greater angle than that of the shaft tracks, by raising the forward wheels. In the Quincy, a steeper slope is given the skips by temporarily changing the grade of the shaft tracks at the loading stations; this does not cause any inconvenience,

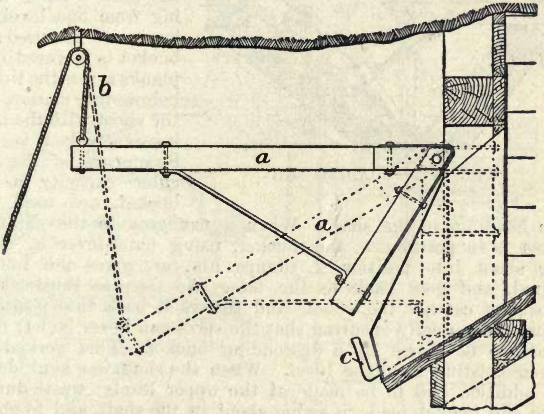


FIG. 171.—BIN-DOOR, TREADWELL.

as the usual grade is maintained by bridging during hoisting. The special arrangement here used is shown in Fig. 172.

About 10 ft. below the floor of the station, a portion of the runner, on which rails are secured, is notched out, so as to receive a pocket of semi-steel, equipped with a tongue about $2\frac{1}{2}$ ft. long, faced with steel, which takes the place of the rail, and is hinged, so that it can be raised by a lever at the station. When these tongues are raised, the skip is lowered until its hind wheels drop into the pocket; here it is held, tilted at an angle of 50°, until loaded. It is then hoisted, and the tongues (which have rested on the hind wheels) drop back into place automatically, leaving the rails continuous, as before.

When hoisting is by bucket, there is always much inconvenience in detaching them, in running them to and from the face on small trolleys, and in filling them. The arrangement shown in Fig. 173 obviates all this trouble. The shaft is 4 × 4 ft. The buckets are 3 ft. high, 2 ft. 4 in. diam. at top, and 2 ft. 2 in. diam. at bottom. A bale is fastened 2 ft. from top, and is never detached from the cable, dumping at the surface being done by withdrawing a pin which holds a clevis on the bale. When hoisting from one level, the levers are not used; the bucket is lowered on to planks across the timbers below the station, and the shoot with the hand-lever attached is used. Scoop-cars, of the same cubic capacity as the bucket, are used from the headings to the shaft. When a car gets to the shaft, the shoot is in position 1; the mucker, using hand-lever *a*, throws the shoot into position 2, dumps his car, gives the hoisting signal, and goes back to the face. As soon as the bucket is raised, it catches the shoot, and throws it back into position 1, thus automatically ensuring that the shoot can never be left in the shaft, to be struck by a descending bucket. This worked well when hoisting from one level. When the shaft was sunk deeper, an addition had to be made at the upper levels; waste dumped into the bucket made it swing about in the shaft, and to obviate this, lever *b* and bucket-rest *c* were added. One end of lever *d* is pivoted to hand-lever *a*, which is keyed to the shaft on which the shoot works; the other, to one end of lever *e* pivoted near its centre to the compartment-piece, and its other end to lever *f*. This is offset to pass behind the guide, and is pivoted at its other end to lever *b*, which is bolted to the shaft on which the bucket-rest *c* works. When the shoot is in position 2, *c* projects into the shaft, fits against the bucket, and stops its swinging while being filled. When the bucket is raised, it throws the shoot into position 1, as before, and the levers (made of 2 × ½ in. iron) assume the positions shown by dotted lines, leaving the shaft clear. The whole thing may be made in the mine smithy at a nominal cost. When lowering timbers, the shoot can be lifted out and replaced in a few minutes. (E. C. Musgrave.)

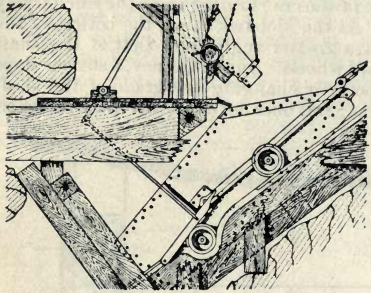


FIG. 172.—LOADING SKIP.

Man Haulage.—So few mines publish statistics that it is difficult to get any reliable figure for this item. At the Crown Reef, in 1897, with a maximum distance of 1500 ft., and an average much less, employing Kaffir labour, the total cost was about 1*s.* 6*d.* per ton; labour alone at most of the S. African mines can be contracted at about 7½*d.* per long ton. Ross Browne places it ('07) at an average of ·833*s.* (10*d.*) per ton mined. In the Lake copper mines, it ranges between 7½*d.* and 1*s.*, and will average about 8¾*d.* per ton milled,—2-t. trucks, trammed 340 ft., double journey occupying 35 min., 37 t. trammed by 2⅓ men per 10 hr. shift (9 hr. effective).

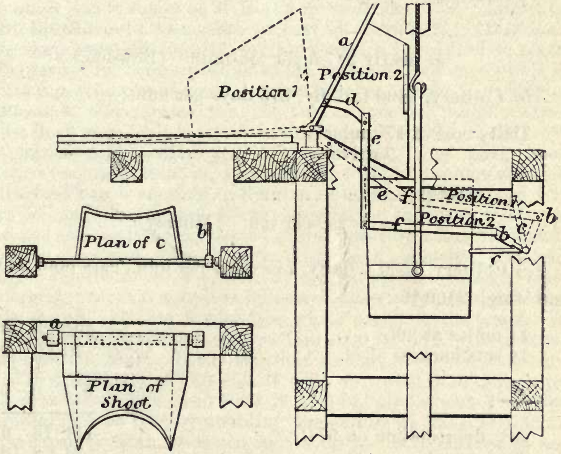


FIG. 173.—BUCKET LOADING.

Mule Haulage.—Mules can enter any portion of a mine unhindered, and rails as light as 16 lb. can be used, while in other systems it is not advisable to lay less than 35-lb. rails; but, in some mines, even the smallest mule is impossible, on account of the low roof; to take down top or take up bottom, to make the entry for mules, means a large addition to cost of mining. Moreover, a mule requires 5 times as much air as a man, so that by doing away with one mule, enough air would remain for additional 5 men. Also the constant pounding of mules' hoofs makes a rough roadbed, and forms holes between the sleepers, where water collects and makes mud puddles; the mule also cuts the sleepers out in the centre, which means frequent renewals. The average active life of a mine mule is 7–10 years; a good supply of reserve mules must be on hand, because of numerous accidents, and this means a considerable expenditure for idle animals.

Costs.

(*a*) Colliery, 1200 ton-miles daily:—

	£	s.	d.
Interest and depreciation on 32 mules	1	3	7
Feed, attendance, harness and repairs	5	5	4
Drivers, 11 at 6*s.* 7*d.*	3	12	5
Couplers and spragmen, 11 at 5*s.* 8*d.*	3	2	4
	£13	3	8

= nearly 2·7*d.* per ton-mile. (Bowden.)

(*b*) Colliery, 1400 t. daily, 275 days per ann.:—

	£	s.	d.
Daily cost of 17 mules..	2	0	5
„ „ 9 drivers	5	1	0
Total	£7	1	5

= 1·2*d.* per ton. (Murray.)

(*c*) Colliery, 1500 t. daily, 245 days per ann., cars hold 3600 lb. and weigh 2400 lb.:—

	£	s.	d.
14 mules at 36*l.*	504	0	0
14 sets harness at 5*l.*	70	0	0
	574	0	0
20% depreciation on 574*l.*	114	16	0
6% interest	34	8	10
Feeding, shoeing and care at 2*s.* 1*d.*	357	5	10
6 drivers at 11*s.* 8*d.*	857	11	0
	£1364	1	8

= ·9*d.* per ton. (Murray.)

Compressed-Air Haulage.—In the case of very large outputs, some form of mechanical haulage is resorted to.

The following particulars relate to compressed-air plant at a Pennsylvania colliery, using a Norwalk 3-stage compressor, compressing about 300 cub. ft. air per min. to 600 lb. per sq. in., and supplying Porter pneumatic motors, operating on over 6000 ft. of roads with a grade of $\frac{1}{2}$–$2\frac{3}{4}$% and averaging about 1% in favour of loaded cars. Cars weigh 2800 lb. empty and 9800 lb. full, and are hauled in lots of 12–20. Two motors replace 32 mules, and, operating 10 hr. daily, they perform a net average ton-mileage of about 1200 per diem. Steam consumed, including loss by condensation in long transmission, is 5200 lb. per hr.; boiler power, 174 h.p.; coal consumed, 10,400 lb. per diem; cost of fuel and firing per diem, with coal at 2*s*. per ton, 9*s*. 8*d*. Air loss in transmission is a little over 4 %; actual consumption is 83·4 %, leaving 16·6 % total loss by slip, leakage, etc. Net cost per ton-mile is a fraction under 1*d*. If the plant were operated at full capacity (2500 net ton-miles daily) all the year round, cost would be less than ½*d*. per ton-mile; and if the air were re-heated (see p. 117) by passage through water subjected to steam at 90 lb., an air economy of 50 % might be gained, which would reduce the net cost by about 20 %. (Bowden.)

In compressed-air locomotives, air-pressure is reduced to about 140 lb. per sq. in. before passing to the cylinders, which, as well as the running gear, are of usual locomotive type. Such a variety of conditions is encountered that no two motors are alike. Locomotives have been built in sizes running from 5 × 10-in. to 12 × 18-in. cylinder, and weighing 9000–45,000 lb.; they have been restricted in height to 53 in. and in width to 39 in.; gauge of track has been as narrow as 18 in. Number of tanks may vary from 1 to 3, and to obtain increased storage capacity, a separate tender is sometimes attached and connected to the motor by a flexible metallic coupling; these tenders have large tank capacity, and add greatly to the distance a locomotive may run with one charge. Air for charging motors is supplied by compressors which deliver into a battery of tanks or into a pipe-line which parallels the track. A storage capacity of 550 cub. ft. in a line 4000 ft. long, will necessitate a pipe 5 in. diam. If the length of line must be 8000 ft., a 4-in. pipe with a capacity of 700 cub. ft. will do: 4000 ft. of 5-in. pipe would cost about 500*l*., and 8000 ft. of 4-in. pipe about 775*l*. (for material alone), so that by doubling the extent we have 27 % more storage capacity at an increased cost of 60 %. Locomotives do not stand more than a minute or two at charging stations, and as they will travel 3000–10,000 ft. with one charge, the time required for charging is not worth considering.

Electric Haulage.—Electric locomotives are much used in America and Continental Europe, some worked by secondary batteries, but the majority taking their power off wires running along the roof or sides of the drift. Except in fiery mines, no method of haulage is more convenient or economical, where large quantities have to be transported considerable distances. The locomotives run at 7–15 miles per hr., and can ascend grades of 5 %; when this is exceeded, rope haulage is preferable. Most are worked with continuous current. The useful effect is usually 70% of the electric power in the conductors, expressed in terms of draw-bar pull; they are mainly 10–15 h.p., the weight of the former being about 2½ t., but some are much larger and heavier (10–15 t.).

The Peabody mine, U.S., producing 2000 t. coal daily, with an average haul of 1800 ft., uses 2 15-t. locomotives, with double-end control and reversible trolley-pole which accommodates itself readily to height of roof. No. 0000 trolley wire is fastened to the roof by hangers 8 in. outside of outer rail. The wheels have steel tires, which give them a tractive effect 20% above that of the chilled rim. The locomotives are provided with two motors wound for 250 volts, and exert a draw-bar pull of 8200 lb. on the level. They have pulled 17 loaded cars up a 2½% grade 1200 ft. long. Cars weigh empty 1950 lb., and hold an average of 6600 lb. coal, so the weight of the loaded trip would be over 72 t. The track (42 in. gauge), measuring 9000 ft. over all, is laid with 40-lb. rails bonded and cross-bonded for return current. Curves are 40–60 ft. radius, which gives 16–18 ft. from point of frog to point of switch on all cross-overs and turn-outs; the outer rail is elevated to suit a speed of 8–10 miles an hour; loaded cars are all caged on one side and empty cars are taken off on the other. The bottom proper is 225 ft. long, 18 ft. wide, 8 ft. high, and has a double track laid to a grade of 1–1½% in favour of loaded cars, which run by gravity to the cages. On the return of an empty car from top, it is bumped off the cage by a loaded car, and runs by gravity down a 4% grade for 60 ft., then up a 3% grade for a short distance, and back-switches itself into one of the run-arounds where empty cars are collected. The installation cost 4411*l.*, including 2 15-t. locomotives (950*l.*), 175-kw. generator (500*l.*), 200-h.p. engine (400*l.*), 2 150-h.p. boilers (600*l.*), 9000 ft. trolley wire (225*l.*), 116 t. tram-rails (260*l.*); and working costs, on 275 days per ann., are ·7*d.* per ton (wages 11–12*s.*), including interest (6 %) and depreciation (8 %). (Peltier.)

The Woodward mines, Penn., exemplify an isolated direct-current station. Bare copper cables run from the feeder panel of the switchboard on poles for about 400 ft., then connect to lead-covered paper-insulated cables suspended in a borehole; the cables loop in at each vein to a dry place, where they connect to distributing bus-bars, from which the various feeders are controlled, their combined weight in the borehole being 7 t. Mine equipment consists of 10 electric locomotives, 9 electric hoists, and 6 electric pumps; total rated h.p. is 1796. Two types of locomotives are used, one weighing about 13 t., the other about 6½ t. The heavier is equipped with 2 50-h.p. (railway rating) motors governed by series and parallel controller; it will develop a draw-bar pull of 4500 lb. at a speed of about 7½ miles per hr., when operated in multiple, and 5200 lb. for a short period with the rails sanded,—when operated in series, the speed is approximately 3·2 miles per hr. at the rated draw-bar pull; it is used for long and heavy haulage in the main gangway. The 6½-t. gathering locomotive is equipped with 2 25-h.p. motors, and has an automatic reel, used when collecting from and delivering to the chambers. Light rail (25-lb.) is used in the chambers, but no trolley wire, as the cable (500 ft.) on the reel is used instead. When it is desired to run int free end of the cable is hooked over the trolley wire way, and the locomotive runs into the chamber, un cable as it goes. When it comes out, the cable is automatically reeled up. It will exert a drawbar pull of 2500 lb., and the speed, with motors in multiple, is approximately 7 miles per hr.; in series, 3·8 miles; and its momentary draw-bar pull with rails sanded is about 2700 lb. One is handling cars between the foot of the shaft and a branch about 4000 ft. away, and distributes cars to 12 chambers, some of which are 5–17% to the rise, and others 8½% to the dip; 5 cars are delivered to each chamber per day. Another handles cars from 15 chambers, the grade of which varies from 2 to 6% to the dip. The maximum grade on which they have been used is 17%. (Warren.)

At the Wesselton mine of the De Beers Co., electric trains, composed of 6–8 mine trucks each 20 cub. ft. (1 t.) capacity, run at

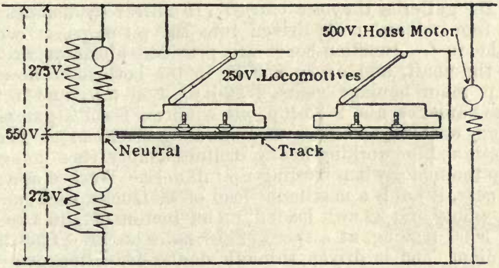


FIG. 174.—ELECTRIC DISTRIBUTION.

a speed of 15 miles per hr., round curves of 55 ft. radius, the track being laid with 45-lb. steel rails, with manganese steel fittings, points and crossings. The cost of transport over a system having ½ mile run is 1½*d*. per ton. (Rathbone.)

Electric distribution at varying potentials without undue cost for cable is ingeniously overcome by the 3-wire system (Fig. 174): 2 275-volt generators are operated in series, and the neutral is connected to the track return; 250-volt locomotives are operated on each side of the 3-wire system, and the motors on the hoists are connected across the 550-volt circuit; the advantage of 550-volt transmission for the hoists, and lower voltage for the locomotives, is thus secured. (Warren.)

Drums for underground haulage, especially up inclines, are largely worked electrically, the compactness and convenience of electric driving being particularly marked in this application. They can be made exceptionally safe, electric "visors" having been devised to cut off current and apply brakes if the load is moving at too high a velocity near the end of the wind: thus overwinding becomes almost impossible. Electric motors can be used both for main-and-tail and for endless rope or chain (see below). The latter is less easy to arrange, on account of the relatively slow movement of the endless rope. Motors may be direct-current, working at 400–500 volts, running at about 600 rev. per min., and developing up to 60 h.p. Friction clutches can be introduced with advantage, especially with a polyphase motor, as they enable it to get up to normal running speed before putting on the load, which can then be done gradually. (Prof. Louis.)

Auckland Park colliery, Durham, uses coke-oven waste gas to drive 3-phase generators giving 200 kw. at 2400 volts when running at 350 rev. per min. Overhead bare copper transmission lines (5 miles) are employed above ground; in the shafts and galleries, current is carried by armoured cables. Special clamps attached at every 6 ft. to the buntings support the cables in the shaft; along the galleries they are carried (15 miles) by hempen slings hung from miners' nails driven into the pit-props, or cemented into the roof. Junction-boxes are provided about every 200 ft. down the shaft, and every 858 ft. in the bottom. There are 5 110-h.p. main haulage gears, 2 30-h.p., 1 46-h.p., and 1 60-h.p. endless haulages, and 2 85-h.p. pit winders, besides pumps, fans, lighting, washing, screening, and handling machinery. The plant was designed for working heavy inclines, empty tubs going down to help the motor when hauling up full ones. Worked as a single-drum gear, it hauls a maximum load of 30 trucks, each weighing 8 cwt. empty and 28 cwt. loaded, up an incline of 1 in 4 to 1 in 6, about 3000 ft. long, at a speed of 6–7 miles an hr. The drum is 4 ft. diam., and is driven through double-reduction gear and a flexible coupling by a 110-h.p. motor running at 695 rev. per min.

Rope Haulage.—Haulage by wire rope may be installed on three different systems—(*a*) the self-acting inclined plane; (*b*) endless rope; (*c*) tail-rope.

(*a*) The inclined plane system, relying on a sloping track to give the necessary momentum for loaded cars to travel down and bring up empties, by winding and unwinding a rope on a drum, is obviously of limited applicability. Length and condition of roadway, weight of cars, friction of rollers, weight of rope, friction of drum, and stiffness of rope, are factors which determine the least inclination at which engine planes can operate. In some old installations, an endless chain is employed, but no one would apply a chain nowadays in a new plant.

(*b*) The endless rope system consists of a wire rope spliced endless, working over a drum at any point of the haul; the rope is kept tight, and, by special grips, a load may be attached or detached at any point. Applied to a single track, the motion of the rope is reversed for return of an empty train; but with a double roadway, the load may be received on one track and returned on the other, the rope moving in one direction only. It is rarely applied in metalliferous mines. The rope passes at one end round a fixed pulley, and at the other round a pulley borne on a tension-carriage, from which is suspended a weight which keeps the rope taut. Movement is communicated generally through the fixed pulley, though it may be done at any point. The rope is carried on pulleys only at proper stations; elsewhere it rests either on the trucks, or on hollow cylindrical cast-iron rollers placed 75–100 ft. apart. The usual size of rope for 16-cub. ft. trucks, running on a track which is mostly horizontal, is $\frac{5}{8}$ in. diam. The trucks are attached to the rope by a clip or jockey—commonly a fork so fixed on one end of the truck that it can revolve about its vertical axis, which is not quite in line with the rope, and the fork is about as much on one side of the rope as the axis is on the other. At the loading station, the truck is run in under the rope, which is pressed down into the fork; it thereupon turns the fork about its vertical axis, and, becoming itself slightly bent, is securely held, the grip being stronger the greater the tension of the rope. At the off-loading station, the rope may be mechanically lifted out of the fork by rising to pass over a pulley, or a boy may be stationed there to knock the rope up. The usual speed is 150–250 ft. per min., and the trucks are placed at regular distances of about 50 ft. apart. This method is not applicable where the track passes from the horizontal to an incline which is greater than 1 in 6; at the junction with the gradient, the rope, unable to follow the angle, would hang in a curve far enough above the truck to pull it out of the jockey.

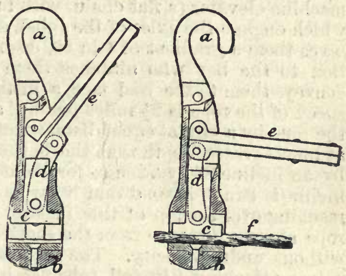


FIG. 175.—ROPE CLIP.

An improvement on the ordinary clip (Fig. 175) was introduced at a colliery where, the roads being undulating, it was necessary that it should hold back on the down gradients as well as hauling up the inclines. Prior to its introduction, the clips became disengaged while passing vertical rollers round curves, necessitating frequent stoppage, and the ropes were split and strands were torn out. The case which carries hook *a* for hanging on to the truck at the upper end, has a fixed jaw *b* at the lower end, and contains a sliding jaw *c*, to which movement is imparted by toggle *d*, whereof the upper link is a spring lever *e*. When the lever is lowered, rope *f* being between the jaws, the two halves of the spring lever are tightly compressed, and the centre of the toggle having been, by the movement of the lever, carried across the centre line of its extremities, the movable jaw is held down under great pressure, and the rope is gripped securely until the lever is raised. When hauling, the clip hangs at an angle, and the rope, being thus bent, tends to force itself out of the jaws at opposite corners. The saving effected in wear and tear of ropes is very marked.

Rope pulleys should be 10 ft. diam. for rope 1 in. diam., increasing 6 in. for every additional $\frac{1}{4}$ in. of the rope.

At De Beers diamond mines 50–60 miles of endless rope haulage, dealing with some 14,000 trucks, are in operation.

It is very general in Midland collieries worked principally by powerful steam engines on the surface. An example at Ansley Hall deals with a gradient of 1 in 2. On arriving at pit bottom, empty tubs are pushed off the cage by full tubs, and are taken round a sharp curve, seized by an electrically-driven creeper or machine elevator (a flat chain, with fingers or catches at intervals, which engage the axles of the tubs), and carried to a height which gives them a gradient of 1 in 80, down which they run by gravitation to the boy who attaches them to the endless rope, which conveys them to the coal face, a distance of nearly a mile. The speed of the rope is $2\frac{1}{2}$ miles per hr., and the tubs are attached to the rope by a clip at equal distances of 20 yd. Coal is here worked at much greater depth than the bottom of the shaft, being reached by an incline 880 yd. long; for about 350 yd. from the bottom, the incline is 1 in 7, beyond that it varies from 1 in 2 to 1 in $1\frac{1}{2}$. After reaching over the top of this incline, tubs are detached from the rope at some distance from the shaft, and run on an easy gradient without undue velocity. The capacity is 110 t. per hr., which involves attaching 240 full tubs per hr., and the same number of empty tubs, this duty being performed by one man at each end. The rope is $1\frac{1}{4}$ in. diam., weighs $2\frac{1}{2}$ lb. per ft., and is passed 3 complete coils round the wheel.

(*c*) The tail-rope system employs two ropes winding on separate drums. The main rope runs along the ground, and is attached to the front of the loaded train; the tail rope is supported along the walls or roof of the tunnel, extending to the end of the haul, where it passes over a return wheel, and along the ground, being attached to the rear of the train. The main rope is then wound on its drum, hauling the load and followed by the tail rope. For the return trip, the motion is reversed, the tail rope is wound on its drum, and the empty cars are drawn to the end of the haul, dragging the main rope. The tail-rope system is capable of more general application. The endless rope must be kept constantly tight, necessitating heavier rope and causing greater friction on its supports, and, though less rope is required, the added expense for mechanical treatment more than offsets the saving; curves and entries, particularly, are sources of difficulty. There are very few conditions under which the tail-rope does not exhibit advantages. It is easily applicable to a straight or curved run of any length and of any variation of grade, and side entries may be worked as easily as the main tunnel.

In S. Wales, mechanical haulage is confined almost entirely to the tail-rope system, and is accomplished mainly by a host of small hauling engines (6-in. and 8-in. cylinders) worked by compressed air.

A modification of it is in use at the Treadwell mine. On the hanging-wall side of the ore-bin is a double-drum winding-engine (cylinders 7 × 10 in.), with drums 2 ft. 8 in. diam. Set directly in front of, and close to, the engine are 4 posts; 2 carry a sheave suspended on a horizontal axle, for guiding the upper rope and causing it to wind smoothly on the drum, and 2 support a roller which answers the same purpose for the lower rope. From the drum to the point where the drifts branch out, the upper rope is supported by snatch-blocks suspended from the back of the drift, and by sheaves at the ends of arms fastened to 10 × 10-in. posts. A horizontal sheave is placed at the point where the direction of the rope is changed to allow it to enter the drift. Since this sheave is subject to severe strains, it is held rigidly in place by horizontal 10 × 10-in. pieces bolted to the posts. From this point to the end of the drift the upper rope is carried by sheaves fastened to the posts of the finger-shoots (see p. 438), immediately under the protruding lip, where it will be out of the way and at the same time protected from blasting. The sheaves are inclined, so that the greatest strain is at right-angles to the axle, and the rope is prevented from jumping out by pegs placed across the top of the sheave. The lower rope is kept in line by horizontal sheaves fastened to blocks, their number depending on the crookedness of the tunnel; and it is prevented from dragging on the ground by iron rollers between the tracks. The lower sheaves (Fig. 176) are placed as near the track as possible, and are mounted on pieces of 10 × 10-in. timber braced against the side of the tunnel. To guide the rope into the sheave, the front end *a* of the block is bevelled off to the height of the rail. On the top of the sheave is a piece of wrought iron *b*, bent as shown, its objects being to prevent the rope from jumping out and to hold the axle vertical. At the ends of the drifts or at convenient points in them, are sheaves to carry the end-loop of the rope. A sheave mounted on a truck, and fastened by clamps to the rails, proved a failure, on account of the strain pulling up the track, and doing other damage. Signals are conveyed by 2 bare iron wires run the entire length of the tunnel, parallel, and

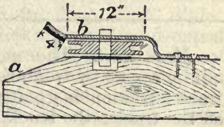


FIG. 176.—GROUND SHEAVE.

4 in. apart; at the winding-engine, they connect with a bell and signal light, the current being obtained from the electric-light circuit. Signals are given by placing an iron candlestick across the 2 wires, or by a special implement. As the wires are bare, signals can be given from any point, which is a great convenience in case of a train jumping the track, or other accident. Two trains are used on each level, consisting of 7 cars, each holding $1\frac{1}{2}$ t.; while one is discharging at the shaft ore-bin, the other is loading. They are run at a maximum speed of 800 ft. per min., and their capacity is 750 t. per shift, or 1500 t. per day.

Rope Haulage Systems Compared. (N. Eng. Inst. Min. Engs.)

System of Haulage.	Average Gradient for Full Tubs.	Cost in Pence per Ton per Mile.							
		Ropes or Chains.	Tubs.	Grease and Oil.	Coals.	Repairs to Engines and Boilers.	Maintenance of Way.	Labour.	Total.
Endless chain	Rise 1 in 59	0·083	0·173	0·155	0·256	0·072	0·068	0·572	1·379
Tail rope ..	„ 1 „ 213	0·276	0·114	0·186	0·558	0·098	0·064	0·583	1·879
Endless rope	„ 1 „ 36	0·252	0·309	0·138	0·323	0·196	0·083	1·692	2·993

Mono-Rail.—The mono-rail, being suspended from the roof, minimises many troubles; whatever the spill, no ore lodges on the track; friction is greatly reduced; trucks can with difficulty get off the track; the truck body, hanging directly over the bin, turns bottom-side-up, and tips clean and quickly; and, however acid the water, rails and wheels are high and dry. Some 3000 ft. of it are installed at the Langlaagte Deep, where it is used on development work (both driving and winzing), and in tramming both in the stopes and from the stope-boxes. The present roads are all equipped with 16-lb. rail, but, while just the thing in a winze, this is far too light for a drive. At Langlaagte, the road consists of vertical iron hangers firmly fixed in the roof of the drive, winze, or stope, with a right-angle bend 6 in. from bottom, to form a step on which is carried a 16-lb. rail. The rail is placed at the edge of the step, to allow clearance between the hanger and truck-wheels on one side, and bridle of suspended truck and edge of step on the other. Hangers are made in two parts (Fig. 177), to allow for adjustment in bringing the rail to grade. The upper part *a*, which is attached to the roof, is made in two styles; for drives and crosscuts, it is of 1 in. square iron, pointed at one end, ragged at the corners, and with a flat wing welded in 1 in. from the other end, at an angle of about 135°, and 2 holes drilled through the wing to bolt on the lower part; for winzes and stopes, it is made with shorter shank and of lighter iron, and the wing is welded in at right-angles to the shank. The lower part *b* is made of 1 × 4-in. flat iron, bent 6 in. from the bottom, at right-angles, with two holes drilled through the 6-in. arm to bolt the rail to, and two near the end of the long arm to bolt to the wing of *a*.

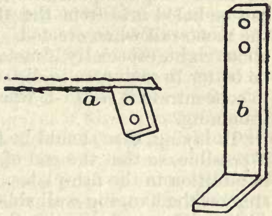


FIG. 177.—HANGERS.

To attach the hangers to the roof of a stope or winze, horizontal holes are hand-drilled to the depth of the shank, say, 12 in.: in the one case at right-angles to the line of the rail, and in the other parallel to it. The hole is then partially filled with dry wood strips reaching the full length, the shank is inserted (wing down) and driven well home. The stress being vertical, and at right-angles to the shank, the hanger cannot pull out.

In a drive or crosscut, the shape of the roof, and the necessity for clearance between the trucks and the side of the drive, forbid side holes, so they are put in as near a right angle with the vertical as the roof will allow, and adjacent holes looking in opposite directions. They are put in by machine, 18 in. deep, 2 in. diam., approximately 8 ft. apart, and so placed that a plumbline dropped from the centre of the hole will be 2 ft. from the footwall side of the drive, to ensure the truck coming under the stope-boxes, and to have a clear travelling road on the hanging side. A mono-rail should always be on the footwall side, and as close as possible. Into the finished hole is driven a perfectly dry and well-fitted plug of pitch pine, to the full depth, and, when it is "home," an inch round hole is bored in it to the depth of the shank of *a* from point to wing. The hanger is then driven home, wing down, the greatest care being taken to keep the wing in a vertical position. If this is not done, the completed hanger will not be plumb. Moisture swells the dry wood plug, causing it to grip the rough sides of the hole and the ragged shank of the hangers. Before bolting up *b*, the exact required length must be ascertained. To find this, one end of the grade-stick is placed on the last completed hanger, and the other is brought to such an elevation that the grade is just right. The distance is then measured from the bottom of the grade-stick to the holes in the wing of *a*, and on this measurement *b* is drilled in the shop ready to bolt up. It is a little cheaper to punch the holes, but they are not as exact, and any variation affects the grade of the road.

In new work, the machine man developing should put in the holes as he goes along; he can do this when rigged up for drilling in the face. From one rigging, 2 holes can be put up with 8 ft. centres, and 1 man can put up 12–16 holes per shift. [It is also advisable to put up box-holes as the drive advances, so that they can be holed into from the stope, to save the danger of blasting the mono-rail when erected.] From experience, 8 ft. centres seem about right, especially if motor traction be contemplated. It would be better in each case to distribute the load over 12 ½-t. trucks than to concentrate it in 6 1-t. trucks; but this does not apply to hand tramming.

In laying, care should be taken to break joints on the hanger, if possible, so that the end of the rail may have a bearing surface in addition to the fish-plates. The hangers are bolted up with the step on the hanging-wall side of the drive, and the rails are laid with the flange flush with the end of the step, and bolted securely to it through the flange. In crosscuts, where roads converge, T hangers are used, the rails from the drive being continued on each arm of the T. If the rail were carried on the footwall side of the hangers in the drives, this would not be possible, as the bridle and wheels of the trucks would then foul the T hangers in the crosscut, necessitating a double line of hangers and a wider crosscut. In stope roads, for which the mono-rail is particularly adapted, the rail may be laid on either the foot or hanging-wall side of the hanger, but it is of advantage in narrow stopes to put it next the hanging, because the footway is thus brought lower down the stope; the tram boy does not have to stoop so much, and his work is thus made easier. In development work, in order to keep the road well up to the face, one or more light jack-bars are set up, fitted with brackets on which to lay the rail. By this means, trucks can be got right into the face, no matter what the bottom is, and the permanent rail is laid as soon as it is safe to put up the hangers.

When the roof of a drive is very high, or unsafe, sticks of timber are made fast between walls to carry the hangers. The length of the rail should always be some multiple of the distance between hangers, and the longer the better.

Switches, turntables, crossings, etc., are easily managed. At one station, an overhead shunt shifts loaded or empty trucks from one track to another. A simple and effective switch consists of a short length of rail, one end of which is pivoted vertically on the last hanger before the branch, while the other is supported by a hanger sliding on a curved strip of iron fixed horizontally in the roof of the drive. There is just enough travel to allow the loose end of the rail to swing from one branch road to the other, where it is secured by a sliding fish-plate. This switch must be moved by hand, and has the objection that one branch of the Y is always open, so that a full truck might run through into the level. To obviate this, an automatic lifting switch may be used.

It is essential for easy tramming that the grade of the road shall be very even, especially if heavy trucks are used.

The trucks consist of a U- or V-shaped body, hung at each end by a single pin to the bridle, which is attached by two vertical draw-bars to a pair of 2-wheeled bogies. There are thus 4 wheels in tandem. Diameter of wheels and depth of bridle vary according to head room. In winzes and very narrow stopes, 6-in. wheels are used; in wide stopes, 8-in.; in drives, 8–12-in. Wheels have a flange each side, ½ in. deep, and cut away at ¼ in. taper, to give a tread at the bottom of the groove of 1¼ in. This allows ⅛ in. side play, the rail head being 1⅛ in., and prevents wheels mounting the rail on a curve. The wheel base is as short as possible (9½ in. with 8-in. wheels), and the bogies are set as close together as clearance will allow. The wheels of each bogie run loose on axles made fast to the bogie frame, which is welded in the centre to the draw-bar, thus forming a T, of which the draw-bar is the stem, with wheels at end of each arm. Axles are fitted with grease-cups.

The vertical centre line of the draw-bar of each bogie bisects the wheel base, and the lower end of the bar is made round, and flanged and held to the bridle by a loose-fitting clip. Weight of truck-body and bridle is thus carried on the flange of draw-bars, while each bogie moves independently of the other on a vertical axis, thus allowing trucks to negotiate sharp curves with least friction. A simple catch secures the truck body in position on the bridle, and the end pins on which it swings are so placed that a loaded truck will turn turtle when the catch is released, and nearly right itself. This saves a lot of time at ore bins, and prevents knocking trucks about.

Truck bodies are made to hold 15 cub. ft.—U-shaped for drives, and V-shaped for stopes. With a perfectly rigid road, and 14-in. wheels, a ton truck will probably be readily handled by one good strong "boy." Such a load will, of course, be hard to start. In drives and wide stopes, a U-shaped body is preferable; but in narrow stopes, a special V body, shaped to fit the floor of the stopes, is advantageous, as it does away with the need of taking up the bottom. In winzes, only one bogie is used, with 6-in. wheels, and a special bucket hung by the bridle from it. Winding is done by hand from short distances, and by air winch from greater depths.

Truck bodies should be just long enough to allow for clearance of the loaded body when tipping. A long bridle tends to more swing on the curves in transit, putting unnecessary strain on road and hangers. For the same reason, the road should be carried as near the roof as is compatible with clearance and even grade, since the shorter the hanger the less the lateral vibration. All roads should be erected with sufficient clearance for 14-in. wheels, if required. In the present 1 × 4-in. hangers, the flat side being towards the rail, full advantage is not taken of the strength of material. The hanger should be bent the other way, so that the 4-in. face would stand at right angles to the rail, but such forgings are too expensive to make locally. In future it is intended to make hangers in 3 sections, the foot and roof piece being so designed that the 1 × 4-in. vertical portion can be bolted up, giving maximum rigidity. No turntables have yet been erected, but one has been designed for special work, and presents no difficulties. Ball bearings might also be used to advantage on the trucks, if not too costly. (Wager Bradford.)

HOISTING.

Windlasses, Whips, and Whims.—In the early stages of all mining, and to very considerable depths in localities where machinery would be costly, difficult to instal, and inconvenient to operate, simple appliances are essential. Fig. 178 shows example of rudimentary hoisting plant used in the Malayan tin diggings: they are all practically costless, can be made on the spot from materials growing at hand, and are quite useful for small-scale operations, such as prospecting pits, within the scope of average native labour. In A, B, an elastic pole *a*, footed in the earth and stayed by a couple of props *b*, serves as a "whip"; to the upper end is tied a strong and supple cane *c*, carrying a natural crook *d* of tough wood or root, from which is slung, by another short cane, a stout basket. In C, D, a growing tree *a* is availed of for supporting a long lever pole *b*, loosely slung; to the short end of *b*, is tied a slight stick *c*, having a natural hook at the lower end, on which is hung a kerosene tin *d*; the stay *e* serves to hold the long end of the lever, and support the weight of the filled bucket, when necessary. E is a Malay windlass, consisting of 4 small poles *a* tied in pairs × wise with cane, and footed in the earth, carrying a simple round log *b*, which has been barked, and to which at *c* is bound a small cross stick; on the barrel *b* is run a rope *d*, made from plaited native fibres of various kinds, having at each end a wooden crook holding in one case a basket (for earth) and in the other a kerosene tin (for baling water). The Chinese windlass is much stronger and more effective, but still dispenses entirely with iron. The barrel is generally 9–10 in.

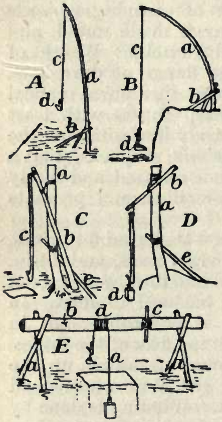


FIG. 178.
NATIVE WHIPS.

diam., giving much better leverage, and has two annular grooves cut out for admitting the standards. The handles are made from small tree branches having a right-angle bend or fork, and are simply wedged or mortised into the ends of the barrel.

The ordinary windlass is made with fixed posts. If needed to be shifted about, it is much better wedged (Fig. 179): *a*, sole-plate; *b*, post; *c*, wedge. (Ashmore.)

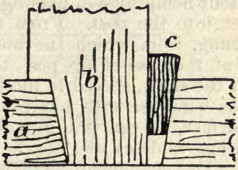


FIG. 179.
WINDLASS POST.

Usual efficiency in windlassing is 12 t. per 8-hr. shift by 2 men in a 60–90 ft. lift, costing ·5–·8*d.* per ton-ft. (See also p. 2.)

The unbalanced load of a single windlass (1 bucket) may be $\frac{1}{6}$ of the load; this is remedied by the double system, but ordinarily the depth of winding is limited to 150 ft. by the available rope-coiling space on drum, avoiding overlapping, and using hempen rope. Flexible wire rope of much less diam., or increased diam. of drum (restricted by handle leverage ratio), or lengthened drum (involving extra cost of shaft), or a divided drum, must be adopted for depths of 500–600 ft., to which windlassing is often applied with cheap native labour. With a drum centrally divided by a collar or a series of pegs, each half of drum having a separate rope wound to overlap, each lap increases leverage of load (partially counter-balanced by greater length of free rope hanging from other side of drum), and there is much friction and wear of rope. Ashmore suggests that the rope only make one half turn round the drum, and be prevented from slipping by means of 12 M-shaped pieces of $\frac{5}{8}$ in. round iron, placed round the centre of the drum. The angle in which the rope lies is approximately 35°; it never touches the wooden portion of the drum, but jams in the angle of the M's, and is prevented from slipping, even if one end is hanging loosely down the shaft, and a full bucket is attached to the other. Moreover the buckets travel in the middle of the shaft, permitting greater depth and increased speed of winding. Destruction of rope must be rapid.

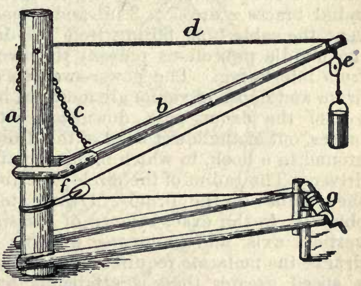


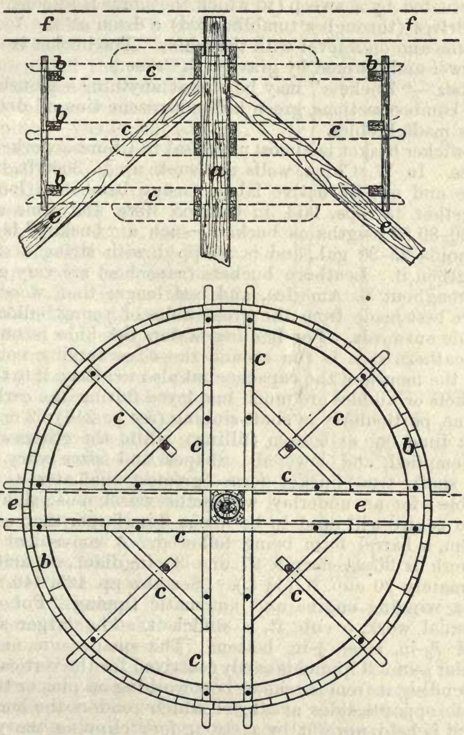
FIG. 180.—WINDLASS AND DERRICK.

A very handy addition to a windlass is used on some of the alluvial tin mines in N. Queensland (Fig. 180). Round a strong central post *a*, a stay *b* is made to swing, by having a loosely-fitted shoulder on its lower end, held at a suitable height by a chain *c* secured to the post, and capable of being moved to any angle by a stout hemp rope *d*, passing from its upper end round a fixed pulley let into the post. From the upper end of the stay, a pulley *e* is slung, from which the bucket rope passes round another pulley *f* at the base of the post, thence to a light windlass *g* at the edge of the shaft, stayed to the upright by two beams. The operator at the windlass thus has control of the bucket, which, on reaching surface, he can pull round and empty on either side, in separate heaps, mullock and ore.

The common form of horse-whim is capable of raising 15–20 t. per hr. from a depth of 240 ft., at a cost of about ·03*d*. per ton-ft.

The Mexican *malacate* is capable of much better things, is composed entirely of wood (Figs. 181, 182), and is extremely serviceable on deep prospecting shafts. The winding drum consists of a shell of boards, fastened vertically to 3 inner rings braced to a vertical centre-post, which revolves with the drum and is pivoted on an iron pin turning in a concave iron or stone plate on the ground. A very common size is $12\frac{1}{2}$ ft. diam. × $5\frac{1}{2}$ ft. high, the centre-post *a* is 12 × 12 in. and 20 ft. long; 3 ring-braces *b* are of two thicknesses 6 × $1\frac{1}{2}$ in. and are counter-sunk $\frac{1}{2}$ in. into the rim; the radial braces *c* are 7 × 3 in., and project through the rim to prevent the cable from falling from the drum when tension is slack; the middle projections prevent the two oppositely-winding cables from interfering. The power-sweeps *e* (2 or 4, according to size of drum and depth of shaft) are mortised into the centre-post near the top of the drum, pass downward between the 3 sets of parallel braces, out at the lower edge of the drum, and terminate 3 ft. above ground in a hook, to which mules are attached, with a seat for the driver. The radius of the circle described by the sweeps is 3 times the radius of the drum. Owing to the great leverage thus obtained, to the exact balance of the weight of moving parts on a vertical axis, and to reverse winding of hoisting cables on the drum—the malacate requires comparatively little power to hoist at a speed greater than is attainable by any other form of whim. Retaining supports consist of massive cap-piece *f* extending in the direction of the cables, and supported by inclined legs, resembling a huge carpenters' "saw-horse." The cap is set just to one side of the centre of the drum, and supports the centre-post *a* in bearings of hardwood bolted to the side of the cap. The only brake is wooden sticks carried by the drivers and braced from the ground to the ends of the sweeps when the malacate comes to rest. A reel on the braces between the sweeps carries surplus cable. Hoisting buckets consist of entire raw hides, sewed together, but leaving an opening to be laced up to retain the load (ore, waste, or water).

The malacate is used to a depth of 1800 ft. With a 16-ft. drum, 9 mules can hoist from 1200 ft. as fast as safety will allow (no guides being used), for a shift of 4 hr. When mules walk, they hoist about 84 ft. per min.; on a slow trot, about 175 ft. per min.—the limit



FIGS. 181, 182.—MEXICAN WHIM.

of safety and economy. On a 600-ft. shaft, 4 mules can easily maintain these speeds, and will work satisfactorily with single cable and bucket. For sinking prospecting or ventilating shafts, or even for development work to a depth of 800 ft., the malacate will remove all that can be broken down, and is cheap in both installation and maintenance. (Nevins.)

Horse-whims of the malacate type are often used on the Joplin (Mo.) lead-zinc mines, but a more usual form is a small whim operated by one horse—the ordinary arrangement of large and small gears actuated by a sweep (to which the horse is attached), which in turn drives (through a tumbling-rod) a drum at the foot of the head-frame and on a level with the whim. The bucket is raised by horse-power and lowered by gravity. (Crane.)

Buckets.—"Buckets" may be almost anything—½-bush. wicker baskets, bamboo sections, green hides, kerosene tins, oil drums, and specially made kibbles.

The wicker basket is almost universal in Chinese-worked Malay tin mines. In W. China, wells are sunk up to 3000 ft. in depth, for brine and oil, by native labour, using bamboo in long strips tied together as rope, laid in bamboo fibre and lime as pipes, and in 50–80 ft. lengths as buckets; such a "bucket" is 3–3½ in. diam., holds 25–30 gal., and is whipped with string at intervals to strengthen it. Leathern buckets (capachos) are very generally used throughout S. America, and last longer than wooden ones. They are best made from the green hides of young bullocks, used hairy side outwards. For hauling water, the hide is cut round, and a leathern lace is run around the edge, serving not only to draw in the mouth of the capacho, but also to attach it to the hook.

Buckets or kibbles are much employed during the early stages of a mine, particularly in shaft-sinking (see p. 248); 2 or 3 are in use at a time, one at bottom (filling), while the others are being raised, emptied, and lowered. Shapes and sizes vary. For a vertical shaft, true bucket form (greatest diameter at top) is admissible; for an underlay, where the vessel must slide on runners, the centre will need to be about 3 in. larger than either top or bottom, a barrel form being followed. A convenient size for heavy work is 36 in. deep × 27 and 30 in. diam., capacity being approximately 20 cub. ft. (1½ t.). (See also pp. 425, 440.) These require a winding engine and automatic tipping. For windlass and manual work, 5 cub. ft. is sufficient. The larger sizes are made of $\frac{3}{16}$-in. steel, ¼-in. bottom. The smaller are more often $\frac{1}{16}$-in. than ⅛-in. Tipping is easily contrived for the vertical bucket by suspending it from a yoke or bow working on pins or trunnions affixed to opposite sides at a level which renders the bucket top-heavy; it is held upright by a strong fork clipping the yoke till forcibly disengaged.

Buckets are often made self-dumping; one of many methods is shown in Fig. 183. The bucket rests on a frame *a*, and is hinged to it on one side *b*; this frame also carries a heavy clip *c*, pivoted, and bearing at its other end a small roller *d*. The bucket is retained in an upright position by the clip *c* embracing a stout pin *e* on its side. At a suitable point in reference to the tip, a guide *f* in the shaft encounters the roller *d* on the rising bucket, depressing it so that the clip *c* is raised from the pin *e*, and the bucket is free to capsize; but this must not occur until a second roller *g* on the bucket is entering a curved channel *h i*, by which the capsizing is controlled. As the bucket assumes a horizontal position, the horns *k* on opposite sides meet a roller *l* and rest on it, while the bucket is further raised till the roller *g* will engage with the curved track *m*, and regulate the bucket during the act of tipping. When emptied, the bucket is simply lowered, and the clip falls back into place and holds it vertical till the next tipping.

When a vertical shaft reaches any considerable depth, gates or crossheads should be used. These slide up and down on the runners, and, as the bucket leaves the last set of timber, the gate encounters a block on the runner, which holds it until the bucket comes up again, the rope in the meantime passing through two holes in the top and bottom bars of the gate. For a gate to be successfully used, the poppet legs must stand at least 20 ft. above the landing stage where the bucket is to be tipped; and there must not be the least likelihood of the gate sticking at any time during descent and then falling suddenly, as numerous fatal accidents have been caused in this way.

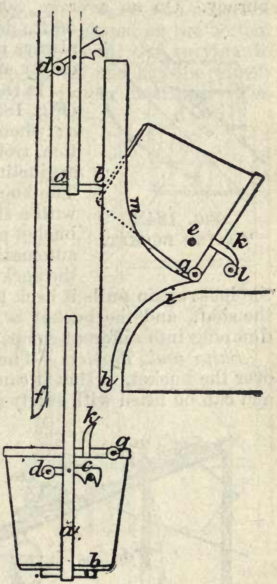


FIG. 183.—TIPPING BUCKET.

A self-emptying bucket for incline shafts, used especially for baling water, Fig. 184, is raised on skids till stud *a* falls into notch *b*, when, on loosening rope *c*, the bucket empties; further hoisting raises stud *a* out of notch *b*, above the lever *d*; this, when released by passage of the pin, falls or is thrown back, bridging notch *b* for return of bucket.

Many forms of the Dawson carrier (see p. 314) are in use, all on the same principle, viz. running a trolley carrying a bucket up a wire-rope tightly stretched between shaft and dump, where, by tightening a rope attached to underside of lip of bucket, it is capsized. On slackening the hauling rope, the bucket is righted, and the trolley returns by gravity along the rope to mouth of shaft.

Arrived here, the trolley is locked; the pulley to which the bucket is attached is released, and descends by gravity. On hoisting, the tightening of hauling rope, when bucket has arrived on top of shaft, unlocks trolley, but locks pulley holding load to car, and trolley proceeds to discharge its load automatically at end of journey. On an average, 300 buckets can be raised from shaft 50 ft. deep in 10 hr., if a loaded bucket is always ready at bottom to be exchanged for empty arriving.

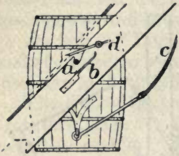


FIG. 184.
TIPPING BUCKET.

In the Fairbanks-Morse self-tipping bucket (Fig. 185), the cable runs over movable sheave a; when the bucket reaches the proper position, trolley b carrying the sheave moves up an inclined track until caught by hook c. The bucket is landed on shelf d, provided with a slot into which a chain on bottom of bucket passes. On releasing a clutch, shelf d automatically dumps; ball e on a chain holds the bucket while being dumped. When empty, the hoist again pulls it back to the trolley, the hook returns it over the shaft, and the bucket is lowered. This arrangement permits dumping into different compartments quite automatically.

Skips and Tipplers.—The skip possesses marked superiority over the bucket, in that it can be made of much greater capacity, and can be fitted with safety-grips and checks to overwinding. In sinking, however, skips necessitate timbering being kept within 15–16 ft. of bottom, and they afford no benefit unless winding is rapid. In many instances, skips replace trucks for regular working. In vertical shafts, they are for the most part provided with shoes, like cages, for embracing guides; in inclines, they travel on wheels; and wheels are necessary where the shaft is partly vertical and partly inclined. There is endless variety in the construction of skips, but their chief interest centres in self-tipping arrangements. They are especially applicable where output is large, loading is effected from shaft-bins (see p. 436), and tipping can be performed at shaft-head. Incline shafts are particularly adapted to them, hence they are much used in S. Africa. The body is there generally made of $\frac{3}{8}$-in. steel plate riveted to lengths of angle-iron, and the back is cut away to facilitate filling, according to degree of inclination. When less than 30°, the bottom plate is

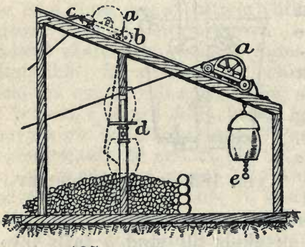
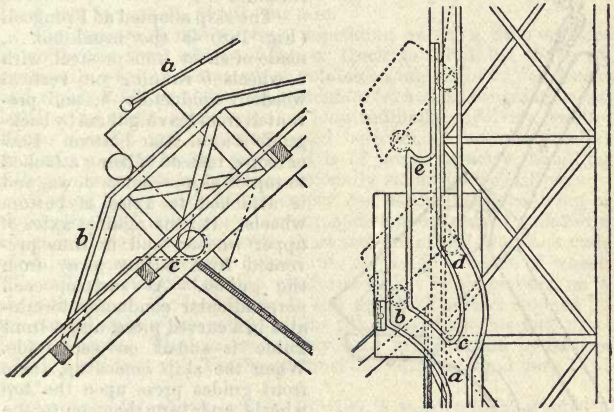


FIG. 185.—TIPPING BUCKET.

about 9 ft. long, and the back about 4 ft.; the back plate at mouth of skip is set back from bottom at an angle of 45°, or, where the shaft is steeper, at 30°, and sides are cut away at same angle; with bins facing down shaft, the back need not be cut away so much. A false bottom of 1-in. wood packing, covered with steel plates, which are bolted through, is sometimes laid on the bottom. The capacity may vary from 20 to 60 cub. ft. or more, and a load of 3 t. is common. The two pairs of wheels are placed on the bottom plate, usually about equidistant from either end, but varying to suit the tip. The better practice is to shrink and key the wheels on axles, so that the latter may run in proper bearings. The



FIGS. 186, 187.—SELF-TIPPING SKIPS.

wheels (preferably cast-steel) are about 15 in. diam. The tipping contrivance generally necessitates the back wheels having a face at least twice as broad as the front—say $5\frac{1}{4}$ and $2\frac{1}{4}$ in. Strong trunnions, held by stiffening plates, are fixed on the sides near the lower end, or a powerful loop is made on the end itself, and draw-bars of 3 or 4 × 1-in. best malleable iron support the whole load.

In incline shafts, tipping is very simply accomplished by causing the wide back wheels to mount a rising outer rail, or, while carrying the outer rail on at same angle, turning the inner rail over to a horizontal position; either causes inversion of the skip. In Fig. 186, *a*, draw-bar fastened to end of skip; *b*, raised rails catching broad hind-wheels; *c*, inner rails followed by front-wheels turned to horizontal position.

For a vertical shaft, the arrangement is necessarily more complicated, and is akin to that adopted for buckets. Where the skip has to run in both vertical and inclined portions of shaft (Fig. 187), upper and lower wheels pass between rail guides, those for the former turning towards the tip in a gentle curve *a b*, while those for the latter (*c d*) continue upwards, though on a course which accommodates the arc described by the upper wheels as the hoisting is prolonged. Simple provision is made for avoiding accidents from overwinding: should the wheels be hoisted out of their guides, their descent is arrested by a shoulder *e*.

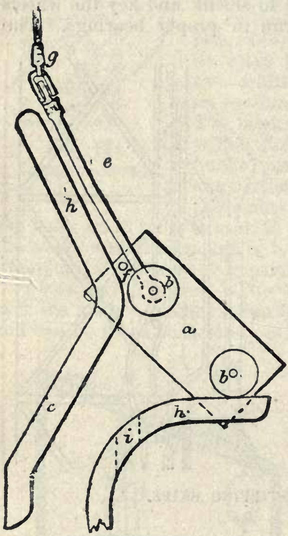


FIG. 188.—FRONGOCH SKIP.

The skip adopted at Frongoch (Fig. 188) is the usual box *a*, made of sheet iron or steel, with 4 wheels *b* running on vertical wooden conductors *h*, and prevented from leaving them by back-guide *d* at or near bottom. Bow or loop *e*, instead of being attached to top of skip, reaches down, and is attached to axles of bottom wheels. It rests against axles of upper wheels, and is thus prevented from falling away from the guides. At surface, each perpendicular conductor terminates in a curved piece, and a front guide is added on each side. When the skip comes up, these front guides press upon the top wheels, and turn them on to the flat ends of the conductors. Partial cutting away of conductors at *i* enables back guide to pass through; bottom end of skip is raised, and contents are tipped into bin or pass. If the engineman does not stop at once, the skip is simply drawn a little way up, resting upon front guides *c*, stop or stud *f* preventing it from assuming a wrong position. As soon as the engineman begins to lower, top wheels drop upon flat ends of conductors, and, pivoted upon these top wheels, the tail end of skip drops, the back guide passes through slot *i*, and the skip, resuming upright position, descends. One great recommendation of this system is that it can be applied to any existing shaft—vertical, inclined, or crooked.

An excellent skip is made by the Grange Ironworks, Durham.

At the Liberty Bell mine, the skip is expected to handle a maximum of 400 t. of rock per double shift of 16 hr., and to hoist and lower 70 men on each 8-hr. shift, with all necessary timbers, rails, pipe, etc. The skip carries $5\frac{1}{2}$ t. of ore, and a cage, linked to it, is made long, so as to carry 14 men on 7 seats. For very long timbers or rails, the swinging cage floor can be laid flat against back of cage, the seats (loosely laid in place) can be laid back, and rails or timbers can be loaded in the skip with their upper ends in the cage.

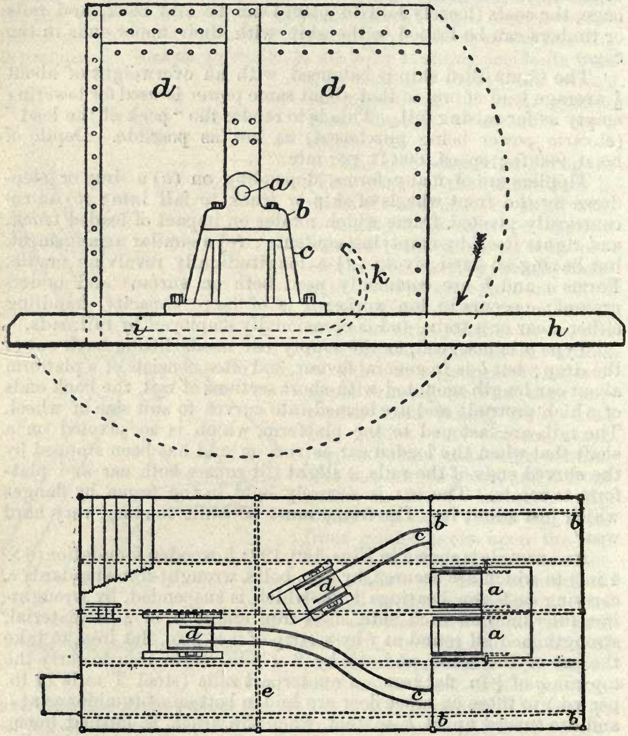
The Camp Bird skip is balanced, with an overweight of about $\frac{1}{2}$ average load of ore, so that about same power is used for lowering empty as for raising full. This is to render the "peak of the load" (electric power being purchased) as low as possible. Depth of hoist, 900 ft.; speed, 600 ft. per min.

Tipplers are of many forms, depending on (*a*) a drop or step-down for the front wheels of skip or truck to fall into; (*b*) an eccentrically pivoted frame which rotates on impact of loaded truck, and rights itself by emptying contents; (*c*) a similar arrangement, but having no fixed pivot; (*d*) a longitudinally revolving cradle. Forms *a* and *b* are commonly used both on surface and underground; *c* occurs in top works; *d* is of large capacity, handling either a car or a train, and is occasionally employed by railroads.

Type *a* is not good, as the empty car needs lifting back out of the drop; but *b* is in general favour, and often consists of a platform about car length mounted with short sections of rail, the back ends of which protrude and are formed into curves to suit size of wheel. The rails are fastened to the platform, which is so pivoted on a shaft that when the loaded car is run on and has been stopped by the curved ends of the rails, a slight tilt causes both car and platform to rotate. The car is securely held in the frame by flanges which just admit it. The frame must be built to stand very hard wear.

An example is shown in Figs. 189, 190: *h*, wooden foundation (6×4 in.) to which are secured, by 1-in. bolts, wrought-iron standards *c*, carrying cast-iron bearings *b*, in which is suspended, by wrought-iron fulcrum *a* on each side, sheet-iron cradle *d*, of $\frac{1}{8}$-in. material, strengthened all round at *f* by a strip of $2 \times \frac{3}{8}$-in. flat iron, to take the rub of the truck, and at *e* by $2 \times \frac{1}{2}$-in. angle iron to carry the top rim *g* of $\frac{1}{4}$-in. flat iron. Counterpart rails (steel T rails 14 lb. per yd.) to those on brace floor are laid in bottom of tumbler as at *i*, and are turned up at *k*, so that, when the truck is pushed home, tumbler and truck at once describe a semicircle, thus dumping the contents, while the swing brings the empty truck back upright.

In the tippler for incline shafts, Fig. 191, truck *a*, running on 4 wheels *b*, is made to ascend a slight incline just before tipping, being drawn by bow *f* attached to rope *j*. The bow is also rigidly united to axles of hind-wheels, and in front it carries door *i* of truck; *k* is the track at top of incline, and *p* is an additional outer line of rails laid on a steeper grade. When the truck in its upward course reaches point *l*, rails *p* pick up small outer wheels *c* on hind axle, so that the hind-wheels travel up the steeper grade while the front-wheels follow rails *k*. Consequently the truck is tilted forward, and, as door or front end *i* is attached to and rises with bow



FIGS. 189, 190.—TRUCK TIPPLER.

f, the contents are shot out. A stud *g* prevents the truck being drawn too far. On slackening rope *j*, the truck rights itself, and descends properly, door *i* automatically closing.

Cages.—In very many cases, ore has to be raised in trucks, because it must be trammed some distance before tipping; and in the great majority of mines shaft-bins are not provided, so that the mine trucks must come to surface. Under these circumstances, a cage is used to carry the truck. This is a light steel framework, with a floor carrying rails. It may be adapted to run in a vertical shaft, or in one inclined at any angle; and may contain one or more trucks, being known as single-deck, double-deck, etc., but in metalliferous mines single-deckers are general, because the output is not in excess of their capacity.

As mines get deeper, economy demands the use of 3- or 4-, or even 6-deck cages. The prevalent objection to multiple-deck cages is the time required in decking, even if carried on simultaneously from two landings or levels. To deck from 3 landings or levels at once makes very complicated stations underground, and is not desirable. With facilities for accommodating both loaded and empty cars at landings, with sufficient help to quickly handle cars, there is nothing to prevent loading and unloading a 6-deck cage in 30 sec., or 5 sec. per deck. A steel multiple-deck cage, with only one car on a deck, can be very light and yet stiff and strong, with ratio of cage-weight to that of contents very low.

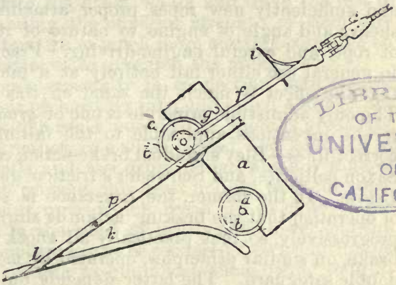


FIG. 191.—SELF-TIPPING TRUCK.

When long drills, timbers and rails are carried in cages, a 1-ft. well in the floor is very useful.

For large outputs, automatic caging saves both time and labour. Sometimes the cars run off of themselves, so soon as the catches are released, the cage floors being inclined; or they are pushed off by empties, which are pushed on by a plunger acting through hydraulic pressure or compressed air; or the cage floor is hinged at one side, and assumes a slope when the cage comes down on the catches. At shaft bottom, similar methods may be employed.

"Safety" mechanism.—The Patent Office records and the technical newspapers are crowded with examples of "safety" appliances destined to arrest a falling cage in case of the rope breaking, or to prevent overwinding; and the mining enactments of many countries render compulsory the use of such "safe-guards." Yet, probably, there is no mechanical engineer who would willingly trust himself to a trial of their reliability, and no mining engineer who really believes in them. For small prospecting shows, working with limited capital, at shallow depths, where the hoisting is slow and engine-drivers are none of the best, they may serve a purpose, and avoid or mitigate an accident once out of ten times that they are called upon. But for large-scale, deep, rapid-hoisting undertakings, they may be condemned as useless, if not worse, for their presence may lead to neglect of the really vital things—good and sufficiently new ropes, proper attachment, proper relation of sheave and angle of engine to nature of rope, efficient inspection of rope, and careful engine-driving. Practically all the devices in general use either fail entirely at a pinch, or act so suddenly that the effect is much the same on the occupants of the cage. The only admissible principle is one of *gradual retardation.* Two recent inventions applicable to the falling cage seem to follow correct lines—Bley's gradual brake-action catch used in some of the Saxon collieries, and Schweder's friction gear tried at Langlaagte Estate. In the former, the resistance is so spread over the field of operation that the braking action is slight at first and increases progressively, so that practically all shock is avoided. A second brake, on similar principles, operated by men in the cage, forms a double safeguard. The latter depends on a cylinder of carbonic acid under a pressure of 1000 lb. per sq. in. acting on pistons which force flat shoes against the shaft guides, and operate independently of the speed of the falling cage. But the momentum of a 3 to 5-ton load travelling at 1000 ft. per min. is liable to produce unexpected results, even if the "catch" acts—such as dislocation or ignition of the shaft timbering. So with prevention of overwinding—unquestionably the best remedy is to give plenty of headroom, to construct the headgear fully strong, and then to arrest an overwound cage by causing it to pinch itself between gradually tightening timbers which will grip it against falling when the rope finally breaks with the strain. The shock of stopping will be quite enough for most people, even though it be moderated in that way. German theorists are loud in praise of a safety-catch invented (and patented) by Prof. Undeutsch: it remains to be seen whether practice supports theory in this case better than in older forms. Anyhow, the *real* remedy is not to overwind. The liberated (broken) rope may easily wreck the engine-house, and cause fatal injuries.

Inasmuch as compressed air is available at most mines, it is surprising that an effective air-brake, controlled from the cage, has not been introduced.

Ropes.—Upon the initial quality of the rope, the care bestowed on it, and its sufficiently frequent renewal, will always depend the safety of the men using it and the freedom from accident. So long as ropes are employed in hoisting, they constitute the principal factor in depth, speed, load, security and cost attainable. No external "safety" appliance and no precaution of any kind will atone for overloading, overworking, or in any way neglecting the rope. Every other consideration is subsidiary to the rope.

Kinds.—Though the hemp and aloes fibre ropes made in Flanders still survive in Belgian mining, it may now be said that for all serious mining the steel wire rope is universal. While there are several peculiarities in the method of "laying" the rope, and several varieties of steel and steel alloys used in the making of the component wires, the shape is in nearly all cases round and uniform —i.e. cylindrical. The intricacies of manufacture do not concern the miner—he should state his exact wants and leave the rest to the maker. Notwithstanding the superior strength of steel wire over iron, some mine waters are so much more actively corrosive on the former than on the latter as to give iron the preference.

Flat and tapered ropes, especially in connection with conical drums, theoretically admit of hoisting from unlimited depths; but practically, with the few flat ropes now in use, there is great risk of their slipping off the nether coils when lapped, and jamming in the space between the coils and the drum flange, and any kind of guide or jockey-pulley to prevent this must unduly wear the rope. In practice, moreover, most ropes which break do so close to the load, which would point to the need of making the lower end the larger. The round tapered rope may be rendered free from trouble in winding, but makers hesitate to guarantee them owing to difficulties in their manufacture. The safe working strength of a flat rope is about $\frac{1}{3}$ less than that of a round rope of equal weight per ft.; its life is shorter, and cost of repairs and maintenance is greater, due to necessary periodic renewing of sewing wires. Tapering ropes greatly increase the working limit of depth for rope of given diam. at drum end. Their first cost, however, is higher than for ropes of uniform section, and though, by their use, the starting moment is reduced, still they afford but little equalisation of load. They are and have been but rarely employed, and the fact remains that cylindrical ropes have already been used for much deeper hoisting than any tapered rope yet tried, and that the best mechanical engineering opinion is wholly in their favour.

Loads.—The ultimate tensile strength of plough-steel varies, according to its temper, from 200,000 to 300,000 lb. per sq. in.: but the elastic limit of very hard, highly-tempered steel makes it inadvisable to exceed 250,000 lb. per sq. in. Beyond this, the metal becomes extremely hard and brittle, demanding very large drums and sheaves, and careful avoidance of abnormal shocks and strains caused by too rapid acceleration in starting the load. With this limit of tensile strength, and a factor of safety of 6, a rope 1·52 in. diam. may be used for hoisting a net load of 8000 lb. in a skip weighing 5000 lb., from a depth of 6000 ft.

Makers guarantee ropes with a breaking strength of 127 t. per sq. in. It is possible, by using nickel steel, to obtain still greater strength, but this has not yet been practically done. To be safe, a breaking strength of 120 t. per sq. in. will be assumed; on this basis, for rope of 6 strands, 19 wires each, and hemp main core, Smart has calculated the following tables:—

Rope diam. required: net Load 8000 *lb., Skip* 5000 *lb.*

Depth.	Factor of Safety.			
	4	5	6	7
	Diam. of Rope.			
Ft.	in.	in.	in.	in.
3000	0·8498	0·9835	1·1180	1·2569
4000	0·8903	1·0478	1·2154	1·4004
5000	0·9372	1·1267	1·3437	1·6077
6000	0 9923	1·2266	1·5236	1·9471
7000	1·0886	1·2810	1·9438	2·0317
8000	1·1401	1·5458	2·3270	2·5327

Rock Load possible with 1·3485 *in. diam. Rope, Skip* ⅝ *net Load.*

Depth.	Factor of Safety.			
	4	5	6	7
	Net Load.			
Ft.	lb.	lb.	lb.	lb.
3000	19,994	14,929 0	11,553·0	9,140·6
4000	18,218	13,154·0	9,775·5	7,363·4
5000	16,440	11,375·0	7,998·2	5,586·2
6000	14,663	9,597·8	6,221·0	3,809·0
7000	12,886	7,281·0	4,443·8	2,031·7
8000	11,109	6,043·4	2,666·5	254·5

These show that with a factor of safety of 6, a rope 1·3437 in. diam. can safely raise a load of 4 t. from a depth of 5000 ft., and a rope of 1·5236 in. diam. from 6000 ft.; and that, with a load of men of 4000 lb. in a cage of 2500 lb., men can be hoisted from 5500 ft. with a rope 1·3435 in. diam. and factor of safety 7. With a cage of 4600 lb. and a rope 1·5 in. diam., and for hoisting men a factor of safety of over 7, and for rock of over 6, 2 t. weight of men, and 4 t. weight of ore, may be hoisted from a depth of 5500 ft.

Loads have grown from 1 t. to 5½–7 t., in order to secure increased output without undue acceleration of hoisting speed.

If the bending stresses of plough-steel rope be cut down by adopting a ratio of sheave to rope diam. of say 150 to 1, the factor of safety of the rope may reasonably be reduced from 6 to 5, provided ropes be carefully inspected each day, and renewed promptly on showing signs of weakness. For a given load, therefore, size of rope may be reduced, decreasing total dead weight to be hoisted, as well as variation in load on engine due to varying weight of rope. The hurtful stresses on rope due to abnormal strains inseparable from rapid hoisting usually are less for long ropes in deep shafts than for similar work in relatively shallow shafts. A long rope has greater total elasticity, and acts like a spring between cage and drum. The severest hoisting stresses are produced by starting a load with slack rope. On picking up a load with only 4–6 in. of slack rope, a strain is produced equal to twice the static load. This may be materially reduced by interposing strong springs between rope and cage. But, with an elastic, 19-wire stranded rope, it is precisely in deep shafts that these shocks are of least importance. For shafts of moderate depth, it is customary to use a factor of safety of 8–10, and sometimes even higher. If a low factor of safety be adopted, it should be with full realisation that the rope must be installed and cared for in a more perfect manner than usually obtains at mines. (Prof. Peele.)

Deterioration.—The material of every rope, whether fibre or wire, undergoes deterioration in working, diminishing the rope's strength till it becomes unsafe. This deterioration of material is something more than mere wear by friction or rusting; it is a sort of fatigue, and in wire ropes, even where testing fails to show any loss of tensile strength per sq. in. of section, there is diminution of pliability and elasticity: the wires become harsh and brittle, whereby the rope is weakened. Though serious deterioration is generally accompanied by unmistakable external indications, it is desirable to trace its progress by actual tests of the individual wires, or of the ends of the rope itself. Deterioration commences at the time the rope is wound on the drum and started with its first load. A few broken wires in different parts may not greatly weaken it; but when several are discovered close together, it means that they have become brittle, and this condition should cause immediate condemnation of the rope. Internal corrosion may arise from simple moisture due to access of air (even while stored), or from action of acid mine waters; to detect its presence, the strands must be slightly untwisted, to enable the interior to be inspected. Evidences of inside corrosion appear on the outside of the rope in looseness of the wires (caused by their not bearing equally): at first the wires can easily be moved aside on the crown of a strand by the insertion of a knife; after corrosion has gone further, the wires spring up, and can be pressed down with the finger. When a rope shows these signs, it is impossible to know how many wires are bearing the load, and it should be removed at once. Acid action expressed by pitting is not considered so dangerous, as it can be readily seen, and is equally outside and inside the rope. The only unfailing method of preventing acid action on wire ropes is to keep the water from them.

Diameter is reduced both by wear and by internal and external corrosion. Violent vibrations induce considerable elongation in ropes bearing heavy loads, and, if long continued, or repeated at frequent intervals, may induce "permanent set" and serious weakening. Reduction in cross-section of ropes can be approximately found by simple measuring, and this done periodically would often give warning of impairment not otherwise noticeable. Careful records should be kept of every rope, giving date of manufacture, exact diameter when new, length of time employed, actual work done (tons hoisted), and description of any important change noticed during periodical inspections. The life of a hoisting rope is influenced by the use to which it is put and the care it receives. A rope kept free from moisture, well lubricated, wound singly on a drum of large diameter, and worked at slow speed, will outlast one carelessly worked at high speed, over small sheave and drum, and allowed to get damp. The quantity of material raised or work done is one measure of a rope's durability; the time it is employed, and the conditions to which it is subjected, are another. Ropes that have been stored for any length of time need to be thoroughly tested, as oxidation by atmospheric moisture is certain to occur on the interior strands; hence, freshly-made ropes should be secured whenever possible, and such storage as is compulsory should be extremely careful. Ropes winding in a single layer on a grooved drum wear much less than those wound on in several courses. In the latter case, also, the rope is subject to shock when it arrives at the drum flange, and has to mount suddenly on a coil just completed. A rope in a vertical shaft, running over large sheaves and drums, especially if winding on the latter in grooves in a single layer, is subjected to very little wear, while one working in an incline shaft wears generally very rapidly. Corrosion is more active with ropes in inclines, since they are more liable to pick up moisture from the ground and drippings from the roof.

In connection with rope breakages, it is most remarkable that they occur almost always in precisely that section of the rope which is never wound—viz. between the cage and the poppet-sheave—towards the end of the ascent, and often with a much smaller load than usual.

Care of.—Careful maintenance is indispensable to the preservation of all ropes, especially of wire ropes. In uncoiling, it is most important to prevent kinking: the coil should be placed on a reel or turn-table, and the rope drawn off from the outside. Wire ropes should be kept in a dry place, and laid upon timbers. They should be oiled over occasionally. Much damage often results from corrosion through improper storage. Hemp ropes want tallowing regularly, and aloes ropes want keeping always damped. Wire ropes, steel particularly, should be greased regularly, and often enough to prevent their ever beginning to rust. The grease should be sufficiently soft to work into the strands, but so stiff as to stick on the outside of the rope. A mixture of oil and grease, well stirred and laid on hot with a brush, answers very well; both oil and grease should be neutral. Before lubricating, it is absolutely essential to clean the rope thoroughly, and examine it for corrosion. Ropes should be particularly looked after in this respect when used in wet and in upcast shafts. In the latter case, vitiated moisture-laden air has quite as great an effect as mine waters in corroding a wire rope. Experiments have demonstrated that a well-lubricated wire rope will endure between twice and six times the use of an ungreased rope.

Every rope used for hoisting men should be inspected full length daily, and no longer be so used when strained, frayed, or spliced. At 3-monthly intervals, the rope should be reversed, and have about 6 ft. cut off the cage end, using that 6 ft. for examination and test of wires. Failing this, it should not be used for more than a year where the speed, load, and duty are great; under strict observation, it will be safe for 2½–3 years probably. In inclines, the lower end of the rope receives much extra wear, because it has

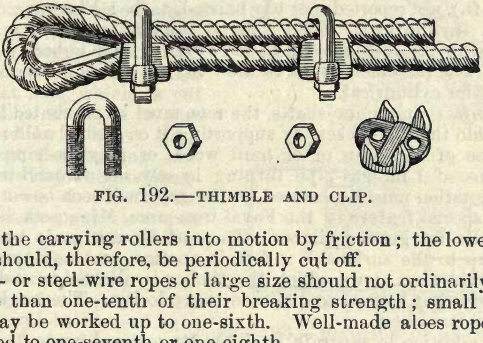


FIG. 192.—THIMBLE AND CLIP.

to start the carrying rollers into motion by friction; the lower 100–200 ft. should, therefore, be periodically cut off.

Iron- or steel-wire ropes of large size should not ordinarily work at more than one-tenth of their breaking strength; small round ropes may be worked up to one-sixth. Well-made aloes rope may be loaded to one-seventh or one-eighth.

Attachment.—The attaching of the rope to the cage is perhaps the most critical point in ensuring safe hoisting. In doubling back the rope end, the loop should be kept as large as possible, by inserting an iron eye or thimble. Splicing of the end is a very difficult job, and often performed in a slipshod fashion, though the whole security of the rope is dependent on it. For this reason, the splice is better replaced by a strong screw-clip. A great variety of clips are in use, many being most destructive of the rope by employing rivets which penetrate it. The only safe principle is one which permits of complete inspection at all times, such as the Crosby (Fig. 192). This clip, forged out of 30-t. tensile-strain steel, withstands hammering, bending, and frost. Even the socket, run in solid with white metal, as approved by many engineers, may possess elements of weakness which are not divulged till a break occurs.

Shackles should be changed and re-annealed every 6 months, as repeated jolts, when the rope tightens on the cage, render steel very brittle. The greatest strain on a hoisting rope occurs when the load is suddenly started or stopped. This sometimes amounts to 10 times the working load, and in addition to exceeding the "factor of safety," causes the metal to become "fatigued" and brittle. Sudden application of the brake is injurious, and starting and stopping should always be done slowly. Any rope which has been subjected to an unusually great strain in this way should at once be replaced, one or more weak spots having certainly been created. The straining caused by ordinary starting and stopping may be greatly diminished by employing a bridle chain between the rope and the cage, and especially by spring couplings, of which there are many kinds. They need to be strong, simple, and absolutely reliable, as, being mostly out of sight, they suffer much neglect.

Costs.—Rope costs on some of the deepest shafts in Europe (3000 ft.), are reported, per ton hoisted, to be: Flat rope, 1·188*d.*; round (cylindrical), ·638*d.*; round (tapered), ·357*d.*; or, at per ton per 100 ft.—·04*d.*, ·021*d.*, and ·011*d.* At Kimberley, hauling from 1200 ft., official figures are ·7*d.* per ton for flat ropes, and ·066*d.* for cylindrical.

Shafts.—In incline-shafts, the rope must be prevented from cutting into the sole-pieces, by supporting it on dished rollers; these may be of cast iron or of hard wood, or may be simply short sections of 1-in. gas-pipe turning loosely on a round-iron core. Free rotation must be ensured, or the rope will soon be cut.

A special feature of the Fayal iron-mine, Minnesota, is the use of slides for getting timber. These slides are made by putting rises up to the surface, 5 × 5 ft. in section, and equipping them with skid-ways, down which the timber is allowed to slide; they are curved at bottom, to diminish the velocity. The angle of 38° is considered best.

In addition to separate landings for ore and waste, it is sometimes convenient to have a third for drills.

When a shaft is partially vertical and partially inclined, especially if the change is sudden, and to an inclination of 30° or less, bins are provided at the junction, and hauling in the two portions is made separate and distinct, thus greatly increasing the delivering capacity of the shaft, and avoiding excessive wear and tear on rope and gear.

Fig. 193 shows a more recent (Rand) arrangement for deep level shaft connection between vertical and incline. In the first row of deep levels, and in some deeper deeps, it is possible and more economical to continue the vertical shaft round a curve to the incline. An engine winding loads from a 2000-ft. vertical shaft can deal with them on such inclines usually to the bottom of the mine; but as depth increases, the mass of rope on the drum becomes cumbersome, and the moment of inertia (on starting) large. As a rule, the curve takes the form of an arc of a circle, which is too abrupt for a change in direction of, say 120°, and involves considerable expense in maintenance. A better curve is the cubic parabola. Good guide pulleys are necessary. (Pettit.)

At all stages where a cage is normally brought to rest for loading and unloading—known as plats—provision is made for supporting it on chairs or keps, so that the floor is exactly even or flush with the flat-sheet or rails from which the truck is run in. There is a variety of pattern, but the principle is always the same: wooden or iron shoulders strongly and securely hinged to side-timbers of shaft beneath plat, and controlled by simple hand-lever. In an inclined shaft, that section of the plat floor which is cut away to allow the cage to pass is occupied by a heavy door, covered with flat-sheeting, and counterpoised by weights hanging away from the shaft; the counterpoise keeps the door open and the way clear till forcibly closed, and the cage abuts against it on descending.

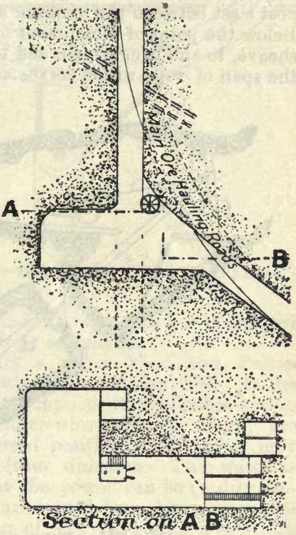


FIG. 193.—JUNCTION OF VERTICAL AND INCLINE.

The top of a vertical shaft should always be covered by a "bird-cage" or substantial lid which rides on the rising cage and falls back into place as the cage descends again. Or a gate, sliding up and down, or swung on hinges, may serve the same purpose of preventing falls into the shaft. At the various plats, the gate generally consists of a single bar hinged at one end and dropping into a loop or slot at the other; but this is a very imperfect contrivance, insufficient to prevent a person slipping into the shaft, or to hinder a loaded truck from being capsized into it. Where large outputs and rapid winding are the order, as in collieries, automatic gates are employed; various forms are illustrated and described in the 'Colliery Guardian,' Nov. 25, 1898, and Jan. 20, 1899.

Shallow Winding.—A simple and effective solution of the problem of hoisting dirt while sinking an incline shaft has been arrived at in the Lake copper mines by the installation of a gravity system of haulage and hoisting on a rope track (Fig. 194). It consists of a wire rope *a* (1 in. diam.) fastened to a stubbing bar *b* fixed at the lowest point in the shaft excavation, on the manway side, carried somewhat in advance of the main excavation, while the other end, after being drawn taut, is securely attached to a post *c* set between the hanging and foot-walls at the station above. Below the point of attachment of rope to post, is mounted a small sheave, to and over which, and thence to the carrier *d* operating on the span of rope which serves as a track, a rope passes from the winding engine *e*. The bucket is supported on a block, and is raised and lowered by the engine. The weight of the bucket is such that the carrier remains in the position at which it was stopped in the shaft, and does not begin to travel upon the track cable until the bucket has been raised as high as it will go, which is when the block supporting the bucket strikes the carrier. Continued hoisting draws the bucket, supported by the carrier, up the shaft, free from contact with any point until it reaches the inclined timbers, upon which it skids, and, on clearing them, rises above the station floor, when it can readily be dumped into the skip standing in the hoisting shaft. The bucket is returned to the foot of the shaft by gravity, regulated by brake on winding drum. On reaching the lowest point that the carrier can go, further lowering permits the bucket to be dropped to the bottom of the shaft. Details of the carrier are given in Fig. 195. (W. R. Crane.)

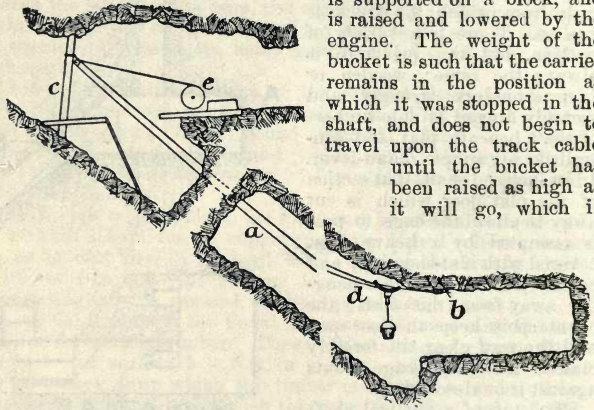


FIG. 194.—ROPE HOISTING WHILE SINKING.

A somewhat similar system of combined hoisting and tramming is used in the Joplin lead-zinc mines. It consists of a wooden rail supported on bents or trestles increasing in height from the shaft millward, thus giving the rail a grade of 5–12 %. Height of rail above shaft is 15–35 ft. and upward at mill end, where a stop-block prevents overwinding, while at the shaft-end there is attached to the rail an adjustable trip and stop. Upon the rail operates a 4-wheeled bucket- or car-supporting truck, which consists of an automatic latch arrangement combined with sheaves, by means of which the carrying receptacle is supported while the truck is passing between the termini of the system; when it reaches a point directly over the centre of the shaft, it is stopped, the latch is tipped, and the bucket or car is lowered into the shaft. The system is entirely automatic.

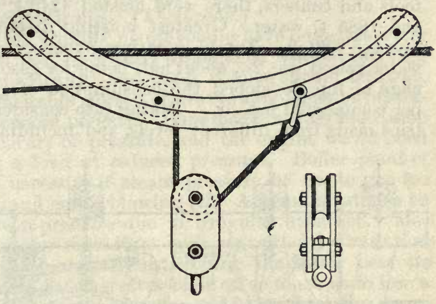


FIG. 195.—DETAIL OF BUCKET CARRIER.

Hydraulic Hoists.—Occasionally a water head and Pelton wheel can be directly applied to winding, as in Fig. 196. At the Utica mine, California, each shaft is supplied with a double-acting water-power hoist. Two Dodd water-wheels, 9 ft. diam., are placed on the pinion-shaft in reversed position; thus power may be applied to run the hoist in either direction. The water is delivered through 3 nozzles, so that the power can be readily proportioned to the load; the nozzles are served by a 10-in. wrought-iron pipe, with water under a head of 420 ft., giving a standing pressure of 180 lb., which falls to 130 lb. when the hoist is in operation. These hoists may be built with engines and water-wheels combined, so that water can be used when available and steam when it is not.

These machines are in use on some of the New Zealand alluvial gold mines for moving boulders. Under a head of 250 ft., they lift 2-t. boulders at the rate of 1 ft. per sec., and a cost of 2*d.* per ton.

Oil and Gasoline Hoists.—These are sometimes of great utility on small isolated mines. One used at Pilbarra, W.A., was a 24-b.h.p. oil engine, continuous-running, geared to a winding-drum and pump-crank. It ran at 180 rev. per min., and hoisted water at 200 ft. per min. from an average depth of 120 ft., using Java petroleum costing nearly 3*d.* per pint on mine. Hoisting cost per 1000 gal. was about 3*s.* (Parker.) Another, installed at Durango, Mex., at 6000 ft. elevation, had a 10-in. cylinder × 12-in. stroke; average gross load hoisted was 1100 lb. During a year's run, there were consumed 2965 gal. of gasoline, including losses by leakage and evaporation. The hoist was run 535 shifts of 10 hr., making the hourly consumption of gasoline ·554 gal. Cost of gasoline at the mine was 2*s.* per gal. Besides raising and lowering workmen, tools and timbers, there were hoisted 4246 t. waste rock and ore, and 1885 t. water. Greatest hoisting distance was 400 ft., and average 322 ft. Total operating expenses per ton of rock and water raised was 2*s.*, about half being for gasoline. Reducing tonnage to h.p. developed, the consumption of gasoline was ·25 gal. per commercial h.p.-hr.. Taking into consideration the number of trips made from different levels, and including weight of buckets

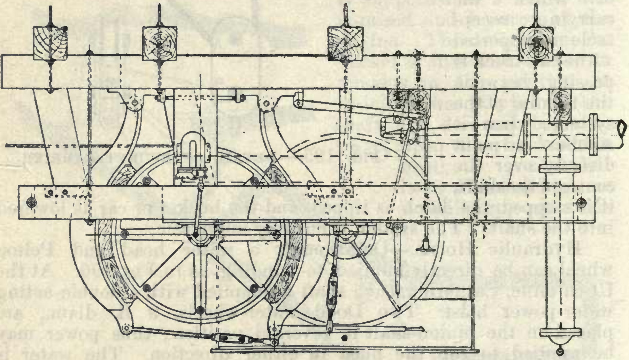


FIG. 196.—WATER-POWER HOIST.

and cable, the consumption of gasoline was ·19 gal. per h.p.-hr. Notwithstanding the exorbitant price of gasoline, hoisting expenses were less than by a two-cable Mexican whim operated by 4 mules. (Kedzie.)

Deep Winding.—More particularly in connection with future hoisting problems on the Rand, there has been much discussion in Trans. Inst. M. M. and elsewhere of various plans for deep winding; from this, the subjoined remarks have been condensed.

The following requirements should be met by an ideal modern hoisting plant: It should use steam at not less than 150 lb. pressure, because of smaller size and cost of engines required, and fuel economy resulting; it should hoist load from bottom in 1 min., and make a round trip in less than 3 min., no matter what the load or depth; net load should be as large as possible in proportion to dead load, and mass of moving machinery be kept as small as possible; the best-adapted plan of balancing should be used; a plain Corliss or other automatic variable cut-off engine is best, of as high rotative speed as is found good in engine practice; drums should be as small as possible for required hoisting speed and capacity, when driven by high-speed engines; there is no advantage to be gained by special designs of cone-drums, rope-reels, or other hoisting systems over straight drums. (Thompson.)

Boilers.—Boilers for winding plants should have large free water surface, so that steam may be readily disengaged without carrying too much water mechanically; and water volume should be large, so as to provide considerable storage of heat to supplement evaporation during heavy-demand periods. Trips made in close succession allow no time for recovery of pressure, and the engine would need greater size to start a load at reduced pressure. Boiler plant of ample capacity is a necessity if steam pressure be not to rise too rapidly during stops, and entail blowing off. Account must also be taken of drops in steam pressure due to irregular attention. Mechanical draft and damper regulators, both controlled by variation in steam pressure, aid materially in keeping the latter near its proper amount. Where safety-valves blow off so much as to lose a great deal of steam, they might be connected to the feed, the steam condensed at atmospheric pressure being returned as hot water to the boiler. Economy of operation is improved by uniformity in steam demand, and life of boilers is prolonged. Superheating considerably enhances economical operation. A most reliable steam-trap is very necessary, because condensation is excessive during rests. Lancashire boilers are best: though water-tube boilers are economical in fuel, they are not suitable to dirty water. But one loco-type boiler in a battery is often of great assistance, under forced draught, for a sudden extra steam demand. (See also pp. 56–65.)

Engines.—Types of winding-engine are too numerous even to catalogue. For permanent work, a double drum of simple, direct-acting, non-condensing pattern, is most generally suitable. Direct-acting engines are preferred to geared, because at high speeds there is less wear and tear; and, with no reducing pinion between, a drum of less diameter gives the same speed as a larger drum with a geared engine, also the load acts at a smaller leverage, so that, after starting, full speed is more quickly reached. The duty required of a hoisting engine is peculiar, and ordinary rules do not fit. The load consists of weight of ore (total live load), weight of cage, car or skip (constant dead load), and weight of rope (an important percentage of the whole, which is a dead load continually varying in amount, being greatest at beginning of trip and nothing at end). Added to this variable and largely unprofitable loading, an extremely intermittent service is required. Engines must start maximum load, attain full speed in 6–7 rev., hoist continually-decreasing load to surface, and stop short—all within 1 min.; and, after longer or shorter period of idleness, repeat the operation. Under such conditions, ordinary methods of obtaining economy of operation in steam-engine practice are worthless. Experience has fully demonstrated that simple Corliss or other automatic variable cut-off gives more economical results than slide-valve or fixed cut-off, and such engines are universally adopted at deep mines. Attempts to gain economy by compound or triple-expansion engines give indefinite if not negative results. (See also pp. 65–7.)

Speed.—As showing modern tendencies, the Lake copper mines hoist loads from all depths in 1 min., and make round trip in 3. The Quincy raises 8-t. skips at 3000–3500 ft. per min.; and Tamarack No. 2, with a single cylindrical drum 30 ft. 6 in. diam., often attains a rope speed of 5000–5500 ft. per min. in a shaft less than 4000 ft. deep.

Drums and Sheaves.—Large diameter for drums and sheaves is of more importance for wire ropes than for hemp, and for steel than for iron. The smallest diameter should be at least 1300–1400 times that of the iron wire in a rope, and 2000 times that of the steel wire. Its relation to the size of the rope itself matters less, because the disadvantage of too small diameter can be obviated by selecting suitable size of wire and suitable make of rope. It is well, however, for smallest diameter of sheave or drum to be not less than 80–100 times diameter or thickness of wire rope, and 50 times for hemp rope. Greater size of drum, having greater moment of inertia, acts like a flywheel, and reduces fluctuations of speed due to varying engine impulses. With greater moment of inertia, greater distance must be passed over before a given speed is reached; in other words, the velocity reached in a given distance will be less, so that if a certain amount of slack rope has to be taken up in starting a load, the shock will be less with heavy drum, because the speed at which load is picked up is less. With light drums, the greatest proportion of stress due to varying engine effort comes on the rope, being of greater intensity and more nearly in the nature of jerks than with heavy drums. These jerks have a particularly detrimental effect near surface, where the length of rope between load and drum is comparatively short, and therefore much less elastic than when the load is near bottom.

The rope should wind smooth on drums or sheaves, without rubbing sideways against them, and run free from jolts and flapping. For wire ropes, it is generally desirable to line grooves of sheaves with wood, but in cases of enormous duties, as in the Lake copper mines, the grooved wooden blocks in the driving sheave of the Whiting hoist have been replaced by Walker differential rings, which have a much longer life. Abrasion of strands by each other in winding on the drum is a frequent cause of wearing. The bending to which all ropes are unavoidably subjected causes them to unstrand, and the compound wires to become brittle or of unequal length; consequently they are subjected to unequal strains.

Side friction should be minimised by using a long drum of ample diameter. A rope winding in a single layer on a grooved drum will considerably outlast one winding in several layers on a small drum. The baneful effect of reverse bends is very great: the life of a rope which goes over the sheave and under the drum is only 50–75% that of a rope going over the sheave on to the top of the drum.

The larger the diameter of head-gear, the less does it matter how small be the angle which the inclined span winding on drum makes with rope hanging down shaft; but with smaller diameters of sheave, the angle should be increased, in order thereby to diminish bending of rope in passing over sheave. Opinions differ as to minimum angle to be allowed; some assign 40° as the limit, while according to others it should never be less than 60°. In plan, the obliquity of a round rope between overhead sheave and drum should always be kept within the smallest possible limits.

Hewitt's investigations on steel rope have shown that, to reduce bending stresses sufficiently to leave one-fifth of the ultimate strength of the rope available for useful work, the min. diam. of bends must be about 80 times diam. of rope. On this basis, it is safe to say that for plough-steel rope, the sheave diam. should be at least 120 times that of the rope, and preferably even greater. In a $1\frac{1}{2}$-in. rope, the ratio of 1 : 120 would give a sheave diam. of 15 ft., and if 1 : 150, the diam. would be 19 ft. Sheaves 20–24 ft. diam. have been occasionally used in England.

Double drums are preferable to single, affording advantages of balanced working, and allowing each drum to be worked independently. The two drums run loose on same crank-shaft, and are thrown into gear by friction-clutches; when both are in gear, one is lowering whilst the other is raising, one rope being wound over the drum and the other under the drum. When one is thrown out, the other is used as an independent engine.

Drums for round ropes are smooth cylinders, of same diameter as the poppet-sheave, and with sides 1 ft. or so deeper than the tread; a drum should be long enough to take the whole rope without overlapping. On one side of the drum is a brake-rim, upon which act post- or bar-brakes, actuated by steam or by hand; they should be sufficiently powerful to hold the engine against full head of steam. Further brake power is afforded by a band working upon the circumference of the crank-disc. This and both drum-brakes should be close together, and with sufficient leverage to be easily under control of the driver, so that failure of one can be immediately remedied by application of another without the necessity of his moving. Where a drum is thrown in and out of gear by a clutch sliding on the shaft, this portion of the shaft should be square in preference to having key or feathers, and the clutches should be square and not bevelled, so that they do not slip, whichever way the drum is working. It is important to have good indicators, so that the cage can be exactly located, and stopped level with the plats. The indicator should be attached to the drum, and not to the drum shaft.

When a drum is working independently, it is usual to lower by means of the brake; but with two drums balanced, one lowers whilst the other hoists, and the weight on one rope is partially balanced by that on the other, enabling the engine to carry a greater load and consume less power. Moreover, the ropes can be adjusted to suit the different levels at which work is proceeding, as each drum's rope can be wound or unwound as desired. With a single drum, one end of the rope is being unwound whilst the other is being wound up, and to alter the working lengths entails considerable trouble. To overcome it, and permit the use of small drums and round ropes, a hoist (Whiting's) has been devised (see p. 488).

In all cases, the useful load is only a small proportion of the total mass that has to be set in motion, brought to a high velocity, and again arrested during every complete wind through a shaft. Power developed by engines is expended in overcoming frictional resistance of winding apparatus, in communicating velocity to all moving parts, and in raising useful load; and the work is intermittent. To accommodate great lengths of round rope, drums are either of big radius or great length, or the rope is overlapped. Length of drum is limited by the rope at either end getting too far from the plane in which the poppet-sheave runs, and this angling is prohibitively injurious to the rope, and a source of loss and danger. Hence the common practice to employ drums of large radius, which are consequently also of considerable weight, necessitating powerful engines to overcome the weight, and the highly-levered load dependent tangentially from the extremity of the great radius. Large-cylinder engines, with only moderately long stroke, are brought into requisition, with the result that, when at work, peripheral speed is high and piston speed comparatively low, the ratio being, in many cases, 6 to 1, a factor that neither tends to economy of working nor to safety as regards overwinding.

From the principle of the conical drum, it is obvious that, while the difference between end diameters of drum increases with depth of shaft (or length and weight of rope), the axial length of drum does not increase proportionately to diameter of larger end. If the end moments be made equal, the angle which the element of the cone makes with its axis becomes excessive for a very deep shaft, and may lead to difficulty in properly fleeting the rope on the drum, and even liability of the rope slipping from a groove, causing dangerous shock or even breakage. For depths of less than 3000–3500 ft., this is not serious; but for depths approximating 5000 ft., the angle of the drum becomes extremely steep, provided it is really designed to attain equalisation. Thus the double conical drum of No. 3 shaft Tamarack, 10 ft. diam. at small end and 36 ft. at large end (angle of cone being about 52°), is designed to hold 5500 ft. of rope on each side, the portion from 17 ft. to 36 ft. diam. being used for hoisting from 4500 ft.: trouble has been experienced with fleeting of the rope on this drum, and the hoisting speed is kept lower than at shaft No. 2, which is provided with a cylindrical drum 30 ft. 6 in. diam. For very deep shafts, a compromise may be made by not attempting to produce complete equalisation of end moments, but merely to reduce the starting moment by adopting a flatter cone (relatively larger at small end), or by employing a cylindro-conical drum as at Tamarack shafts Nos. 1 and 5. For a double compartment shaft, the larger part of the drum is cylindrical, terminating at each end in a short frustum of a cone. Each rope in turn winds up the conical surface and then over on the middle cylindrical part, the latter being occupied alternately by the over and under ropes. If a taper rope be employed with conical drum, the large diameter of the drum is reduced, and an objectionably steep angle of cone may be avoided. (Prof. Peele.)

In hoisting in balance from many levels or constantly increasing depth, the width involved by the usual construction (2 small drums end to end on one shaft) causes the rope to lead from head-sheaves to drums at an undesirable angle, unless the hoist is situated far from the shaft. Where room is not important, the width of a single 6-ft. drum carrying 2000 ft. of rope, say, 11 ft., does not give an undesirable rope-lead, with the hoist reasonably near the shaft. Where two drums are necessary, the advantages of small-drum design (structural strength and low first-cost) warrant its adoption, and the rope-lead difficulty is overcome by placing the two drums side by side, on parallel shafts (instead of end to end on a single shaft), driving them by side-bars, like the side-rods of a locomotive. Second-motion plants are often designed this way, and the driving pulleys of Whiting hoists are operated with side-rods; no first-motion plant has yet been so constructed, but there is no mechanical objection to it. Furthermore, the small straight drum, driven by high-speed engines, has all the advantages sought by conical drums—ability to start the load with engines of reasonable size, and sufficient rope-speed for hoisting capacity desired. The Whiting system has been much advocated for deep hoisting, on the ground that a balance-rope or tail-rope hanging from bottom of cage renders the load uniform, and admits more economical engines; but a balance-rope can be used on an ordinary balanced hoist just as well. (Thompson.)

To Compute Diameter of Drum using Flat Ropes Overlapping.

Rule.—Divide depth of shaft (in.) by product of rev. $\times$ 3·1416; from quotient, subtract product of thickness of rope $\times$ rev.: remainder = diameter (in.).

Ex.—If engine makes 20 rev., depth of shaft being 600 ft., and rope 1 in., what should be diameter of drum?

$$\frac{600 \times 12}{20 \times 3 \cdot 1416} = \frac{7200}{62 \cdot 832} = 114 \cdot 59 - (1 \times 20) = 94 \cdot 59 \text{ in.}$$

To Compute Number of Revolutions.

Rule.—To area of drum, add area of edge surface of rope; from diam. of circle having that area, subtract diam. of drum; divide remainder by twice thickness of rope: quotient = rev.

Ex.—Length of rope 2600 in., thickness 1 in., diam. of drum 20 in.; what is number of rev.?

Area of 20 + area of rope = 314·16 + 2600 = 2914·16, diameter of which is 60·91, and $\frac{60 \cdot 91 - 20}{1 \times 2} = 20 \cdot 45$ rev.

To Compute Meeting Point of Ascending and Descending Buckets.

To Compute Meeting Point. *Rule.*—Divide sum of length of turns of rope by 2, and to quotient add length of last turn; divide by 2; multiply quotient by half number of rev.: product = distance from centre of drum at which buckets will meet.

Note 1.—Meeting will always be below half depth of shaft.

2.—At half number of rev., buckets will meet.

Ex.—Diam. of drum 9 ft., thickness of rope 1 in., rev. 20: what is depth of shaft, and at what distance from top will buckets meet?

$$\frac{8 \cdot 54 + 38 \cdot 48}{2} + 38 \cdot 48 \div 2 \times \frac{20}{2} = \frac{71 \cdot 99}{2} = 35 \cdot 995 \times 10 = 359 \cdot 95 \text{ ft.}$$

To Compute this Depth. *Rule.*—To diam. of drum add thickness of rope (ft.) and ascertain its circumference; to diam. of drum add quotient of product of twice thickness of rope and number of rev. less 1; divide by 12 for diam.; circumference of this diam. is length of last turn, also in ft.; add these two lengths together, multiply their sum by half number of rev.: product = depth of shaft.

$$9 + \text{thickness of rope} = 9 + \tfrac{1}{12} \text{ of } 1 = 9 \cdot 0833,$$
$$\text{which} \times 3 \cdot 1416 = 28 \cdot 54 \text{ ft.} = \text{length of first turn.}$$

$$9 \cdot 0833 + \frac{1 \times 2 \times \overline{20 - 1}}{12} \times 3 \cdot 1416 = 38 \cdot 48 \text{ ft.} = \begin{cases}\text{length of} \\ \text{last turn.}\end{cases}$$

Then $28 \cdot 54 + 38 \cdot 48 \times \frac{20}{2} = 67 \cdot 02 \times 10 = 670 \cdot 2$ ft., depth of shaft.

Clutches.—Friction-clutches are used on the largest drums in existence, and sometimes under very trying conditions. At the Calumet and Hecla, a 5000-h.p. engine operates 4 drums 20 ft. diam., and over 9 ft. wide, geared to a common line shaft driven by the engine; it runs continuously, like an ordinary mill engine, being also used to drive air-compressors from same line-shaft; the drums are clutched to the driven gears on the drum-shaft by means of large band friction-clutches, which are thrown in with the line-shaft running at its full speed of 52–60 rev. per min., and the speed at periphery of drums is about 1200 ft. when the clutches have seized. Other engines each drive a single cone-drum, the large and small diameters of which are 26 ft. and 14 ft. 3 in., width being 12 ft.; the drums are coupled by means of hydraulic clutches to a large wheel keyed on the shaft. Other two winding engines operate each a cylindrical drum 25 ft. diam. and about 8 ft. wide through spur-gearing, the drums being coupled to the driven gear by friction-clutches; speed of winding is about 2000 ft. per min. The flat-rope engines at several of the deep Anaconda shafts, for hoisting from a prospective depth of 5000 ft., are also operated by clutches, the gross load at present depths being 13 t., and the speed 3000 ft. per min.

Hoisting Men.—The first consideration is safety and speed in hoisting men. Rand statistics show that about 8 men are sent underground per stamp per diem. For a 400-stamp proposition, this means sending down 3000 men or 1500 per shaft (2 shafts being compulsory in the Transvaal) in 24 hr. Assuming 24 men at a trip, the live load is say, 4000 lb., and weight of cage 4600 lb. At a speed of 1500 ft. per min. for hoisting, and allowing 6 min. for each trip, including loading, there will be 62 trips down and 62 trips up with men, say about 80 double trips in all = 480 min. or 8 hr. But the number of men per stamp is greater on the Rand than elsewhere.

It will probably be necessary in all deep winding propositions to use separate winders for men and rock. This will necessitate handling men during long periods, unless extraordinarily large winding capacity for main hoists is provided. Assuming 2 7-compartment shafts, the entire force may be lowered and raised in 2 compartments of each shaft, the remaining 4 hoisting compartments being used for ore only. Practically all the daily running time of the man compartments would be occupied in handling men. If all 6 compartments of each shaft were utilised for men between shifts, the total time occupied would be quite within reason. By using 4-deck cages, with 48 men per trip, hoisting in two stages in a 6000-ft. shaft (practically the same as hoisting 3000 ft. in a single stage), 4 compartments would have a capacity of 2000 men per hr., or 3000 men for 6 compartments, on the assumption of a max. speed of 2000 ft. per min., or 1500 ft. average. If this be considered too high for man-cages, with a max. of 1500 ft. and using 6 compartments, 3000 men could be handled in 1½ hr. After resuming hoisting of ore, man-cages would be used for lowering material, etc. A load of 48 men would weigh 6500 lb., and the 4-deck cage 5500 lb., making a gross load of 6 t. As compared with the regular hoisting load of 6½ t. (8000 lb. ore, and 5000 lb. skip), there would be a margin of safety for men on these cages, and the safety of men travelling in ore compartments would be further assured by reduction in rope stress due to lower hoisting speed. On the other hand, if men are raised and lowered in 2 compartments only, these would be kept in nearly constant use, and there would be appreciable elements of danger in possibility of breakage of ore-hoisting ropes in adjoining compartments, where high-speed traffic is simultaneously carried on. The mode outlined above is substantially that in many deep shafts of the Lake Superior district. In the incline of the Calumet and Hecla, separate engines and ropes are used for man-cars; but at the Quincy, in a 5400 ft. incline, average dip about 55°, the regular hoisting rope is used. At change of shifts, the skip is taken off, and a man-car is lowered into place by a crane over the mouth of the shaft, and is coupled to the rope in a few minutes.

Safety of life in winding depends primarily on strength of rope and maintenance of its condition. Breaking is brought about by the joint action of static and dynamic strains. Static strains can be taken as constant for any winding system with established factors of safety. Dynamic strains are brought about by jerky and irregular winding, sudden starting and stopping, and imperfect condition of hoisting ways in connection with maximum speeds. The smaller the drums and cylinders of a winding engine, and the more uniform the speed, other things being equal, the less liability of excessive shocks from imperfect handling. Where an average speed of 2500 ft. per min. is required, the max. speed must be much higher in the system demanding greatest time for acceleration and retardation.

Ropes for men's winders should be same size as those for rock, and new ropes should always be used first on men's engine.

Traversing-hoist.—To overcome the pernicious effect of great radius, Morgans, at Dolcoath mine, uses a drum of small diam. and a small engine; and to overcome angling, he causes the drum to travel at right angles to direction of rope at a distance equal to diam. of rope at each rev. of drum; the rope is thus kept constantly in or about a straight line with its poppet-sheave. It is built to work at 140 lb.: during sinking, it worked simple, at 100 lb.; subsequently, a low-pressure cylinder was put in, and it worked compound. Ropes do not exceed 5¼ in. circ., or 28-29 lb. per fath. Cages, rope, and loaded tubs are estimated to weigh 6¾ t., of which the useful load will be 3 t.; and allowing 2 min. for each wind from 3000 ft., and ½ min. for changing tubs, the ore raised per hr. will be 72 t. A speed of 3000 ft. in 2 min. is, however, not regarded as the limit. It consumes about 20 lb. steam per i.h.p.-hr.

Compressed-air Hoists.—Compressed air as a motive power is not economical, even when re-heated, which, at a depth of 3000 ft. and a temperature of 82° F. is undesirable; and a power hoist situated underground (for stage winding) is liable to neglect, owing to inconvenience of overhauling, which brings up maintenance cost. Still, for large underground winders at heads of main inclines, re-heated air, where possible of application, or Cummings's high-pressure return-pipe system, promises to be a competitor with electricity. Two such winding plants are already in course of construction on the Rand. Advantage in first cost would generally be with this system as against electricity. Efficiency of compressed air on the return-pipe system in a well-designed and maintained plant, should be higher than for electric winding, and cost of operation should be less than purchased electricity, if the load factor is small, as would be the case with single shift operation.

Two-lift Hoist.—Pettit suggests a winding-engine on surface with a rope leading down the shaft (Fig. 197) for very deep hoisting. By this means, a small load is brought up in two lifts, which are approximately equal in time occupied and in power exerted to a large load hauled the whole distance; and the engine making the first lift is not called upon to store the whole rope on its drums: in fact, half the rope is never on the drums, and the rope required is not so cumbersome. An engine on surface is cheaper than an engine underground, both in maintenance and running cost, and considerable expense is saved as against stage winding, in that it is unnecessary to cut a large and costly chamber. The first lift centre line is at *a*, second at *b*, and delivery from first to second lift at *c*.

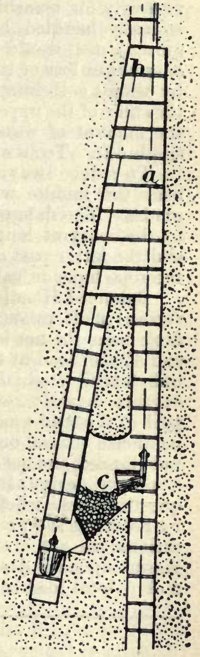


FIG. 197.
TWO-LIFT HOIST.

Stage Hoisting.—It is considered by many Rand engineers as probable that, in the third row of deep levels, stage hoisting will have to be resorted to, as the size and weight of rope on drum involve a tremendous mass being set in motion each time the engine is started. For a shaft 6000 ft. deep, and a load of 6 t. of rock, rope 2 in. diam., weighing about 20 t., besides a heavy drum to support the weight, would be required. Even with tapered ropes and conical drums, the engine would be cumbersome.

Stage hoisting greatly increases the capacity of shaft, since cages or skips of the stages can be run simultaneously, either in the same or opposite directions, equivalent practically to doubling the hoisting speed of a single engine to do the same work. Conversely, a lower hoisting speed may be adopted, easing the work of all parts of the plant, and reducing shocks liable to occur at high speeds. If engines for both stages be placed on surface, and supplied by same boilers, the skips or cages would be run in opposite directions, to equalise steam consumption and reduce size of boiler plant.

A point that presents trouble is shaft-sinking with safety and speed. A suggested solution for a vertical depth of 5500 ft. is to provide temporary cylindrical drums to go to 2000–3000 ft.; then take cylindrical drums off, use the sheave system with tail-rope, and establish a station at, say, 2500 ft.; here start with a second Whiting hoist on surface, and use it to hoist a kibble to this. This is practically stage winding, using Whiting hoists instead of conical drum engines; and, when the shaft is sunk, each engine commands total depth of shaft without a stage.

It has been urged against double stage for permanent plant the necessity for transhipping ore, men, water and materials; but ore is easily handled by self-dumping skips (4 attendants being required), men would simply walk from one cage to the other with only slight loss of time, and water can be handled most easily of all by using a self-dumping skip or bailing tank in the lower stage to fill the sump of the upper—with a given load and engine, nearly double the amount of water can be handled with double stage as with single lift. Tools will doubtless be sharpened underground.

To reduce swaying of long rope at lower stage, it may be fitted near the middle with a guide-piece or cross-head travelling in guides. The danger of serious negative moments will be overcome by the constant length of rope always in the shaft (say 2500 ft.); and the latter part of the sinking can, if desired, be done by making the rope taper in parallel rope sections from 2500 ft. to bottom.

Balanced Hoisting.—In a typical hoisting problem of a 2000-ft. vertical shaft raising 2 cars, each holding 2000 lb., on a double-deck cage, the net load of ore is only 32% of total load at bottom, and 47% of load at top, average 39·5%. This means that, with an unbalanced hoist, about 60% of total work done is dead work, i.e. engines must do 2½ times as much work as would be needed if no part of the load was dead weight, or if all dead weight was counterbalanced. In an ordinary balanced hoist where the rope remains unbalanced, the net load will average 66·66% of entire load. Hence, a proper design for hoisting in balance is very important. A question arises whether more important economy cannot be obtained by reducing the proportion between dead load and net load. Conditions are not always such that a balanced hoist can be used. Assuming same depth of shaft, but changing load to one of 5 t. in a skip, total load is as follows:—

——	Weight.	Per cent. of Total at Bottom.	Per cent. of Total at Top.	Average per cent. of Total.
	lb.			
Ore	10,000	57	74	65·5
Skip and rope from sheave to bale..	3,500	20	26	23
2000 ft. 1½ in. plough steel rope ..	4,000	23	..	11·5
Total load at bottom	17,500	100	..	..
Total load at top	13,500	..	100	100

Net load is then 57% of max. load (in place of 32), and 76% of the min. load (instead of 47), averaging 65·5%. This means that with an unbalanced hoist only 33% of total (instead of 60%) is dead work. An unbalanced hoist under this assumption is practically as efficient as a balanced hoist under the first conditions. If this plant is made to work in balance, 83·33% of total work done is consumed by net load. Sometimes it would be feasible to fully balance dead load by attaching a balance-rope to bottoms of skips, extending from one skip to the other around a pulley placed at bottom of shaft. With addition of a balance-rope, the hoist becomes theoretically perfect—the entire effort being utilised in lifting ore, and variable load of engines overcome.

In an ordinary balanced hoist, skips balance each other, and the average weight of ascending rope is equalled by average weight of descending rope, so that, theoretically, the load on engines is due only to net weight of ore. In starting, however, the load is much greater, being the entire weight of one rope added to net weight of ore, and considerably larger engines are required than if perfect balance of rope weight existed at all times. At end of trip, the load is due to net weight of ore diminished by entire weight of one rope, and engines are excessively underloaded.

The disadvantages of a tail-rope in vertical hoisting are the extra weight imposed on the connection between cage and main hoisting rope, and the danger that safety-catches will fail to hold a cage in case main rope should break. Mechanically, it has much to recommend it. Its cost is nominal, because it may be made of discarded main ropes, so long as they are strong enough to carry their own weight. It produces almost perfect equalisation of load, reducing starting-moment to a minimum, and completely eliminating negative-moment at end of hoist. Omitting friction, and the effect of moment of inertia of large and heavy cylindrical drum, no more power is required for 6000 than for 1000 ft. As the mass set in motion increases, the periods of acceleration and retardation in starting and stopping the load must be lengthened. No reason is apparent why the tail-rope could not be satisfactorily employed for 4000 or even 6000 ft. It has no effect on the strains in a loaded rope at time of starting from bottom (as all its weight is then on the other skip), but, as the load advances, this weight is gradually transferred, until it reaches full value at the moment when the skip is emptied. Thus, at the instants when hauling ropes receive their greatest oscillating strains (at beginning and end of trip—which latter, with careless drivers, may be very great), they are automatically ready for them; and though the tail-rope appears to bring serious stresses to bear on that part of the hoisting rope which most frequently breaks—near the skip—the most serious strains, oscillating, should only occur in greatest intensity when the rope is best prepared to withstand them.

Sheave-Hoisting.—The best-known type of sheave-hoist is the

Whiting, though originally it had no tail-rope. It has a friction drive, with two grooved pulleys, both driven, and the rope passed several times round each. The first sheave is driven direct by main connecting-rods; the second receives motion through a pair of parallel rods, as used on locomotives. The second shaft is slightly inclined, to prevent the outgoing rope fouling the second Walker pulley; and the driving-rods are made with a simple compensating device to avoid any risk of binding. A tightener is used for adjusting length of rope to wind from different levels, being a tension-pulley on a carriage and a strong winch for moving the carriage on its track, similar to the arrangement of endless-rope haulage. Sheave-winders have been much used for depths between 1500 and 5000 ft., and one of the deepest (Calumet and Hecla) uses a tail-rope most successfully. Yet it is said that this type of Whiting is not satisfactory on the Rand, in some opinions, owing to slipping of the rope before the tail-rope is put on, and undue strain set up between the two drums, besides the disadvantage (in case of breakdown) of inability to uncouple one rope and do repairs in the other skip-road. On the other hand, it is urged that, by placing one engine on surface and one underground, with drums only 16 ft. diam., a depth of 11,000 ft. can be reached if required—which would be entirely impracticable, even with taper-ropes, by stage-winding with two conical-drum engines from surface. At De Beers, it is found that when the engine is started too quickly, there is a slip in the rings, which does no harm, and eases the strain on the rope. It is possible that, with greater moment of inertia (fly-wheel effect), it would be impossible to get the jerk which causes this slip, an advantage in heavy drums being steady running.

It may be urged that the Whiting hoist is not as advantageous in sinking as a conical-drum hoist—that it is less flexible, always requiring the use of two skips, and, if one is out of order, work cannot be continued. This is true of the conical system for moderate depths; but for great depths, only one skip can be used, and it would always have to run unbalanced. There would be no provision in one fixed drum for varying length of rope as depth increases; consequently, it would take twice as long running time of engine to clear the bottom of shaft as with a balanced load, 2 skips and the Whiting engine. This also applies to large quantities of water. Again, the Whiting system, by its take-up arrangements (composed of a sheave on a carriage running on a track, and adjusted by a small steam winch), can use balanced skips with ease and advantage for sinking to 5500 ft., and an accident necessitates no greater stoppage than with large conical drums.

On the Rand, there is another consideration which makes it important to always hoist in balance, and to use a tail-rope to make the balance perfect. This is, that each of the large groups of mines operates a great number of different shafts at varying depths. If each one of these shafts is to be equipped with winding engines using conical drums or their equivalent, then it will be necessary to employ almost as many sizes of winding engine as the company owns shafts. By running in perfect balance, using a tail-rope, and fixing beforehand on a uniform net load, it becomes possible to design a hoisting engine which may be regarded as standard, and which is equally well adapted to a comparatively shallow shaft, and to one of the deepest, within the limit of strength of ropes. The Whiting system is especially well adapted to be used for such standard winders, because its drums are exactly the same for 5000 as for 2000 ft. For a shallower mine, it would be possible to build drums for a smaller rope, but there is no objection to using a smaller rope on a drum designed for a larger. Rand Mines Ltd. have adopted as a standard an engine to lift 4 t. of rock from a max. depth of 5500 ft., requiring a 1½-in. diam. plough-steel rope on drums 12 ft. diam.; and such a rope over 12-ft. drums is strained considerably less per sq. in. of net section than is a 1·6-in. taper rope proposed for a drum 10 ft. diam. at the small end.

The conditions fulfilled by a Whiting hoist at the Geldenhuis Deep are raising a load of 5200 lb. up an incline shaft (446 ft.) at an angle of 35°, round a bend whose radius is 75 ft. from incline to vertical shaft, thence 800 ft. on the vertical to the automatic tippler on surface. The fundamental requirements for hoisting—minimum initial expenditure, and minimum running costs—are said to be fulfilled.

The Whiting hoist at the Robinson Central Deep embraces improvements which 5 years' experience have taught—16 ft. diam. sheaves; 5 Walker rings per sheave instead of 3; Wheeler surface condenser plant of 3000 sq. ft. cooling surface; post and crank-disc brakes operated by steam and foot-levers; indicator of dial type, constructed with slip motion, to permit engine to automatically correct its reading once for each run if necessary; a safety brake applying pressure directly to the rope between sheaves, consisting of two strong V clamps, having brass faces grooved to the rope, and operated through toggle joints by a foot-lever and screw-down wheel. The engineer's platform is so placed that he can see over the top of drums, and on this platform all hand-levers, foot steps, and gauges are mounted. Main brakes are applied by dead weights, and released by direct-acting steam-engine equipped with oil-cataract cylinder. Reversing gear is of the Allan type, with straight link, and is operated by a direct-connected steam-engine with oil cataract and floating levers. Cut-off is effected entirely by Corliss gear, under control of a standard centrifugal governor. The engine is to raise 8000 lb. of ore from a vertical depth of not more than 5000 ft.; skips to weigh 4500 lb. each; steam pressure at throttle, 140 lb. per sq. in. A tail-rope is used to fully counterbalance working rope. Every engine of this type is of same cylinder dimensions, minimising spares carried in stock, especially when one group of mines has 9 Whitings working, and another 20–30 on order.

The Koepe system much resembles the Whiting. It consists in substituting for the ordinary cylindrical drum a grooved pulley, round which the rope makes rather more than half a turn, and thence passes over the pit-head pulleys, and down two divisions of the shaft. A balance-rope is beneath the cage. With a rope passing only one-half turn round the driving-pulley, the co-efficient of adhesion between steel rope and wood rim is in practice 30 %. This is not sufficient, and slipping takes place, especially just after the rope has been greased. Moreover, when the cages reach landing-places, and rest on stops (if any are used), weight is removed from the rope, and sufficient adhesive power may not exist on the rim of the motive-pulley to enable loads to be restarted. This can be guarded against by continuing the rope past the cages by means of cross-heads above and below each cage, connected together by side pieces passing outside; the bridle chains are hung from the top cross-head, and, when the cage rests on stops, the weight of winding- and tail-ropes still remains on the motive-pulley. A disadvantage of the system as compared with the Whiting is that in practice a larger diam. drum is required for same depths and loads, the driving power being frictional, and dependent on arc of contact. Another objection is lack of flexibility; it is impossible without a tail-rope, and is impracticable for sinking. It is also difficult to make adjustment for stretch of rope.

Electric Hoisting.—The advantages of electricity in hoisting are being more widely acknowledged every day, and, despite some failures through ignorance and ill-adaptation, the electric hoist is growing in favour. The chief problem for the electrician lies in finding the most suitable method of control. The motor itself is a well-tried machine and presents no difficulty, but the controlling apparatus must be designed to meet the arduous conditions of frequent starting and stopping, must provide great accuracy, and must permit steady running at slow speeds for purposes of repairs, inspections, etc.

There are two distinct types of electric hoist—one operating by alternating multiple-phase current, the other by direct current. (See pp. 123–41.)

Alternating Current.—Some prominent examples of this system are :—

The Preussen Shaft No. 2, at Harpen, is 2300 ft. deep; max. hoisting speed, 3150 ft. per min.; net load of coal, 4840 lb. Koepe system of hoisting (see above), the driving sheave being 19·7 ft. diam., driven by a triphase asynchronous motor of 1400-h.p. Capacity, 800 t. coal per 8 hr.

At the Arnim coal mines, Zwickau, the hoist has a triphase motor of about 80-h.p., taking current at 500 volts; it is placed directly above the shaft, there being no sheave; an advantage of this is lessened wear of rope, which is bent but once.

A Dortmund colliery, hauling from a depth of 2296 ft., 100 t. coal per hr., uses a 2000-volt triphase induction motor, the armature of which is keyed directly on the pulley-shaft. Starting and speed regulation are effected by liquid resistance in the rotor circuit: each of the 3 phases of the rotor circuit is led to electrode plates, insulated, and suspended in a tank containing a solution of soda. The resistance in the circuit decreases in proportion as the solution is made to rise in the tank, causing the motor to run at higher speed; full speed is reached when the solution is at highest level.

The electric hoist at the Village Main Reef will be placed at a vertical depth of 1100 ft., and will haul on a 1700 ft. incline. The generating plant will be of 900, and the hoist of 500-h.p. Resistance for starting and controlling the motor is effected by water constantly kept in circulation by a centrifugal pump: by an ingenious arrangement of valves, the water is raised or lowered, thus cutting in or throwing out resistance in the motor armature. The jerks so common in metal resistance methods are thereby obviated.

The Comstock mines have adopted a decided departure from usual practice in deep-mine hoisting plants, embodying a balanced or tail-rope system. This is done to reduce cost of operation, and size of motor to lowest compatible with duty required—hoisting 500 t. daily from 2500 ft. by double-deck cages carrying 3600 lb. of rock. The hoist consists essentially of a main driving-drum and an idler, around which is wrapped a 1⅛-in. plough-steel wire rope. The rope passes down one compartment, round a movable tail-sheave and up the other (see p. 487). The main driving-drum is geared to a 200-h.p. variable speed, triphase induction motor, which operates at a maximum speed of 580 rev., moving the cages at 1250 ft. per min. The speed of the motor is readily controlled by variable resistances inserted in the secondary winding, but external to the motor itself, the controller resembling that of a street-car, while the primary circuit is controlled by an oil-break switch. The hoist is equipped with heavy post-brakes, hydraulically operated, and is handled with remarkable ease. Tests show a net efficiency of about 75%, counting all electrical and frictional losses. Power is developed on the Truckee river, 33 miles away; at the station the potential is raised from 400 to 24,000 volts, at which pressure it is transmitted over a double circuit of No. 4 hard-drawn copper wire. At a sub-station, in Virginia City, the potential is reduced to 2300 volts, and in this form is distributed to the various mining companies. In the case of each hoist but one, it is again reduced to about 450 volts. As the power is purchased upon a continuous rate basis, fixed by a peak load of 2 min. duration, it has been the endeavour of the mining companies to secure a hoist that will operate at highest possible efficiency, and at same time affect regulation of the general system to as slight a degree as is consistent with good service. To meet the condition of high efficiency, the motor must operate continuously at or near its full-load capacity, and the nearest possible approach to continuous full-load condition is secured by a balanced system, where the load is reduced to weight of rock alone. Cages are started slowly, the dip in line voltage being about 7% at starting. By running on the second notch of controller, $\frac{1}{3}$ max. speed may be maintained for the full length of the shaft. The hoist itself consists essentially of main driving-drum and an idler around which the rope is wound 4 times to secure necessary friction. At one of the mines, 2 hoists side by side are driven by a single motor, one hoist serving the vertical and one the incline shaft.

Comstock Electric Balanced Hoist.

Kind of Hoist.	Double Hoist, 4 Compartments.		Single Hoist, 2 Compartments.	
	Yellow Jacket.	Belcher.	Union Shaft Co.	Con. Cal. and Va.
Daily Capacity from bottom ..	500 tons.	500 tons.	500 tons.	600 tons.
Make of motor ..	Gen. Elect.	Gen. Elect.	Gen. Elect.	Westinghouse.
Type of motor ..	7200 alternations, 440 volts.	7200 alternations, 440 volts.	7200 alternations, 440 volts.	7200 alternations, 2240 volts.
Size of motor ..	75 h.p.	75 h.p.	100 h.p.	200 h.p.
Speed of motor ..	450 rev.	450 rev.	450 rev.	550 rev.
Weight of rock ..	3200 lb.	3200 lb.	3200 lb.	3760 lb.
Weight of double deck cage	2200 „	2200 „	2100 „	2951 „
Weight of 2 cars (max.)	1700 „	1700 „	1700 „	1730 „
Weight of rope in each shaft ..	1896 „	1390 „	2528 „	5000 „
Weight of total load raised ..	8996 lb.	8490 lb.	9528 lb.	13,441 lb.
Weight of unbalanced load ..	3200 „	3200 „	3200 „	3760 „
Diameter steel rope used ..	1 in.	1 in.	1 in.	$1\frac{1}{8}$ in.
Weight of rope per foot	1·58 lb.	1·58 lb.	1·58 lb.	2 lb.
Distance load to be hoisted.. ..	1175 ft.	850 ft.	1550 ft.	2500 ft.
Max. rope speed per min.	600 „	600 „	750 „	1250 „

Continuous Current.—There are some 30 electric hoists working in Germany, mostly at collieries, and on continuous current system.

The most striking is that at the Zollern No. 2 shaft, where a motor absorbing only 300-h.p. constant affords power varying from nil to 1500-h.p. The shaft depth is 1640 ft.; max. hoisting speed 3936 ft. per min.; net load of coal, 9240 lb. Two flat-rope reels are run on one shaft, on each end of which is a 1400-h.p. motor. The capacity is 1300 t. of coal per 8 hr. This 2800-h.p. plant is probably the largest yet erected.

A direct-current motor using the Leonard connection is very satisfactory, as regards safety of working, ease of control, and simplicity of attendance. Speed of winding motor is changed by altering the field intensity of the steering dynamo, with the aid of a simple shunt-regulated resistance. Direction of travel is reversed by aid of a small switch in the field coil. Winding speed is independent, within wide limits, of the magnitude of the useful load, being the same in lifting and lowering load; in fact, it is almost exclusively dependent upon the position of the controlling lever that actuates the resistance, and thus it is possible to connect with the indicator a simple and reliable safety apparatus, which, acting on the controlling lever, prevents both too rapid starting and too slow stopping. Any speed, down to practically nothing, may be provided for, with all loads, without actuating the mechanical brake; and during the slackening stage, any surplus energy, deducting generator losses, can be recovered.

Plants designed on the Ilgner system, utilising the effect of fly-wheels, are remarkable for uniformity of load on converter motor, and extensive alteration in current intensity of winding motor. The consumption of energy for each run, even when the number of runs per hr. is decreased considerably, is relatively low. To further decrease this in small outputs, the fly-wheel in some recent plants is connected to the motor-generator by a clutch. Uniformity of load effects considerable decrease in fuel consumption. Where two or more plants are installed side by side, there may be a common fly-wheel, or separated fly-wheel converters coupled together electrically or mechanically, thus compensating for any load difference in converter plant without requiring increase in weight of fly-wheels.

In the Ilgner plant, two motors are directly coupled to winding-drum, without use of intermediate gearing. A battery of electric accumulators furnishes means of storing power during intervals of no load or light load, and assists the generating station during times of heaviest demand. During retardation at end of trip, the kinetic energy of moving parts is paid back into the accumulators instead of being wasted in brakes. The battery also furnishes a series of voltages for effecting speed-control. It is divided into four groups, and the two motors may be connected in series or parallel. A rheostat is only used to smooth out the steps between successive groupings. Current from generator is taken straight to armature of motor which drives the winding-drum, without passing through any switches, fuses, or controlling gear; no manipulation of heavy main current is required, all control being effected by light switching gear, dealing only with small currents exciting field-magnets. Drums may be gear-driven, but for powers over 100-h.p., it is considered better practice to couple the motor directly to the drum. The slow speed thus necessitated makes the motor larger and more expensive in the first place; but the disadvantages of gear for heavy powers, with the noise and wear-and-tear inseparable from its use, more than set off the extra cost. Working brakes are only used to hold the load; there is no absorption of power, any excess of energy in the moving system at end of journey being taken up electrically.

Emergency brakes are entirely distinct, and are applied by a counterbalance weight, normally supported by pressure of compressed air on a piston; they are applied, therefore, if through any accident the air pressure fails; also, by a special trigger on the indicator, should the cage overrun; and finally, in case of emergency, they can be applied by hand. In any case, the falling weight opens a special emergency switch, which entirely disconnects the winding-motor. A total efficiency of 60 % is counted on under daily working conditions; i.e., of the total electrical energy supplied to the terminals of the motor generator, 60 % appears as net work in raising useful load. Assuming only 50 %, there will be, with steam-actuated generators, a consumption of 40 lb. steam per effective h.p.-hr. on the useful load—a great deal less than the best (and less than half what may be regarded as an average) working result with steam-winding.

Some further examples of direct-current electric hoists are:—

(*a*) 100 t. coal per hr., from 1500–2500 ft.; useful load 5 t.; max. speed, 2700 ft. per min.; 2 fly-wheels, each 43 t.

(*b*) 175 t. coal per hr., from 2500–3000 ft.; useful load, 6 t.; max. speed, 3500 ft. per min.; 2 fly-wheels, each 43 t.

(*c*) 125 t. coal per hr., from 1600 ft.; useful load, $3\frac{1}{2}$ t.; max. speed, 2400 ft. per min.

In the opinion of competent authorities, it is probable that before the deeper Rand shafts have reached a depth when second winding engines (stage winding) will be required, a suitable and economical electric hoist, if not for alternating current, then for direct current fed from high-tension alternating-current cable through transformers and rotary converters, or even by direct-current cable carried underground at 500 volts, will be on the market. It is questionable if the additional cost of large cable required for direct current and lower voltage will be greater than the smaller cable and accessory transformers and converters. It has been decided, at the Cinderella Deep, to sink two inclines away from the main hauling shaft, and to use 2 800-h.p. direct-current winding engines to pull rock in these inclines and deliver on a mechanical haulage or conveyor belt to the main ore-bins at bottom of the vertical shaft.

Hoisting Costs.—It is so seldom that anything like segregation of accounts is attempted on mines, especially in the matter of fuel consumed or power applied, that reliable figures relating to hoisting costs are most difficult to come by.

Steam hoisting on English collieries with a daily output of 450–4000 t., hauling from 1000–2500 ft., has an actual cost of $\frac{1}{3}$–$2\frac{1}{4}d.$ per ton. With improved engines, using steam at 135–150 lb., raising 1000–2000 t. daily from 1500–2100 ft., it is reckoned to be done at an $\frac{1}{2}$–$\frac{3}{4}d.$ Electric hoisting, balanced, steam-actuated, under some conditions, is estimated to cost $1\frac{1}{4}$–$1\frac{1}{2}d.$ All these figures include interest and depreciation.

Alaska and the Lake copper mines reckon on about $5\frac{1}{2}d.$

Rand and Kimberley mines show 7–11*d.*, averaging about $8\frac{1}{2}d.$; but the new City and Suburban claims to cover costs with 2·2*d.*

Cripple Creek admits 11*d.*

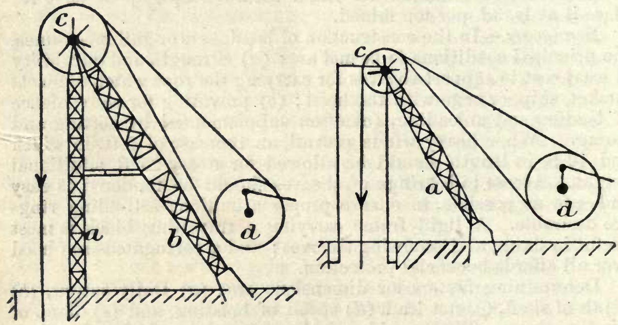
Conservative estimates on Rand future deeps, up to 6500 ft., place it at 1*s.* 3*d.* per ton mined.

Headgears.—In the construction of headgears or gallows-frames, the principal conditions to be met are: (*a*) Strength and durability at least cost to support sheaves for carrying the rope which connects bucket, skip, or cage with the hoist; (*b*) providing for convenience of loading and unloading, (*c*) often supplemented by sorting and storage. When heavy winds prevail, an increase of 5 ft. in width and 10 ft. in length should be allowed for every 25 ft. additional height. Access to bearings of sheaves should be rendered as easy and safe as possible, to ensure proper attention: self-oiling rings are desirable. A light frame carrying a travelling block is most useful in fixing and renewing sheaves; and a corrugated-iron hood over all affords beneficial protection.

Determining factors for dimensions are: (*a*) Daily output, (*b*) depth of shaft, (*c*) net load, (*d*) speed of hoisting, and (*e*) diam. of winding drum. With rapid hoisting from deep shafts, abundant headroom should be given for possible errors in overwinding, certainly not much less than a distance equal to one turn of the rope on the drum of a large first-motion engine; 20 ft. is ample in small plants.

Resistance has to be provided against the united pull of the load in the shaft and the engine raising it, and against tendencies to twist, spring and oscillate under varying influences of the load in motion and on arrest. The pull toward the engine is met by bracing-timbers set on a line more or less equally dividing the angle between the rope-lead from sheave to engine and the vertical. English practice mostly follows the principle (Fig. 198) of placing one support *a* parallel with the vertical strain, and another *b* parallel with the diagonal strain, *c* being the poppet sheave and *d* the hoist. Thus one post is virtually on the wall of the shaft, and the other beside or below the hoist.

It is submitted that both are faulty in adding unnecessary loads to foundations which already have enough to support, and in needlessly cumbering space just where it may be wanted. Often, also, the inclined post requires to be very long in order to give the most appropriate bend for the rope over the sheave, and then it demands stiffening; all this spells consumption of timber. German practice (Fig. 199) relies on one main support *a*, placed on a line which is the resultant of the two forces pulling on the rope. It therefore needs only sufficient vertical support to take up its own weight, this serving at the same time as a guide for the cages; the foundation of this at the shaft does not present difficulties, as it is subjected to very small compressive strains. Height depends firstly upon the so-called "rope angle," i.e. the permissible bend of the rope at the sheave; as 60° is most often chosen, the inclination of the sloping support is 30°.



FIGS. 198, 199.—HEADGEAR PRINCIPLES.

In shallow mining, such as the lead-zinc deposits of Joplin, small headgears only are needed; but they are sometimes relatively high, and, to render them more stable, it is considered necessary to hang a heavy weight to the frame of the landing-floor, by suspending from rods a large box filled with rock. If placed on the opposite side of the shaft from the hoist, it equalises and distributes the weight; but a more usual arrangement is to place the weight directly beneath the hoist, where it serves as an anchor, and aids in resisting the upward pull transmitted through the hoisting rope. (Crane.)

A small 2-post frame (Fig. 200) is well adapted for shafts up to about 300 ft. Simple in construction and serviceable, it is usually mortised, but may be simply toe-nailed. The spread is 26 ft.; height, 20 ft.; bedlogs *a*, posts *b*, and struts *c*, 8 × 8 in.; platform joists *d*, 6 × 4 in.; ties *e*, 8 × 2 in.; cap *f*, 8 × 8 in. It consumes about 1500 ft. (b.m.) of timber, and costs 10*l*.

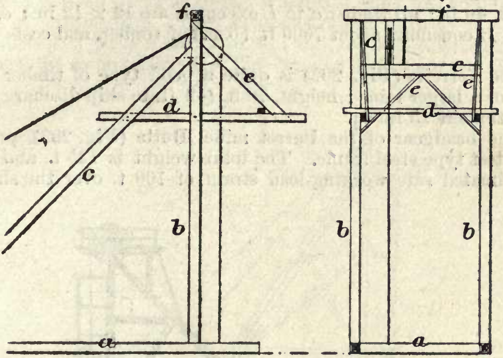


FIG. 200.—TWO-POST HEADGEAR.

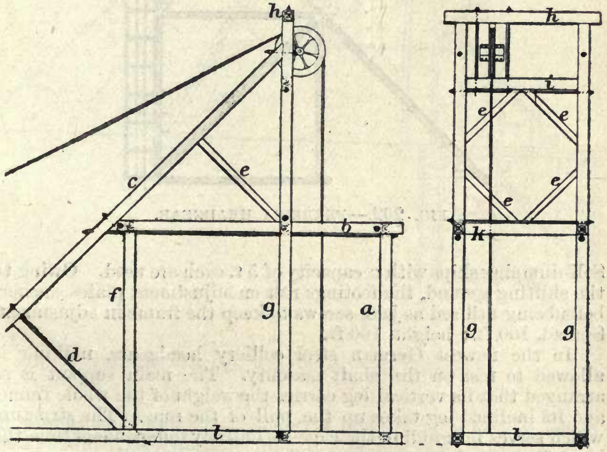


FIG. 201.—SMALL FOUR-POST HEADGEAR.

A larger 2-post frame (Fig. 201) may almost be regarded as 4-post, but the front posts *a* do not go to the top, though they serve to support the cross timber *b* for the bucket-dumping chain. The main batter braces *c*, are well stiffened at *d e*. In some frames of this type, the posts *a f* are set on a side batter. The spread is 47 ft.; height, 40 ft.; all timber *a* to *l*, except *e*, are 12 × 12 in.; *e*, 12 × 6 in. It consumes about 7000 ft. (b.m.) of timber, and costs nearly 100*l*.

The Ferreira (Fig. 202) is quite a good type of timber headgear for a larger mine: height, 70 ft. (42 ft. to skip discharge); all main timbers 15 in. sq.

The headgear of the Parrot mine, Butte (Fig. 203), presents the latest type steel frame. The total weight is 125 t., and it has an estimated safe working-load strain of 100 t. over the sheaves.

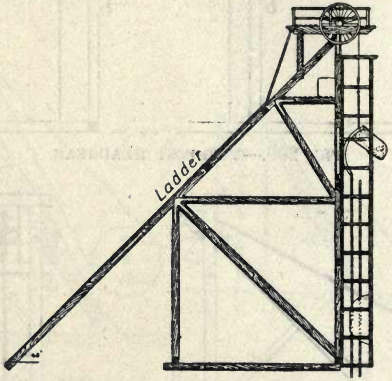


FIG. 202.—FERREIRA HEADGEAR.

Self-dumping skips with a capacity of 5 t. each are used. Owing to the shifting ground, the footings rest on adjustment plates, anchor-bolts being utilised as jack-screws to keep the frame in adjustment. Spread, 100 ft.; height, 100 ft.

In the newest German steel colliery headgears, nothing is allowed to rest on the shaft masonry. The main support is so arranged that its vertical leg carries the weight of the whole frame, and its inclined leg takes up the pull of the rope. The structure which serves for guiding the cages is entirely independent from the head-frame, and does not transmit any pressure on the masonry of the shaft. Broad construction gives the frame great stability, but at the cost of greater weight and greater initial outlay.

An example of headgear and bins for an incline shaft (Fig. 204) is taken from the author's own construction at the Wentworth mine, N.S.W. The bins *m* are in duplicate, each measuring (internal) about 18 ft. deep by 12 × 16 ft.; when full, each will accommodate 225 t. of milling dirt. They are constructed entirely of Colonial "hardwood"; posts *n* are 12 in. sq., and are seated in substantial bedlogs, the same as legs *a*. The wooden ties *o* are 6 in. sq., and

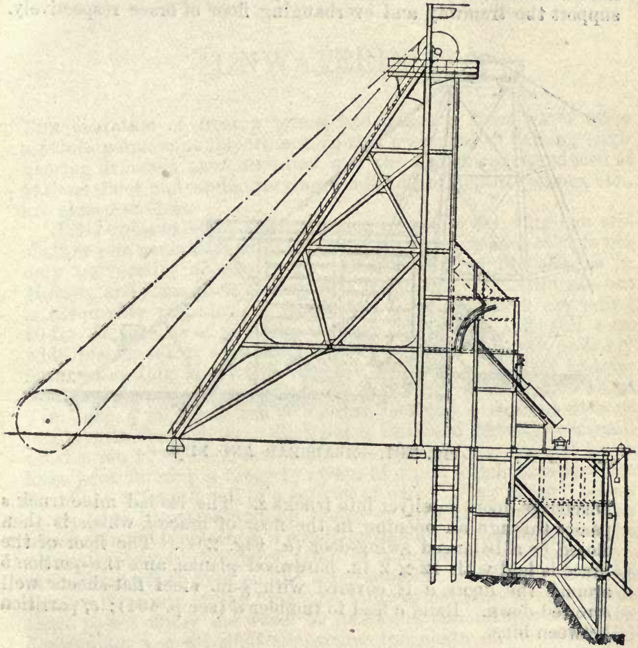


FIG. 203.—STEEL HEADGEAR AND ORE-BINS.

are strengthened by intervening steel (old) railway rails *p*, which were obtained very cheaply from the Government railways. Inside planking is 12 × 2 in., and is lined with $\frac{1}{8}$-in. steel sheets. The sheets in the lower portion were all new; but for the upper parts an economy was effected by using a number of discarded flat-sheets from the mine, some of which were not steel, and many of which did not exceed $\frac{1}{16}$-in. thick. Runners *r*, carrying tram-rails by which truck-carriage *s* reaches brace *l*, are 8 × 4 in. The underpinning of the bins consists of posts and sills *t* 12 in. sq., and struts *u* 10 × 8 in. Experience showed these to be insufficient; after about 2 years' life, the ends of the bins began to bulge, and the floor joists to sag. Supplementary supports *w* of 8 × 8 in. hardwood were added, and served satisfactorily. Struts *x*, 6 in. sq., support the tramway and overhanging floor of brace respectively.

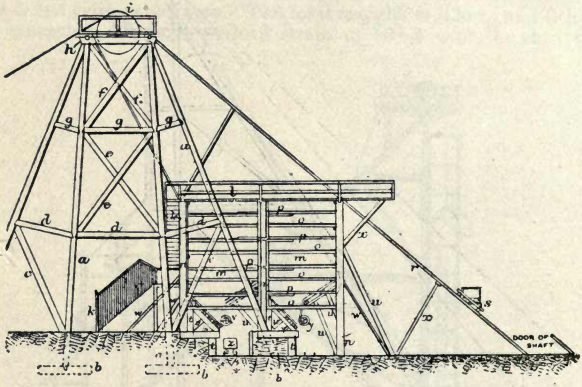


FIG. 204.—HEADGEAR AND BINS.

Discharge-doors *y* deliver into trucks *z*. The loaded mine truck *s* passes through an opening in the floor of brace *l*, which is then closed by a balanced swing-door (*a*, Fig. 204). The floor of the brace is laid with 12 × 2 in. hardwood planks, and the portion *b* around the doors *a* is covered with $\frac{1}{8}$-in. steel flat-sheets well screwed down. Rails *c* lead to tumbler *d* (see p. 464); *e*, partition between bins.

UNWATERING.

The operation of freeing mines and quarries from water often assumes paramount importance, so that no branch of mining engineering demands more attention, and the appliances introduced at various times and under varying conditions of depth, volume, etc., are almost endless.

Pit Draining.—For raising water from alluvial diggings and shallow pits generally, the Chinese chain-pump or *chin-chew* is not only universally adopted where Chinese labour prevails, as in Malaya, and some parts of Australia, Tasmania, and California, but is favourably regarded in Alaska, for lifts not much exceeding 20 ft. It consists of a wooden launder about 6 in. deep and 4 in. wide inside, set at an angle to suit the situation, but probably never exceeding about 30° from the horizontal. In this launder continuously travels an endless chain of small wooden slats about 9 in. deep, ½ in. thick, and of a width to make a neat fit without jamming in the launder. Each slat is traversed midway by a small wooden pin 12 in. long, fastened to the next by a mortice and tenon loose joint, forming a hinge by means of a pin, which usually takes the shape of an old nail or a snippet of wire. The "chain" passes at top and bottom round an armed or spoked spindle. To the upper spindle is attached a windlass handle, if the pump is to be manually driven; but more often it is actuated by a small overshot water-wheel fixed on the same spindle. Each arm or spoke catches the chain at the hinge, *i.e.* midway between the slats; the chain passes under the wheel on its ascent, and returns over it on descending, a small shelf supporting the chain on its downward passage, so that it shall not dip into the launder and catch the ascending slats. The pump delivers a practically solid stream of water immediately beneath the motor wheel, into the same launder or ditch which carries away the water used as motive power. In Fig. 205: *a*, buckets of motor-wheel; *b*, spindle carrying both motor-wheel and spoked wheel; *c*, spokes engaging in hinges of chain; *d*, wooden chain-pump; *e*, hinged joints.

Water-Lifting Wheel.—Another typically Chinese machine belonging to the hydraulic branch of engineering, is the water-wheel shown in Fig. 206. It is even more remarkable than the chain-pump, and is built entirely of rough round logs and spars, bamboo sections, and supple canes, by Chinese, but the same principles might be applied to a light iron construction. Actuated as an undershot water-wheel by a portion of the stream, which is confined between the sides of a short wooden flume, it has attached at regular intervals to its outer periphery a number of "joints" of bamboo about 3 ft. long and 4–5 in. diam., set at such an angle that, at each rev. they fill with water on rising, and discharge their contents into a high-level launder as they approach their max. ascent. The effective height to which water can be elevated by this simple contrivance is just about the theoretical limit, and wheels 40 ft. diam. are not unknown; but a certain loss occurs by splash and leakage. Lubrication of all parts liable to friction is accomplished by tiny trickles of water through bamboo shoots. Such a wheel, built exclusively by ordinary Chinese coolies, and in an incredibly short time, will deliver a good stream of water for ground-sluicing tin-bearing land lying above the stream, and will last long enough to allow of the ground being exhausted, say 2–3 years. A new one can be built more cheaply than an old one can be removed.

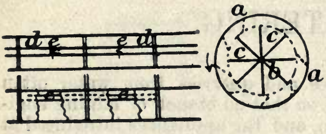


FIG. 205.—CHAIN PUMP.

Siphons.—Siphons may often be used for draining open workings, when it is not necessary to raise the water to a greater height than about 30 ft., and where the necessary fall for delivery can be had.

Raymond describes a siphon over 1000 ft. long and 4 in. diam. The pipe was made of No. 24 galvanised iron, in sections 30 in. long, riveted and soldered together. The water was raised 18 ft., and the outlet end had a fall of 40 ft., so that delivery was 22 ft. lower than inlet. The two ends were fitted with 4-in. brass taps, which were closed when the siphon was to be filled. This was easily performed in about 2 hr. by a 3-in. force-pump, throwing water in at the highest point through a vent-cock, by which also smaller quantities of water might be supplied from time to time to displace air that gradually found its way in through leaks. An air chamber at the bend was projected, but not found necessary, as, on shutting the taps at the ends, it was easy to fill the siphon by means of the pump during men's meal hours.

At Byer Moor colliery, 5 siphons are in use, working over a distance of 3557 yd. The greatest lift of any one is 21 ft.; and this siphon is 1275 ft. long, 4 in. diam., has right-angle turns in it, falls 27 ft. (thus working under 6 ft. head, giving a pressure of 2·59 lb. per sq. in.), delivers 40 gal. a min., and is set by an Evans force-pump. Another drains two sumps, respectively 2766 and 1887 ft. point of delivery; it is 4 in. diam., lifts 14 ft., falls 35 ft., pressure from 9 lb. per sq. in., discharge 35 gal. a min. The longest has 3 branches: the main trunk is 996 ft. long and 8 in. diam.; branches are 2310 and 1227 ft. long respectively, and 4 in. diam., lifting 8 ft.

At Chester South Moor colliery, a siphon 1800 ft. long and 6 in. diam. lifts 26 ft.

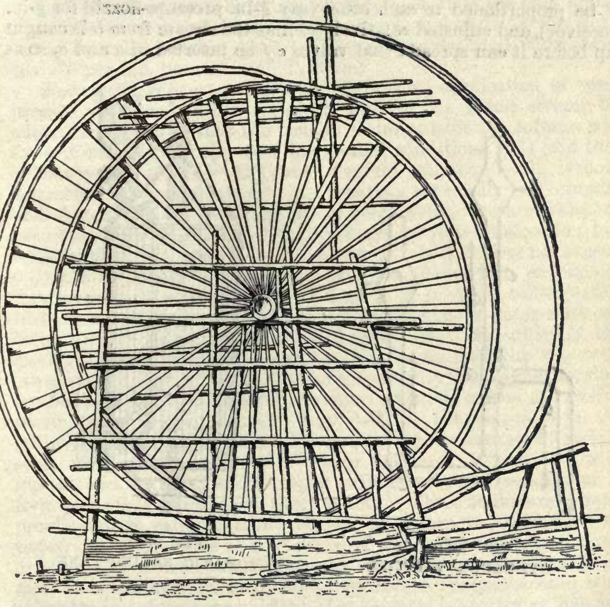


FIG. 206.—WATER-LIFTING WHEEL.

Hydraulic Ejectors.—Where the quantity of water to be raised is small, and no fall is available for a siphon, while a head of water can be obtained, a most useful contrivance is the hydraulic ejector, which depends on the principles of an induced current created by the force and velocity of a falling stream. This simple and effective method is much in vogue on deep gravel mines in California (where a great head of water can be had), and entirely replaces pumps for limited duty, practically at no cost for either operation or repair. The arrangement is shown in Fig. 207: *a*, pipe bringing water from surface; *b*, suction-pipe for drawing water from mine-sump; *c*, discharge-pipe. The suction created in *b* by the rush of water from *a* into *c* induces the water in *b* to flow upwards. Necessary precautions are that the diameter of *c* shall be great enough to accommodate the flow from *a* and *b*, but not so great as to nearly counterbalance the pressure (less the friction) in *a*; that the nozzles inserted in ends of respective pipes united in T *d* be proportioned to each other (say $\frac{3}{8}$-in. pressure-nozzle for $\frac{5}{8}$-in. receiver), and adjusted relatively so that the stream from *b* is caught up before it can spread; that valves *e f* be inserted in *a* and *c*, so as

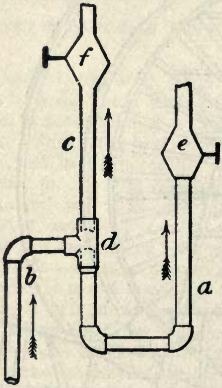


FIG. 207.
HYDRAULIC EJECTOR.

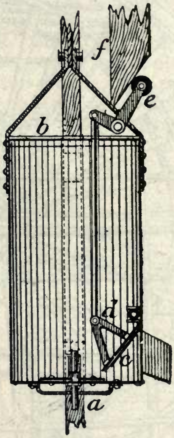


FIG. 208.
AUTOMATIC BALING-TANK.

to shut off water in case of anything going wrong; and that bends be avoided as much as possible, especially after the pressure-water encounters the suction-water. The effective power of the apparatus is about 30% of the pressure-water, and a lift of 200 ft. is easily accomplished. In some cases, efficiency reaches 60%. Cost of operation is virtually nil, and very little attention is needed.

The same principle is carried out in utilising the pressure of water from a high-level to a motor at lower level, a larger volume of water being raised through a lesser height. In the absence of a natural head of water, the necessary pressure may be derived from a steam-engine. Thus the water column of a main pumping-engine may be made to raise its feed from sumps, dips, or winzes 200 ft. or more below, the pumping-engine thus standing above flood level.

At the Comstock mine, Nev., 2 Evans hydraulic elevators raise the water from the 2480-ft. level to the 2250-ft., where they deliver it to the Riedler pump-sump. At present the valves of these elevators are on the 2150-ft. level, while the elevators themselves are anchored at the 2480-ft. One has a 12-in. column, the other a 14-in. Each elevator lifts 2800–3000 gal. water per min. 330 ft., and though its efficiency is only 20% at this head, it is the only appliance capable of withstanding the hot water, and of doing the work in the limited available space. The water power used costs about 16*s.* 8*d.* per h.p. per mo. (C. T. Rice.)

Pneumatic Ejectors.—The "air lift" is an application of compressed air to reduce the sp. gr. of water in the rising column by admixture of air, so that the head of water outside the column will force it out. Under the most favourable conditions, it is said that efficiency may reach 50–60%, but in actual practice 17–20% is more common. Several plants are in operation, especially in Germany and America. For pumping from deep wells, whether water or petroleum (see p. 371), 32–35% efficiency is quite attainable; but when applied for unwatering a "drowned" shaft, it may not exceed 6–12%, for a 200-ft. lift, with air at 90 lb., depending on ratio of "submersion" (vertical depth of bottom of air-pipe below water-surface) to "lift" (vertical height of discharge above water-surface), being greater with increasing submersion. But the utility of the method lies not in efficiency per h.p., but in applicability where no other system is possible, and it has occasionally done signal service.

Baling-Tanks.—These form a very effective means of raising water from shafts, and are employed in connection with the ordinary hoisting apparatus. In vertical shafts, buckets of various sizes and designs are used. Where the shaft is provided with guides, and ore is hoisted in cages, baling-tanks are rectangular in form and made to run upon these guides. These tanks are usually provided with safety-catches, similar in design to those used on cages. A hinge-valve at the bottom of the tank permits automatic discharge of water into launders at surface. A more expeditious method is to empty tanks by adopting the arrangement used with self-dumping skips (see p. 461). Tanks have a capacity of 300–800 gal. Where hoisting is done through incline-shafts, self-dumping skips are used to raise water. At the Utica mine, California, 675 gal. water can be raised in 1½ min. from a depth of 560 ft., through a single compartment of the shaft. This skip is made of sheet iron, 3 ft. square and 8 ft. long; it is filled through a butterfly-valve in the bottom, and, on reaching surface, one wing of the valve is opened by automatic levers actuated by a bumper on the head-gear.

A simply arranged automatic baler is shown in Fig. 208: inlet valve *a* in bottom of tank is operated by impact with the water, weight of tank, when hoisting rope is slackened, causing it to sink; top *b* is open. On reaching surface, outlet-valve *c* in one side near bottom is opened by means of crank-rod *d*, of which top *e* carries a little roller which travels up the inclined side of wooden guide *f*. Approximate sizes, weights and costs of such baling-tanks are:—

200 gal.	700 lb.	20*l*.	600 gal.	1600 lb.	40*l*.
400 „	1100 „	30*l*.	800 „	2200 „	50*l*.

Various more complicated forms of baling-tank have been introduced, some equipped with necessary apparatus to create a vacuum, so that the water may be sucked up a certain distance through a short length of rubber hose. This is useful in shaft-sinking, as the water can be thereby drawn off more completely; but both installation and operation cost much more.

In a two-compartment shaft, the baler can be duplicated, or balanced by a cage.

At the Maypole colliery, Wigan, during shaft-sinking, a baling-tank having a capacity of 1010 gal., and a bottom clack 24 in. diam., for several months coped with an average of 47,000 gal. water per hr., from a depth of over 300 ft., and, in a single experimental hour's work, raised 59,000 gal.

On some large mines, where pumps are ordinarily used for unwatering, baling-tanks up to full capacity of the winding-engine are held in reserve against emergency.

It is the opinion of many engineers that a balanced hoist (see p. 486) can raise water through a straight shaft from great depths at almost as low cost as any pumping plant; and mines having acid water, causing corrosion of pump-valves, chambers and pipes, handle their water in this manner.

The Lackawanna colliery, Pennsylvania, unwaters by 2 buckets, each holding 4100 gal. (20 t.) water, run by a 800-h.p. electric motor, and everything working automatically, one filling while the other discharges. The output is 4100 gal. per min., or nearly 6 million gal. per 24 hr., one man watching the motor, and another the buckets. Other American collieries and iron mines are raising 3½ million gal. per diem in the same way.

Baling-tanks are extensively used in the Rand "deeper deeps" during development, with capacities of 300–1500 gal., from average depths of 1250 ft., at a cost of 10·5*d*. per 1000 gal., or ·169*d*. per 100 ft.-ton, being less than half cost of pumping. (Pettit.)

Pulsometers.—The distinguishing feature of the pulsometer is that it acts by direct steam pressure upon water in a column. The only wearing parts (valves) can be readily and cheaply renewed. No expense is entailed for skilled labour. It will run for long periods unseen, and requires no oil, tallow, packing, or foundation, working as well suspended from a chain as when fixed. It can be operated whilst being lowered, which is a great advantage in sinking; and it occupies less room than any other pump of equal capacity. Having no exhaust steam, it can be used in confined spaces without heating them up. It is noiseless in operation, and will pump very dirty water to a total height of 100 ft., and under special circumstances much higher. It is best adapted to a suction not exceeding 6–10 ft. for smaller size, and 10–15 ft. for larger, the figures being modified by circumstances. Steam pressure at pump for lifts of 20–40 ft. should not be less than 20–30 lb. per sq. in.; for 40–80 ft., 30–50 lb.; and for 100 ft., 70–80 lb. For lifts above 90–100 ft., the discharge of one may be taken into the suction of another above; or a small tank may be placed midway, the bottom pump discharging into it and the higher one drawing from it. It is of course very wasteful of steam; but no appliance is more useful in an emergency, as it can be readily brought into play, and will deal with grit and even stones that no pump would tolerate.

Pulsometers.

No.	Height of Pulsometer.	Space occupied.	Steam Supply Pipe.	Suction Pipe.	Discharge Pipe.	Gallons per Hour.
	in.	in.	in.	in.	in.	
1	18	10 × 10	$\frac{1}{4}$	$1\frac{1}{2}$	1	900
2	22	15 × 13	$\frac{1}{4}$	2	$1\frac{1}{2}$	2,000
3	28	23 × 15	$\frac{1}{2}$	3	2	3,800
4	32	24 × 20	$\frac{1}{2}$	$3\frac{1}{2}$	$2\frac{1}{2}$	6,000
5	39	25 × 25	$\frac{3}{4}$	4	3	10,000
6	42	27 × 27	1	$4\frac{1}{2}$	$3\frac{1}{2}$	13,000
7	48	32 × 26	$1\frac{1}{4}$	5	4	17,000
$7\frac{1}{2}$	51	39 × 28	$1\frac{1}{4}$	5	4	22,000
8	56	39 × 32	$1\frac{1}{4}$	6	5	28,000
9	66	39 × 36	$1\frac{1}{2}$	7	6	40,000
10	79	48 × 42	2	8	7	52,000
$11\frac{1}{2}$	80	56 × 42	2	10	8	80,000

The quantities are given on a total lift of about 20 ft.

A remarkable illustration of the capabilities of the pulsometer was given in sinking at the Maypole colliery, Wigan. The volume of water to be dealt with was 160,000 gal. per hr., or a weight of 12 t. per min., from a depth of 459 ft. The engineer in charge would use pulsometers alone for pumping during sinking, in a series of 100-ft. lifts. One is now raising 30,000 gal. per hr. against 107 ft. head, with 85 lb. steam. When sinking, it is advisable to use a few feet of strong flexible rubber hose in the suction, to avoid injury by shot-firing, if pumping cannot be suspended.

Cornish Pumps.—These have many advocates, on account of their reliability and general economy in heavy, deep, and permanent work. Their coal consumption is only about $\frac{1}{16}$ that of steam-pumps. With the best patterns of Cornish pump, such as the Bull and the Davey, the efficiency is about 114 million ft.-lb. per cwt. of coal burned; and with ordinary patterns, not less than 50 and often more than 75 million. The average consumption of coal is 3–4 lb. per effective h.p.-hr. Cornish pumps work well with dirty or clean water, require next to no attention, and are so simple as not to need much skill. But they are costly to instal, and they occupy enormous space in the shaft. Maximum throw at a single lift is 300 ft., and it is not advisable to exceed 250 ft. if it can be arranged; thus, at every 250 ft., a chamber must be cut to accommodate another pump and its cistern, though the motion is communicated to all simultaneously by the same rods. The space occupied by a 12-in. Cornish pump is such that the shaft compartment in which it is placed (usually comprising also a ladder-way) must measure about 5 × 6½ ft. On the other hand, it must be admitted that, in a very great number of instances, the influx of water in a mine is chiefly confined to quite shallow depths, and that, by simple measures of damming and collecting, the bulk of the water can be gathered say on the 200-ft. level. In such cases, underground pumps of much less capacity, driven by steam, air, or electricity, may be used to deal with the lower supplies, and to feed the Cornish-pump cistern; and the pump-way can from that level be much contracted. The weight of the Cornish-pump column and rods is very great, and demands special timbering. The pump itself can operate even when flooded.

An unusual arrangement in the deep leads of Loddon, Victoria, consists of Cornish pumps in a double set of 20 in. diam. plungers, with a stroke of 8 ft. max. and 4 ft. min., and a capacity of 108,000 gal. per hr. to 310 ft., driven by an electric motor, power being generated by steam at some miles distant and transmitted electrically. The column is 22-in. lap-welded steel pipe. The top lift is 135 ft.; the bottom, 175 ft. Counter-shafting, fitted with friction-clutches, permits necessary modification of speed and control of starting. The two sets of pumps are connected, and counterpoise each other, no balance-boxes being used.

The first row of Rand deep levels are, as a rule, equipped with Cornish pumps of ordinary type and arrangement; in the inclines, the rods are either carried down the slope and used in compression, or a steel rope is fixed to the end of the pump-spear, and, passing under a sheave, has a sufficiently heavy weight at the end (supported on a carrier) to work the pump plunger.

A drawback to the Cornish pump lies in the fact that all the plungers must operate at the same speed, and with practically the same length of stroke, making it imperative to draw back some water to make up deficiencies at some of the pumps. The plunger, however, so long as it is raising, is doing work against full pressure, however much of its water is being drawn back.

Yet several of the largest mine pumping plants, newly installed, are on this principle—three, at least, dealing with 10 million gal. a day each from over 1000 ft., situated in Victoria, Tasmania and Japan, all of the Hathorn-Davey type. The plant at the Tasmania gold-mine is divided into 3 units, each consisting of a steam engine (150 lb.) on surface, actuating 4 pairs of plunger pumps in the shaft, raising the water 2000 ft. in 4 stages each of 500 ft.

Differential gear controls admission and release of steam throughout the stroke, and is not only a sensitive governor (regulating point of cut-off to suit any variation in steam pressure), but also a great safeguard in case of loss of load. A "pausing" gear is also provided, consisting of a small subsidiary steam cylinder, which actuates the admission valve of the differential-gear engine, and by which the main engine can be made to pause for any desired time, enabling regulation of the number of strokes per min. to suit the make of water in the mine. Further, this pause enables the pump-valves to settle on their seats by their own weight, instead of being forced down by the return stroke of the pump. By having 8 suction and 8 delivery valve-boxes to each plunger, it is possible to get a full water-way and still use small valves and small valve-boxes, no part being too heavy for one man without tackle. The rising main is of steel, 16 in. int. diam., and has welded flange joints. The weight of spear-rods and plates being in excess of that of the water to be forced to the surface does not prejudice efficiency, one set rising while the other is falling; balance-beams are provided for taking up surplus weight.

On the excellent Cornish pumps made by Robey, Lincoln, all bolts are pivoted, and it is impossible to drop bolts or nuts during changing clacks, and so on.

The Cornish pump at the Comstock mine is driven by water power, under 288 ft. kinetic head, the first speed reduction being by belt and the second by gear. Principal dimensions are: Stroke, 6 ft.; max. strokes per min., 8¾; 1 plunger, 14-in. diam., under 128 ft. head, 400-ft. level; 1 plunger, 12-in. diam., under 250 ft. head, 700-ft. level; 1 plunger, 12-in. diam., under 287 ft. head, 1000-ft. level; 1 plunger, 12-in. diam., under 230 ft. head, 1250-ft. level. The rod is of Oregon fir, 12 in. sq., and is supported by rollers in the customary fashion. There is an angle-bob at the 400-ft. level, where the shaft changes dip, an ordinary counter-balanced operating-bob on surface, and a balance-bob at 900 ft.

Efficiency, based on 181 h.p. in motive water and 75% of full displacement (65 h.p.), shows about 51%, including drawback (automatic, by floats and siphons) to keep pumps solid.

Electricity may similarly be applied to the driving of a Cornish pump. Perhaps the first in the world was installed by the Author in 1903 at the Raub mine, Malaya (Fig. 209). The type of motor available being one-speed only, large fluctuations of load had to be provided for by a series of holes in the crank-disc for a movable pin, giving varying radius of crank; small differences were accommodated by giving a stroke somewhat in excess of needs, and returning sufficient water to keep the pump solid. The installation saved nearly 100*l.* per mo. for firewood, the electric power being water-generated. An obvious modification would consist essentially of a variable-speed motor direct-coupled to the toothed gear for reducing speed. The normal speed of the motor might give 3, 5, or 7 rev. per min. at crank-disc, according as one winding or another were put into circuit by a two-way switch; while additional speeds of 4 and 6·66 rev. per min. could be obtained by altering the regulating resistance in the starter. The crank-disc would have 4 holes for a movable pin, giving a crank radius of 3 ft., 3 ft. 6 in., 4 ft., and 4 ft. 6 in., respectively.

Steam Pumps.—The varieties of pump operated directly by steam are legion, every manufacturer having some particular feature for which superiority is claimed. All that can be done here is to briefly indicate the main characteristics of the several classes into which they may be divided.

Ordinary steam-pumps have an efficiency of about 4 million ft.-lb. per cwt. of coal burned, rising to 10–15 million when furnished with condensers, etc.; their consumption of coal per effective h.p.-hr. is rarely less than 10–12 lb. and often more than 15.

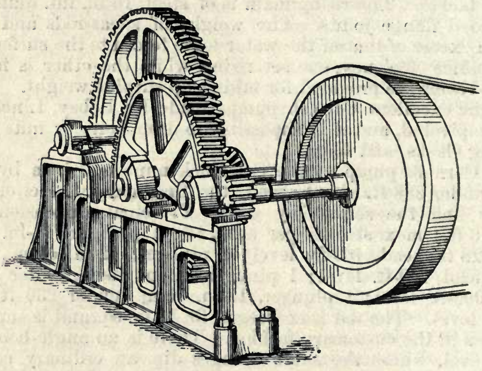


FIG. 209.—ELECTRICALLY-DRIVEN CORNISH PUMP.

Underground pumps driven by steam carried down the shaft are very wasteful of fuel, and possess the great inconvenience of heating the air of the mine. For heavy duty, where fuel is costly, they are out of the question. Their liability to break down is much greater and their life does not average much more than $\frac{1}{10}$ that of Cornish pumps; but they are much cheaper to instal, even when duplicated (as they should be, to provide for contingencies), and they work much more rapidly. At the same time, provision can generally be made at moderate cost for storing 3–4 days' ordinary flow of water. In America, it is usual to have sumps holding 15,000–20,000 gal., and steam-pumps are regarded very favourably. The same pumps may be operated by compressed air. Steam may be economised by carrying the exhaust into the suction-pipe, the best forms of suction condenser producing a vacuum of 10 lb. per sq. in. on the steam-piston, and saving 20–50 % of the fuel. But if the water possesses any corrosive action, this is very much intensified by the heating. Access to valves should be made easy; and, if the water be gritty, the cylinders should be lined or readily replaceable. There should be a retaining or check-valve at bottom of both suction and delivery pipes; and when suction is deep or long, an air-vessel should be provided. A strainer, placed where it can be attended to, is most desirable.

Non-rotary simple pumps are light, cheap, easily fixed and run, and occupy little room. Their momentary pause at end of stroke is beneficial to valve action. But they require 1–3 in. of clearance between piston and cover, are liable to become centred, and are wasteful of fuel, using 30–50 % more than rotary pumps. They may be of piston or of plunger type. The former are suited only to clean water (no acid or grit), and to lifts not exceeding 500 ft.; the latter are equal to heavy work and up to 1000 ft., and tolerate bad water. Sometimes effects of corrosion are diminished by lining the cylinder or using gun-metal. Working pressures are: up to 300 ft., 30–40 and not exceeding 50 lb. per sq. in.; 600–1000 ft., 70–80 and not exceeding 90 lb. Examples:—

Gal. per hr.	Ft. in one lift.	Steam lb.	Dimensions.
2,500	480	30	15 × 4 in.
3,500	400	40	21 × 6 in.
8,000	1040	..	26 × 6½ in. × 6 ft.
10,000	120	40	12 × 6 in.
13,000	260	100	12 × 10 × 18 in.
17,000	150	40	15 × 7½ in.
18,000	1070	35	(2) 32 × 7 in. × 6 ft.

Ft.	Gal. per hr.	Cost. *l.*	Ft.	Gal. per hr.	Cost. *l.*
100	18,000	100	200	10,000	120
,,	25,000	150	,,	15,000	160
,,	35,000	200	,,	20,000	200
,,	45,000	275	,,	25,000	250
,,	60,000	350	250	5,000	80
150	10,000	80	,,	8,000	100
,,	15,000	120	,,	12,000	130
,,	20,000	175	,,	15,000	160
,,	25,000	225	,,	20,000	200
,,	35,000	300			

Working pressure: 40 lb. per sq. in.

Rotary pumps are rendered cumbrous and heavy by fly-wheels, and cost 15–25 % more than non-rotary, besides requiring better foundations. But their stroke is continuous, and has no inertia to overcome; action is certain and regular; and, as steam may be used expansively, they effect an economy of 30–50 % in working. The Riedler type has superior valve mechanism, permitting full stroke and very high speed, while being simple and easily adjusted; it possesses great power, and is equal to a single lift of 2000 ft. The space occupied by its delivery-pipe is not $\frac{1}{5}$ as great as that required by a Cornish pump of equal capacity, and the shaft-timbering may be much lighter. One of these pumps, at the Chapin iron-mine, Michigan, raises 2200 gal. per min. from 1700 ft., either with steam at 110 lb., or with air at 60 lb. at the pump, and consumes only 8 t. coal per 24 hr. where 30 t. were required before. Another, at the Drumlummon mine, Montana, at 90 rev., lifts 400 gal. per min. against 1200 ft. head, and its working cost is only 25 % of that incurred with ordinary steam-pumps which it replaced, the saving in fuel alone being 40 %. Examples:—

Ft.	Gal. per hr.	Cost. *l.*	Ft.	Gal. per hr.	Cost. *l.*
100	7,000	50	250	3,000	50
,,	12,000	80	,,	5,000	60
200	10,000	120	,,	7,000	110
,,	16,000	160	,,	12,000	150

Working pressure: 40 lb. per sq. in.

The vertical form of simple rotary pump is 30–65 % more costly than non-rotary equivalent, but is more economical, because of heavy fly-wheel and positive cut-off. It is lighter than horizontal pattern, on account of diminished bed-plate.

Compound steam-pumps, using steam expansively, effect considerable economy in fuel, though always far short of surface-driven pumps. Pressures of 80 lb. per sq. in. and upwards must be used to secure any real benefit, and, at 130–150 lb., triple expansion is desirable. These pumps occupy much space, which may be somewhat reduced by arranging cylinders side by side; they are well adapted to collieries, where depth is constant, and fuel costs little, their efficiency being about 80 %. Sometimes they are placed at 200–300 ft. from bottom of mine, and are fed by hydraulic pumps (see p. 503), which are not affected by flooding; and, when a water-tight connection is made between them, giving access through floods, repairs to the hydraulic pump can be effected if necessary. Examples:—

Gal. per hour.	Ft. in one lift.	Dimensions. in. high.	Dimensions. in. low.	Stroke. ft.
24,000	1100	28	50	6
30,000	450	25	50	5
30,000	900	(2) 22	54	6
37,000	1200	33	50	6
40,000	900	35	60	6
42,000	900	(2) 35	60	6
60,000	350	(3) 33	54	6
72,000	300	33	54	6

Ft.	Gal. per hr.	Cost. *l.*	Ft.	Gal. per hr.	Cost. *l.*
350	72,000	2000	700	150,000	7000
600	180,000	3000	1800	30,000	7000

Operating cost in a special trial at an English colliery, coal price not quoted (probably not exceeding 5*s.* per long ton), on a vertical lift of 1341 ft., and boiler pressure of 80 lb., was 4·88 lb. coal per i.h.p.-hr., or 5·7 lb. per pump h.p.-hr., or 2·85*d.* per 1000 gal. Other trials are reported as having given 3½ lb. per i.h.p., but the general average is not far short of 5 lb. per i.h.p. Triple-expansion pumping-engines, using steam at a minimum of 100 lb., and better at 180–200 lb. per sq. in., are much more economical of fuel, their consumption ranging between 1·3 and 1·8 lb. per i.h.p.-hr.; but costliness and complication make them ill-adapted for mines.

Compressed-air Pumps.—Pumps actuated by compressed air are almost as diverse in type as steam-pumps, and vary from the simple air lift (see p. 505), which is not really a pump at all, to the complexities of compound expansion and extraneous heating.

In pumps of the Merrill type, two chambers are employed, submerged in water, compressed air being admitted directly and displacing the water, the chambers acting alternately. An efficiency of 22 % is claimed, but this pump exhausts into the atmosphere at full pressure, and all expansive work contained in the air is lost; compounded, it can be made very efficient.

By the Harris system, the air, after displacing and raising the water, instead of being at once exhausted into the atmosphere, is allowed to do work in expanding against the compressor piston, and thus, practically speaking, all its expansive energy is saved; but losses in leakage and friction amount to 15 %. It has an efficiency of 60–70 %, and is useful for mine station pumping; in fact, it is quite the most economical of all compressed-air systems. It involves the use of two tanks, preferably submerged, to which are connected water inlet and outlet pipes fitted with check-valves. Two air-pipes lead to the upper part of the tanks, from a switch which automatically reverses the flow of air from the compressor, in such a way that, while the water in one tank is being forced out, the other tank is filling, and the discharged air is returned to the intake of the compressor. The full expansion of the air, therefore, is obtained, and energy is conserved to the greatest possible degree. The return air, entering the compressor under pressure, exerts its force upon the receding piston, and supplies part of the power required for compressing air on the pressure side. The additional power to be drawn from the engine is only that required to make up the difference between the pressure of air entering the compressor and that required to lift the water. There is some loss of air due to absorption by the water, which must be made up from time to time; and a small valve is inserted in the compressor, allowing a suitable amount of air to be drawn from the atmosphere during suction. Efficiency depends upon ratio of volume of air tanks to air lines, and, under normal conditions, it is safe to say that an average of 55 % may be secured. Installations have returned, under favourable conditions, as high as 63 %, and never, under unfavourable conditions, less than 50 %, this being the ratio of h.p. of water lifted to i.h.p. in steam cylinders of air compressor, and including all losses. The system is well adapted to the operation of direct-acting pumps, and is self-regulating if properly designed. Reheating can be applied, increasing efficiency and capacity still further. Such an underground closed-air system will work equally well when submerged in water or in a drowned mine, which is a particularly valuable feature.

Direct-acting pumps, using air at full pressure only, have a mechanical efficiency of 65 % and a total efficiency of 15–22 %. In these, the lower the pressure, the greater the efficiency. With properly designed cylinders, efficiency may reach 30 %; but with ordinary mine pressure of 90 lb. per sq. in., efficiency may range between 12 and 17 %. To render these pumps compound, the air must be heated, by (*a*) the water which is being pumped, or (*b*) extraneous heating before initial cylinder, or (*c*) before compound cylinder, or (*d*) both. By passing the air from the initial cylinder through coils of pipe surrounded by the pumped water, the air will assume nearly the temperature of the water, and will be delivered to the second cylinder at practically the same temperature as the first, thus permitting a number of expansions. The efficiency of any ordinary compound pump may be made 37½–40 % by this simple method, or almost double the water can be pumped for the same amount of air used in a simple pump. When extraneous heating is used before the initial cylinder and between the two, efficiency may be 30–72 %.

The Wheeler system is a combination of displacement and air-lift, and gives 34 % efficiency.

The Cummings, or two-pipe system, consists in compressing the air to about 200 lb., and exhausting it back from the pump at 100 lb. This may be made to give an efficiency of 35–70 %, according as reheating is used or not.

The use of compressed air for operating mine pumps has many advantages, but ordinarily it is not understood, and is accompanied by low efficiency, due to applying usual methods of transmission under pressure of 6–8 atmos., arriving at the motor cylinder with reduced temperature, and resulting in using the air with limited expansion, causing freezing of moisture in exhaust-passages.

Electric Pumps.—Electrically-driven underground pumps are sure to come more and more into use since motors have been built so securely enclosed that they may suffer complete immersion without interference with their operation. Efficiency has been computed as follows: dynamo at full load, 90–94% of engine, at

$\frac{1}{2}$ load, 85–90%, at $\frac{1}{4}$ load, not below 75%; line losses, 5–10%. Approximate costs per h.p. delivered by motor are:—

	ft.	volts.	amperes.	*l.*
For 100 h.p. ..	25,000 ..	5000 ..	8·75 ..	40
„ 200 „ ..	„ ..	„ ..	17·50 ..	20
„ 500 „ ..	„ ..	„ ..	26·25 ..	16

With increasing distance, proportions are maintained. Combined efficiency of triple-plunger pumps and motors is said to reach 75%. Economy of electric over steam transmission of power, even on moderate distances, ranges from 30 to 50%. Working voltage is often 400–500.

In this country, continuous-current motors are employed; on the Continent, polyphase motors have been largely used. The objection to the latter is that it is not so easy to arrange matters that, if the pump be stopped suddenly, it can be started again without running off the column of water in the rising main. In many cases, provision is made to drain the whole or part of it into a sump. In others, the pump is joined to the motor by a friction coupling; the motor can then be started light, by interposing resistance, and, as soon as it has reached full speed, the friction-clutch can gradually be thrown in, resistances being simultaneously cut out. The motor is thus not required to take any load until it has its full running torque. The best type of pump for electric driving is, according to Prof. Louis, the three-throw ram, which distributes the strain more uniformly than any other throughout the revolution, an important consideration, as the torque of the motor remains constant. For this same reason, differential plungers are preferable to plain plungers. Most makers prefer horizontal pumps. Louis thinks it probable that the introduction of electric driving will bring with it the construction of fast-running short-stroke pumps, which will be capable of being driven direct by electric motors without running at excessive plunger speed.

Electric pumps are generally finding more favour on the Rand, especially as depth increases. They do not require so much room in a shaft, and, though higher in initial cost, do not cost so much in maintenance. High-tension current, up to 3300 volts, is taken underground, and there transformed down to 110 volts, without danger. Three-throw plunger pumps, with barrels $6\frac{1}{2}$ in. diam. and 8-in. stroke, connected to 50-h.p. triphase induction motors, working through cut gearing with a motor speed of 450 rev. per min., running the plunger crank-shaft at 60 rev., and pumping against 500 ft. head, are general. Some shafts have high-lift triplex single-acting Riedler pumps, $5\frac{1}{2}$-in. plungers, 15-in. stroke, capacity 500,000 gal. per 24 hr., geared to 175-h.p. motors, to work against 1300 ft. head. Such high-lift pumps save considerable expense in cutting chambers (which usually average 3–4*l.* per cub. yd. excavated), and require fewer attendants. If these come into vogue generally, they will probably be fitted with ordinary suction and delivery valves, which are far simpler, and, with due attention, can give a higher factor of efficiency. (Pettit.)

At the 2150-ft. level Comstock mine are 3 Riedlers using 2200-volt current, and, in 6 years, not a single short-circuit nor a single accident has occurred.

The tendency in plunger pumps is to increase speed to the utmost limit of practicability, to drive with comparatively large slow-running motors, and so eliminate gearing.

Recent rapid development of high-pressure rotary pumps has led to construction of motors with abnormally high speed, which are smaller and cheaper than normal motors. In a recent installation, the diameter of rotor of 600-h.p. motor, running at 1035 rev. and 5000 volts, is about 2½ ft.; while in an older slow-running plant, with 60 rev. at 2000 volts and 650 h.p., it is about 15½ ft., the length of the former being 1·63 ft. and of the latter 1·5 ft.

Some think the pump of the future will be of centrifugal type (see below). Advantages are that it costs less for same capacity, occupies less space, requires less attention, lubrication, and repair, and is particularly suited for driving by electricity, as both pump and motor give best results when running at high speed. But efficiency is not quite so high. On the other hand, the electric centrifugal or turbine may be submerged, with its vertical shaft connected direct to horizontal motor, or by bevelled gear to vertical motor. It is often provided with water-tight housing for both motor and pump, and the entire apparatus may be lowered several hundred feet in water, and still perform its work. Still, it is probably not the best pump where supply to suction is erratic, and this is what generally happens in a mine. The location of engine-house is not arbitrary, and the generating station may be in any position most suitable for distributing purposes. Transmission plant is the simplest and most convenient of all systems.

Centrifugal Pumps.—For regular volumes, and not inordinately high lifts, the centrifugal pump is one of the handiest, as it tolerates very dirty water (even containing 30% solids). Efficiency ranges between 25 and 45%, and the general average is about 35%. As centrifugal pumps will not create a vacuum unaided, the casing and suction line need priming before pumping can commence. Massive foundations are not necessary, but the pump should be so placed that perfect alignment is assured. The suction line must be free from air-leaks and, if more than 20 ft. long, should be larger than the pump suction, to avoid undue friction. The discharge line must be of liberal size, and a gate-valve should be located in it, near the pump. Priming may be done as shown. In Fig. 210, A, priming ejector *a* is attached to the highest point of pump casing, and water, air, or steam, under pressure, may be admitted to create a vacuum; in B, an auxiliary hand-pump *b* is mounted on top of discharge casing: gate-valve *c* being closed, hand-pump *b* draws in water till suction and casing are full; while in C, where a foot-valve *d* is used on suction *f*, water is run in at *h* till it reaches discharge-flange *g*. When the pump has been primed, it should be run till full-speed is reached before opening gate-valve on discharge. The "compounding" of the centrifugal pump has given it a new sphere of usefulness in high lifts, and its peculiar adaptation to electric driving has brought it into great favour.

The Burma ruby mines use 12-in. centrifugals direct-coupled to diphase motors running at 800 rev. per min., the lift being 40 ft.

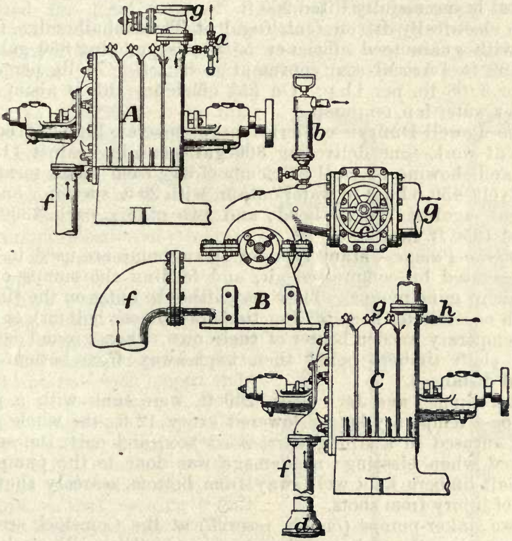


FIG. 210.—PRIMING CENTRIFUGALS.

Lindal Moor iron mines have a number of centrifugals driven by triphase motors, at 3300 volts, current being generated by steam turbines with steam at 200 lb. and superheated to 600° F. (altogether 3000 h.p.), duties varying from 1000 to 4000 gal. per min., and lifts from 300 to 612 ft.

Horcajo lead mines, Spain, pump 1000 gal. per min. from 1450 ft., in 4 lifts, using quadruple centrifugals driven by triphase motors, guaranteed efficiency being 68%, and coal consumption (actual) 4·8 lb. per water h.p.-hr. with poor local coal. With steam turbines and good coal, 3·66 lb. per water h.p.-hr. is guaranteed; and with gas engines, 2·7 lb.

A copper-mine at Butte, Montana, has 2 units each raising 1000 gal. per min. from 1350 ft. in 12 stages.

Many of these pumps are operating in the anthracite coal-field, Pennsylvania. When the mine-water is acid, a special alloy is used (7 copper to 1 tin). At the Hampton colliery is a 5000-gal. 500-ft. installation driven by 800-h.p. motor. At Auchincloss, 1000-gal. 600-ft. At Avondale, 2800-gal. 100-ft.. Water carrying 25% fine coal is successfully lifted 200 ft.

An electrically-driven centrifugal at Tywarnhaile mine, Cornwall, with guaranteed efficiency 55·8%, is pumping 850 gal. per min. 190 ft. Actual coal consumption is 1·5–1·74 lb. per elect. unit, or 1·08 lb. per i.h.p. On 55% efficiency, this is about 2 lb. coal per water h.p. on pump.

The Powell-Duffryn collieries have several high-lift centrifugals at work—one delivering 800 gal. per min. against 1150 ft. head, and showing over-all efficiency of 65% from motor terminals (3000 volt, 450 b.h.p.) to water output, with 20 ft. suction; another 800 gal. against 700 ft. head; and two others, each 1300 gal. against 1650 ft. head.

Sinking-Pumps.—Many types of steam-pump are used in sinking, operated by compressed air, and feeding the sumps of the permanent mine pumps. They may either be hung on the timbers (which necessitates keeping the timbers too near bottom), or fixed on a temporary wooden bearer of their own, when ground is good (main shaft timbers being then kept away from bottom their normal distance).

At a Simmer and Jack shaft, 200 ft. were sunk with a pump fixed on a temporary bearer lowered every 12 ft., the whole pump being encased in a strong karri-wood box, and only the suction removed when blasting: no damage was done to the pump, and the shaft timbers, kept well away from bottom, scarcely showed a trace of injury from shots.

Two sinker-pumps (one in reserve) at the Comstock are used to pump directly to large electric pumps; they will work submerged, connected to a vertical driving rod, on which, at pump level, is mounted an electric motor.

Latest improved sinking centrifugal pumps, direct-connected to electric motors, have capacities of 100–500 gal. per min. against heads of 100–300 ft. They are designed so that they may be raised and lowered in the shaft by rope and pulleys, or be supported in place by timbers laid across the shaft under the Y, or by hooks fastened to the two discharge-pipes. (See also pp. 249–50.)

Pump Columns.—In European countries, cast-iron pipes of considerable thickness and excessive weight are almost universally employed, as they are very cheap, and present a mass of metal to withstand corrosion, while their transportation is generally limited and not costly. Elsewhere, as in the various Colonies, India, the United States and South America, wrought-iron or steel is nearly always substituted, because, though initial cost per foot is greater, this is entirely extinguished by immense saving in long transport. Material economy may be secured by graduating the thickness of metal employed to the pressure which each section of say 100 ft. of pipe will have to withstand. With a riveted pipe, it is necessary to bear in mind shearing effects of rivets, as well as tensile strength of metal. Every pipe should be submitted to hydraulic test at double the pressure it will be called on to bear in work, and be rejected for least sign of weakness. In estimating working pressure, it is not sufficient to calculate simply lb. per sq. in. represented by column of water at rest; a very wide margin must be allowed for pulsation and jar—ordinarily not less than 50% extra, and in extreme cases as much as 150–200%. In this respect, resisting strengths of cast iron, wrought iron and steel plate have proved to be in proportions of 1, 6, and 20, which fact should suffice to determine selection of steel pipe under all circumstances. (See also pp. 30–41.)

Where possible, the weight of pump column should be taken off the mine timbers by using clips and chains suspended from short iron rods with looped ends secured in the rock.

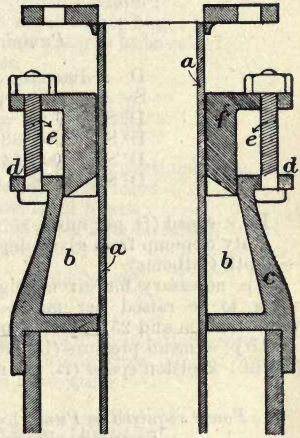


FIG. 211.—EXPANSION-JOINT FOR SINKING.

An ingenious expansion-joint for sinking-pumps is made (Fig. 211) from an ordinary straight piece of pipe *a*, filed smooth, so that packing *b* will not be destroyed. A stuffing-box *c* is cast shaped like a reducing coupling, save for shoulder utilised to confine packing, and lugs *d* on top to receive bolts *e* connecting with gland *f*. Asbestos or other pump spiral packing is used for steam inlet and exhaust, while hemp suffices for water. The sinking-pump is hung from a differential pulley, which permits lowering of pump to full length of joint, and, by disconnecting flanges, speedy insertion of a full length of fresh pipe, instead of short lengths. The complete change is made in ½ hr., at intervals of 10–14 days, according to progress in sinking, instead of changes in suction or discharge every 2–3 days, occupying several hours. The joint is applicable to all patterns of sinking-pumps.

Wooden pipe is used with remarkable success for horizontal situations, where impure water would corrode iron or steel pipes in a very short time. It is regularly made in sizes from $1\frac{1}{4}$-in. to 20-in. bore, and can be made to almost any size. Bound with steel hoops, it will handle any pressure up to 400-ft. head, or 160 lb. per sq. in. The joint is simple in the extreme, and pipe can be laid by driving lengths together, without skilled labour; natural expansion of wood when wet makes joints perfectly tight. Pipe is made in lengths of 4–8 ft., and each piece is tested hydraulically. When used underground, it is coated outside with asphaltum pitch, which preserves both hoop and wood. Wooden elbows, tees, bends, crosses, reducers, plugs, tubes, and pump and valve connections with iron nipples, are furnished ready for laying, without cutting or other preparation. (See also pp. 35–6.)

Pumping Data.

D = diameter of pump (in.).
S = stroke of pump (in.).
$D^2 S \times \cdot 7854$ = cub. in.
$D^2 S \times \cdot 002833$ = gal.
$D^2 S \times \cdot 0004545$ = cub. ft.
$D^2 S \times \cdot 02833$ = lb. fresh water.

$D^2 \times$ speed (ft. per min.) $\times \cdot 034$ = delivery (gal. per min.).

Duty of pump from given depth (gal. per min.) = (i.h.p. $\times$ 550) $\div$ depth (fathoms).

h.p. necessary for given height and weight = weight (lb.) of water to be raised per min. $\times$ height (ft.) $\div$ 33,000 + 25% for water friction and 25% for loss in steam cylinder.

i.h.p. = mean pressure (lb. per sq. in. of piston) $\times$ area of piston (sq. in.) $\times$ piston speed (ft. per min.) $\div$ 33,000.

Power required to Pump 1 gal. Water against Various Pressures, allowing 40% for friction.

Pressure per sq. in.	h.p. calculated 40% added.	h.p. required.
700 lb.	·68	say ·75
1500 „	1·47	„ 1·75
2240 „	2·19	„ 2·5

To find Diameter of Single-acting Pump.

L = Length of stroke (ft.).
G = Number of gal. to be delivered per min.

F = Number of cub. ft. to be delivered per min.
N = Number of strokes per min.
D = Diameter of pump (in.).
$F = \cdot 00545\ D^2\ L\ N.$
$G = \cdot 034\ D^2\ L\ N.$

$$D = \sqrt{\frac{G}{\cdot 034\ L\ N}}.$$

$$D = \sqrt{\frac{F}{\cdot 00545\ L\ N}}.$$

Note.—These formulæ give the net diameter of the pump-plunger; it is usual to increase the area of the plunger $\frac{1}{4}$, to allow for leakage, etc.

To find horse-power of Pumping Engine.

(*a*) Let G = number of gal. required per hour;
C = number of cub. ft. required per hour;
F = height (ft.) to which water is to be raised;
h.p. = actual horse-power.

$$\text{Then h.p.} = \frac{G \times F}{198{,}000} \text{ or } = \frac{C \times F}{31{,}750}.$$

70–80% must be added, to allow for friction and contingencies.

(*b*) G = Number of gal. to be raised in 24 hours.
F = Number of cub. ft. raised in 24 hours.
h = Height (ft.) to which the water is to be raised.
h.p. = Actual horse-power required.

$$\text{h.p.} = \frac{G \times h}{4{,}752{,}000} \text{ or } \frac{F \times h}{762{,}088}. \quad \text{Add 70–80\%.}$$

Pumping Costs.—Only exceptionally are such detailed accounts kept as will permit segregation of pumping from general mine costs, but the following examples are interesting:—

Transvaal Pumping Costs. (Pettit.)

—	Electric Pumps.			Cornish Pumps.		Baling.
	a	*b*	*c*	*d*	*e*	*f*
Duty: gal. per mo.	1,500,000	7,020,000	26,920,000	6,000,000	4,950,000	8,486,057
Depth: ft.	1500	1000	1000	1000	1035	1250
Cost per 1000 gal.	2*s.* 0·4*d.*	2*s.* 11·5*d.*	1*s.* 5*d.*	1*s.* 1·25*d.*	1*s.* 7*d.*	10·5*d.*
Cost per 100 ft.-ton	·325*d.*	·710*d.*	·340*d.*	·265*d.*	·367*d.*	·169*d.*

Victorian Deep-lead Pumping Costs.

—	Berry Mines.	Chiltern and Rutherglen Lead.	Avoca Lead.
Depth	550 ft.	400 ft.	200 ft.
Cost of plant per 1,000,000 gal. pumped per day	6000*l*.	4000*l*.	2500*l*.
Cost per 1000 gal.	1·4*d*.	1*d*.	·6*d*.

Baling, at Llanbradach colliery, costs ·052*d*. per 100 ft.-ton. (Galloway.)

Cornish pump (beam-engine) at Holmbush mine, Cornwall, had a duty of 81,400 ft.-tons per cwt. of coal; with coal at 18*s*. per long ton, pumping costs, including wages in engine-room and shop, were about ·1*d*. per 100 ft.-ton. (Fischer Wilkinson.)

Cost of pumping by gasoline engine (with crude Californian oil at 2*d*. per gal.) on 12½ million gal. raised 164 ft., including salaries, but no repairs or renewals, was under 6*d*. per 1000 cub.-ft., or ·92*d*. per 1000 gal. (Harroun.)

VENTILATION.

IN metalliferous mining, natural air-currents are largely relied on, sometimes assisted by artificial inducements. No mine of any magnitude is without at least two shafts, and this in itself is sufficient to generate a flow of air through the main workings, fresh air descending through one (downcast) and foul air ascending through the other (upcast). It is curious that their relative functions are not constant, and that interchanges of direction of current are not infrequent, which must be borne in mind in arranging doors, etc.; but each will be predominantly upcast or downcast, and the timbering of the former will suffer therefrom. In a single shaft, upward and downward currents are separately accommodated by bratticing off pump-way from hoisting-way. Thin boarding is perhaps best as a permanent structure; but often recourse is had to strong coarse canvas, known as brattice-cloth, and this is quite effective, much cheaper, and surprisingly durable, especially if tarred (in a wet place) or soaked in a solution of tungstate of soda (as a preventive of fire, in a dry one). The same material is frequently used in drives for splitting the current.

In long or dead ends, it is advantageous to lay thin and light galvanised-iron pipes along the roof as an intake for foul air. They may discharge simply into the shaft, or it may be necessary to carry them to surface, and even to a considerable height above it. With a sufficiently large shaft-pipe, a number of ends may be simultaneously served by branch-pipes. Sometimes it is necessary to supplement natural out-draught by a fan or a jet of compressed air, either below or at surface. Tops of rises are always very hot and foul, and here it would sometimes pay to put a 1-in. diamond-drilled hole through to the next level, and keep a tiny jet of compressed air blowing across it. Where compressed air is used for driving drills, the exhaust is in itself a powerful aid to ventilation. But transmission of compressed air is much too costly and wasteful of power for such an air-current to be used as a direct means of ventilation; besides this, it is unhealthy for breathing—fresh air should be induced to flow in by the displacement caused by foul air being forcibly drawn out.

With an electric drill plant, an exhauster can be run by small motor at very small cost. Thus, in one case, for exhausting foul air from over 900 ft. in a tunnel, 2 h.p. for 2 hr. sufficed, at a cost of 1*s.* 6*d.* for fuel, and a small charge for attendance; a 10½-in. exhauster connected to 3-h.p. motor, with 1000 ft. of 12-in. galvanised piping, cost 75*l*. (Dane.)

Where Cornish pumps are at work, the old "duck-machine" or Hartz blower may be used successfully. This can be made by any mine carpenter, and consists simply of two boxes, one fitting within the other, the smaller being attached to the pump-rods, and moving up and down with them; a valve in its top admits fresh air on each stroke, and a corresponding valve in an outlet pipe in the outer box emits it, or, by having the valves reversed, foul air can be exhausted in the same way. The outer box contains some water to act as a seal. If high-pressure water is available, or can be obtained from a small pipe let into the rising main of the pump, it is often utilised more simply by merely turning a jet into the bell-mouth of a ventilating pipe, the jet driving the air forward. The jet may also be arranged to exhaust the air. Air driven in by a water blast is often much liked by the men on account of its coolness and freshness.

In deep-level mining, abundant cool air becomes still more necessary, to moderate natural increase of temperature due to heat of rock, which augments at rates varying from about 1° F. in 50 ft. to 1° in 500 ft. In some Victorian gold mines, this is a most vital question, temperature of air and water at below 2000 ft. being near 150° F., which is barely endurable by men working, even though the "shift" be reduced to 4 hr. Powerful exhaust-fans somewhat remedy this, and, in summer, air passing down is partially refrigerated. The Silver King mine, Park City, Utah, has a system of artificial ventilation with pipes on the different levels. In the Comstock, the rock itself is hot, but the air is fairly good; underground water is 160° F. in some parts. Owing to the many shafts, there is naturally a good circulation of cool, fresh air through the main workings, directed by partitions and brattices, and in this way the temperature is kept down to about 90°. Sometimes both a suction fan and a blower have to be used at the same time to keep the drifts cool enough, and in some places even they do not suffice, and a pipe of cold water is carried from surface and sprayed upon the men as they work, blower and suction fan being kept running; in this way only is it possible to work in the hottest places.

Cost of mine ventilation at Center Star, Rossland, B.C., per lineal ft. of work, varied from 1*s*. 7*d*. in driving to 5*s*. 3*d*. in sinking main shaft.

At great depths, such as are anticipated on the Rand, with a high consumption of explosives and a liberal allowance of men per ton broken, ventilation is likely to be extremely important.

Formulæ.

(1) t = temperature of air in downcast shaft.
T = temperature of air in upcast shaft.
D = depth of shaft in ft.

m = periphery of transverse section of air-course (ft.).
s = area of section (ft.).
l = length traversed by current (ft.)
v = velocity of current (ft. per sec.).

$$v = 96 \sqrt{\frac{\frac{(T - t)}{T + 448} D s}{m\, l + 368\, s}}$$

(2) C = length of downcast.
c = length of upcast.
T = number of degrees in excess of 32° F. in C.
t = number of degrees in excess of 32° F. in c.

$$c = C \left(\frac{480 + t}{480 + T}\right).$$

Duty of Furnaces at Collieries in Northumberland and Durham Coalfield. (Cochrane.)

Name of Colliery.	Down-cast Shaft. Diam.	Down-cast Shaft. Depth.	Upcast Shaft. Diam.	Upcast Shaft. Depth.	Area of Furnace Grate.	Temperature of Air. Return Air near Furnace.	Temperature of Air. Bottom of Upcast.	Temperature of Air. Top of Upcast.	Volume of Air circulated per Minute.	Water Gauge in the Mine.	Consumption of Coal per hr. per h.p. in Air.
	ft.	yd.	ft.	yd.	sq. ft.	F.	F.	F.	cub. ft.	in.	lb.
Rugeley	12	160	12	160	64	61	141	110	103,325	·62	37·0
North Seaton..	15½	250	9	266	72	65	225	186	99,750	1·10	49·2
Ryhope	15	508	10½	460	160	76	170	134	126,336	1·00	56·3

Comparative costs of equal volumes of air, ascertained by Forster, are approximately, per 100 cub. yd. per min., 365*l.* by Root's blower, 1510*l.* by Körting blower, and 9235*l.* for compressed air, per ann., with steam power costing 12*l.* 10*s.* per gross h.p. per ann.

Efficiencies of Mechanical Ventilators. (N. Eng. Inst. Min. Engs.)

No.	Name of Ventilator.	Dimensions of Ventilator. Diameter.	Width, &c.	Theoretical Displacement per Minute.	Diameter of Inlet.	Weight.	Dimensions of Engines. No. of Cyls.	Diam. of Cylinders.	Length of Stroke.	Direct-acting or Geared.	General Results. Volume of Air per Minute.	Mean Water Gauge at Drift Door.	Percentage of Useful Effect.
		ft. in.	ft. in.	cub. ft.	ft. in.	tons		in.	ft. in.		cub. ft.	in.	
1	Guibal ..	Fan .. 50 0	.. 12 0	..	15 0	50	1	42	3 6	Direct.	108,422	3·30	40·00
2	„ ..	„ .. 46 0	.. 14 10	..	13 0	..	1	36	3 6	„	246,509	1·85	52·95
3	„ ..	„ .. 40 0	.. 12 0	..	14 0	24	1	36	3 0	„	170,581	1·46	47·95
4	Waddle..	„ .. 45 0	{ Inlet .. 6 6 Periphery 1 5 }	..	15 0	..	1	32	4 0	„	163,312	3·08	52·79
5	Schiele ..	„ .. 12 0	.. 2 1	..	..	..	1	25	2 0	2·57 to 1	157,176	1·91	46·12
6	„ ..	„ .. 9 6	{ Inlet .. 3 2 Periphery 1 8 }	..	8 0	..	1	20	1 8	2½ to 1	106,570	2·03	49·27
7	Lemielle	{ Chamber 22 6 Drum .. 15 0 }	Height 32 0	9·9 rev. 108,900	..	..	1	55	6 0	Direct	47,307	1·37	23·40
8	Struvé ..	2 pistons 18 3	Stroke 7 0	6·50 „ 47,827	..	..	1	24	4 4½	4 to 1	43,793	5·11	57·80
9	Nixon ..	{ 2 pistons— Long.. 30 0 High.. 50 0 }	Stroke 7 3	7·19 „ 120,790	..	..	1	36	6 0	Direct	72,595	2·74	45·91
10	Root ..	2 drums 25 0	.. 13 0	16·71 „ 96,918	..	..	2	28	4 0	„	89,772	3·29	47·84
11	Cooke ..	{ 2 drums 15 0 Casing .. 22 0 }	.. 11 6	17·92 „ 80,640	..	..	1	25	3 6	„	54,190	1·12	37·33
12	Goffint ..	2 pistons 13 2	Stroke 10 7¾	9·25 „ 53,020	..	..	2	15¾	10 7¾	„	36,286	0·71	25·79

SANITATION.

This important matter is very commonly disregarded entirely, men being allowed to relieve themselves almost wherever they please. Such a filthy custom is most injurious to their health. There is no difficulty whatever in selecting convenient corners, screened off by a few yd. of brattice-cloth, and placing there large galvanised-iron pails. The author's custom is to make it a duty of the boss of the night shift to see that every pail is sent up, emptied, and sent back to its place filled with charcoal dust from the smithies or with ashes from the boilers; this is tipped out alongside the pails for periodical use. Disinfectants should also be provided. Similarly, places are appointed where the various groups shall assemble for eating their meal at mid-shift, and each

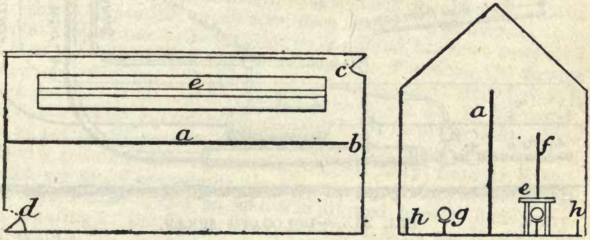


FIG. 212.—CHANGING-ROOM.

is furnished with an old tin-lined fuse-tub, into which all fragments, paper wrappers, and so on, are thrown. This greatly lessens the attraction for vermin of all kinds, and excludes much greasy and otherwise undesirable matter from milling-dirt.

Changing-rooms at shaft-heads are made compulsory in many countries, and should always be erected. The room (Fig. 212) is split longitudinally by a partition *a* about 7 ft. high, with a passage through it at one end. In one half *c*, men hang their surface clothing on a series of numbered pegs affixed to partition *a*, and pass through *b* to take their underground clothing from corresponding pegs in *d*. A double seat *e*, with rail *f* above it (for spreading wet clothing on) and waste-steam pipe *g* beneath it (for warming room and drying garments) occupies the centre of each compartment, and a boot-rack *h* runs along each outer wall. The door of compartment *c* faces away from shaft and that of *d* towards it. Abundant windows for ventilation and light are provided in outer sides, but none in end walls. Where it is necessary to safeguard against ore-stealing, a trusted attendant stands at *b* during change of shift, and examines every man as he passes through after stripping. An improvement would be to make the floor of concrete (to permit hosing out), and to provide a washing compartment, preferably simple pails.

Condensation of fumes from explosives and removal of dust from air underground are both very desirable to minimise lung troubles. Nothing is so effective as a water spray. This may be

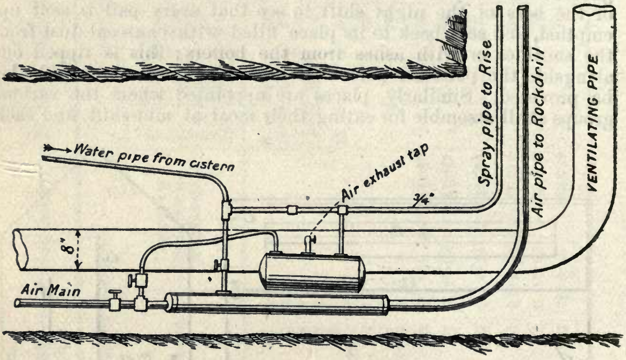


FIG. 213.—DOLCOATH SPRAY.

simply furnished by carrying an iron water-pipe along with the compressed-air pipe, and conveying the water into the jet through a piece of armoured hose of very small bore. A spray used at Dolcoath (Fig. 213) consists of a steel cylinder, holding about 2 cub. ft., placed near the drill but out of the way of blasts, and furnished with 3 taps, one for filling with water, one joining compressed-air main, and one leading to spray-hose. Ordinarily, pressure suffices for creating spray at 100 ft. above cylinder; and a garden-syringe nozzle, with a filter of fine copper gauze inside, is suitable. The most effective spray is obtained by a small quantity of water, when compressed air is used as a carrier. By adjusting taps, any variety of spray, from merely moist air to a full flush of water, can be directed on the dust it is desired to settle. (See papers in Tr. Inst. M. M.)

LIGHTING.

In metalliferous mining, the popular illuminants are oil (chiefly colza) lamps in many European countries, and ordinary candles in most others. The preference for oil lamps is not easily understood; they need much more attention, emit more smoke and smell, and are more awkward.

Candles need to be of good quality, or they will readily bend and become useless under the influence of warmth, and will "gutter" and waste excessively in a draught. Where operations are extensive, it is advisable to get the manufacturer to insert a coloured strand in the wick for identification, as a check on pilfering. Costs vary widely, depending much upon fiscal bases. The average item for total underground illumination on the Rand has been computed at about 4*d.* per ton of ore raised; at the Ferreira, the figures quoted are ·67*d.* per ton for stoping, 6·17*d.* per ft. driven 7 × 5, 8·68*d.* per ft. 7 × 10, and 6·05*d.* per ft. risen 9 × 5. At Lucknow, New South Wales, consumption averaged 3·32 lb. per long ton of ore won, and 4·68 lb. per lin. ft. of development work, and the cost gradually receded (with diminishing import duty) from 1*s.* 5·82*d.* to 9·3*d.* per ton of ore raised; the issue to miners and truckers was under strict control, and no man was allowed to carry ends away. Mine lighting at the Center Star, Rossland, B.C., per linear ft. of work, costs—

	Sinking Main Shaft.	Sinking Small Shafts.	Rising.	Driving.
	s. *d.*	*s.* *d.*	*s.* *d.*	*s.* *d.*
Candles	3 1	1 8½	1 11½	1 0½
Electric	3 9	..	0 8½	0 10

The candle-"stick" may be simply a bit of wire coiled at one end and hooked at the other, or a more elaborate product of the blacksmith's art, embodying a spring clip for the candle, a hook for catching on hard projections, and a long point for inserting in crevices, timber, or soft ground. A very effective holder is a small lump of well-worked tenacious clay; in fact, except for carrying about, it is perhaps best of all, and not worth stealing. A portable candlestick, suited to the mining engineer rather than the miner, (Fig. 214) consists of: *a*, piece of bent sheet brass about ¼ × ⅛ in.

the fore-finger passing through loop *b*, and third finger through *c*, leaving space *d* for middle finger; a piece of No. 16 copper wire *e*, is soldered to top loop *b* and to brass thimble *f*, the latter being split to clasp candle *g*. It might be simplified by uniting thimble *f* to back of *a* without intervening wire; and still further by using wire only, the coil which holds the candle coming midway between two finger-loops. This last is perhaps best of all. Iron wire is as effective as copper, and offers less inducement to petty thieves.

☞ According to the Mine Commission, the safest oils are rape, colza, and seal. None of these is explosive. Petroleum alone is liable to explode. In brilliancy, seal oil is superior to rape and colza, and the wick is less prone to become charred. By addition of 1 part petroleum to 2 parts rape oil, light is increased. Many oils in common use have a tendency to incrust the wick and lower the flame. Addition of petroleum or benzine reduces this, and yields a better flame for testing purposes in coal-mines.

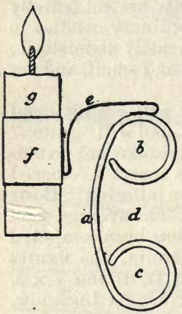


FIG. 214.
CANDLESTICK.

The Wolf lamp has met with practically unanimous approval in Europe. Its chief characteristics are: (1) adaptation for burning naphtha, with greatly increased illuminating power and sensitiveness of flame; (2) igniting arrangement which permits relighting lamp without danger in case flame is extinguished; (3) lock or fastening which can only be opened by a very strong and heavy magnet. The flame is particularly bright, and has about double the candle-power of the best safety-lamps of other types. Cost of naphtha is about $\frac{1}{4}$ that of oil. The lamp is not intended for testing purposes: it will show the presence of gas when as low as 1 %, but it will go out before the mixture reaches explosion point. The Wolf-Stuchlik lamp is constructed as an acetylene lamp, and gives light equivalent to 7 c.-p., as against ·6 c.-p. of the best oil-burning lamps, and 1 c.-p. of the Wolf lamp; a horizontal reflector throws light against the roof when needed. For night operations in open workings, such as quarries, alluvial mines, dredging plants, tips, landing-stages, engine-rooms, changing-rooms, picking-belts and sorting-floors, and in crushing and dressing-mills, electric lighting is very useful. Preference is given to arc lamps, where distribution is not essential, because it is much more economical of power, 1 b.h.p. yielding 800–900 c.-p. as against 200 c.-p. with incandescence. Arc lamps are often 500–1000 c.-p., carried at a height, on wooden or iron poles, and taking a current of about 10 ampères at 40 volts. They should be readily adjustable at suitable elevations, and capable of withdrawal into safety during blasting operations.

Incandescent lamps of 8–32 c.-p. are most useful underground at fixed points where work is constant or concentrated, such as plats, loading-stations, winches, pumps, fans, diamond-drills, and so on. At all such places, occasional whitewashing of surrounding rock, timber, etc., is well worth the trouble.

No satisfactory portable electric lamp has yet been made.

Electric lighting at collieries is usually independent of the power plant, and, to economise labour, lighting dynamos are frequently placed where the fan engineman can see to them. From various causes (chiefly coal-dust, steam from condensing engines, and vibration due to winding gear and screens), it is more difficult to maintain high insulation resistance on surface than underground, and it is good practice to use a separate dynamo for underground lighting. Multiphase currents are more suitable than direct currents for colliery work; oil switches are satisfactory for alternating, but cannot be safely used for direct currents.

Costs by different Systems at Colliery raising 1000 *t. a day.*

Paraffin.—Taking 2 × 1¼ in. wick duplex lamps as 16 c.p., consuming $\frac{1}{10}$ pint of oil (at 8*d.* per gal.) per hour.

	£	s.	d.
1. 40,600 lamp hours = 507·5 gal.	16	18	4
2. 34,800 " = 435 "	14	10	0
3. 6,960 " = 87 "	2	18	0
4. 12,180 " = 152·25 "	5	1	6
5. 58,000 " = 725 "	24	3	4
6. 69,600 " = 870 "	29	0	0
Total cost of oil	92	11	2
Wick, say	3	0	0
Labour, trimming, etc.	25	0	0
Interest on capital outlay of, say, 80*l.* at 5 %	4	0	0
Depreciation, repairs, and breakages, at 20 % on 80*l.*	16	0	0
	£140	11	2

Or ·116*d.* per ton.

Gas.—Capital outlay may be taken, for main, pipes, and fittings, at about 150*l.* At 3 c. p. per cub. ft. per hour, and 5 ft. per 15 c. p. lamp:—

	£	s.	d.
1. 40,600 lamp hours at 5 cub. ft. = 203,000 cub. ft.	25	7	6
2. 34,800 " " " = 174,000 "	21	15	0
3. 6,960 " " " = 34,800 "	4	7	0
4. 12,180 " " " = 60,900 "	7	12	3
5. 58,000 " " " = 290,000 "	36	5	0
6. 69,600 " " " = 348,000 "	43	10	0
Taking the cost at 2*s.* 6*d.* per 1000 cub. ft. Total cost of gas	138	16	0
Interest and depreciation on 150*l.* at 10 %	15	0	0
	£153	16	9

Or about ·13*d.* per ton raised.

Electricity.

	c. p.		Hours per day.	Lamp hours per annum.
1. Pit head, 2 × 200 c.p...	= 14 × 16	averaging ..	10 ..	= 40,600
2. Winding-engine, fan, boilers, pumps, etc. ..	= 12 × 16	,, ..	10 ..	= 34,800
3. Shops, offices, etc. ..	= 16 × 16	,, ..	1½ ..	= 6,960
4. Screens, sorting, etc., 4 × 200 c.p	= 28 × 16	,, ..	1½ ..	= 12,180
5. Underground	= 20 × 16	,, ..	10 ..	= 58,000
6. ,, (continuous lighting)	= 10 × 16	,, ..	24 ..	= 69,600
	100 × 16			222,140

Total candle-power is about 2124, though with electric light this will be more effective than with other lights, on account of greater facilities for reflection. To get i.h.p. per ann., divide total lamp hours by 10 (each 16-c.p. lamp takes ·1 h.p.); allowing 10 lb. slack coal per i.h.p., at 1*s*. 6*d*.

					£	*s.*	*d.*
Coal, 100 tons at 1*s*. 6*d*.					7	10	0
Renewal of lamps at 1500 hours' burning:—							
2900 hours.	1.	2 × 200 c. p. lamps renewed	twice	at 18*s*. each	3	12	0
2900 „	2.	12 × 16 „ „	„	4*s*. „	4	16	0
435 „	3.	16 × 16 „ „	⅓ times „	4*s*. „	1	1	4
435 „	4.	4 × 200 „ „	⅓ „ „	18*s*. „	1	4	0
2000 „	5.	20 × 16 „ „	twice „	4*s*. „	8	0	0
6960 „	6.	10 × 16 „ „	four times „	4*s*. „	8	0	0
Interest and depreciation at 10 % on capital outlay—200*l*. ..					20	0	0
Oil, water, waste, etc.					5	0	0
					£59	3	4

The total cost of electric lighting will therefore be about 60*l*. per ann., on 290,000 t. per ann. = ·05*d*. per ton raised.

Thus electricity compares well as to cost. There remains the question of safety as regards ignition of explosive gas, in case of fracture of the glass bulb containing the carbon: see Tr. I. M. E., xvii. 88.

SIGNALLING.

THERE is hardly any operation connected with mining where more care has to be exercised than in the signalling between engine-room and underground, and it is of truly vital importance that communication shall be certain and that misconstruing of messages shall be avoided. Yet in very few instances is any provision made for signalling from the engine-room to the mine, though there can be no assurance that a signal has been understood unless it is repeated by the recipient.

The signals almost invariably consist of a certain number of strokes on a bell, but nearly every mine adopts an arbitrary code of its own, and the writer has seen different codes in separate shafts on the same mine. This may easily lead to an accident. A universal code for all countries would be a desirable consummation, and the following is suggested as meeting practically all cases:—

1	Bell.	Stop, if in motion; lower, if at rest.
2	Bells.	Raise.
3	„	Start or stop Pump.
4	„	Start or stop Compressor.
5	„	Send Tools down.
6	„	Send Timber down.
7	„	Put on Baling-tank.
8	„	Put on Man-cage.

Each signal would be followed by as many bells as the number of the level at which the engine is to stop: thus 6—5 would mean "send timber to the 5th level"; 2—2, "raise to 2nd level." Only signals 1 and 2 can involve moving men, and on these the engine should be started slowly; or the presence of men may be intimated by a preliminary single bell, with twice the interval between it and the signal proper as is used between two halves of a signal; or, where electric bells are used, the warning "men" may be conveyed by a prolonged ring. Intervals or pauses in ringing by any system should be regular and constant, and be twice as long between portions of a message as between the bells constituting one portion. The whole message should be repeated by the driver before he proceeds to carry out instructions, but, on receiving message, he should move the engine sufficiently to ensure that the cage is hanging free, and to enable chairs to be withdrawn from beneath the cage, if necessary.

By far the most common method of signalling is by a knocker or clapper striking a sheet of metal, or a bell, or a drill bent into a triangle, either of these being suspended freely, so that the sound may not be muffled. By the arrangement in Fig. 215 (Herzig), no greater pull on the lever is necessary from one level than from

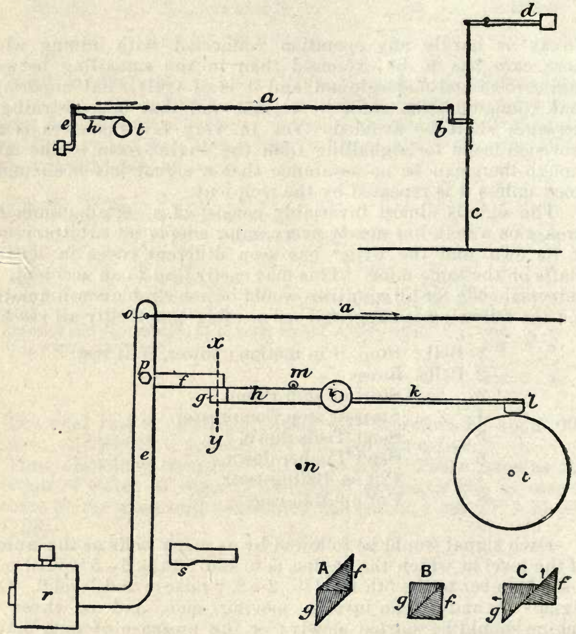


FIG. 215.—KNOCKER-LINE BELL.

another; weight of pendent bell-rope is counterbalanced, so that there is no direct weight on the working parts of the bell itself, and only a light pull on the lever is needed in signalling; whenever the signalling-lever underground passes through a certain arc, a signal is given in the engine-room. The bell-rope is of galvanised-iron wire 7/32 in. diam.; after determining the length needed, its weight is estimated so that it may be counterbalanced properly. On locating the bell in the engine-room, wire *a* is led off horizontally, and, by means of bell-crank *b*, is connected to main signal-rope *c* passing down shaft. Weight *d* counterbalances *c*, and should be heavy enough to carry wires back to normal position, therefore it must be equivalent to weight of pendent rope plus a factor to overcome friction. Working parts of bell are weighted lever *e*, to upper end of which connecting-wire *a* is attached; arm *f* projecting from it is made of steel, about 5 in. long, and has a bearing surface of 1 in. at further end, resting on a similar surface *g* on *h* (cross-section through *x y* is shown in A); *h* is a piece of flat spring-steel about 8 in. long; *i* serves as a hub, to which *h* and *k* are attached, *k* being a flat spring similar to *h*, but 12–14 in. long, at end of which is knocker *l*; *h* is vertical; *k* lies flat, and should not be as heavy as *h*. When the rope is pulled to signal, wire *a* moves in direction of arrow, and arm *f*, travelling downward in a circular path, forces *g* to move downward around *i* as a centre, until the arcs in which they are travelling diverge far enough for gravity to carry *h k* back to normal position, *h* being arrested by stop *m*; the force with which the arm flies back causes the knocker to strike gong *t*, and thus give signal. As soon as the bell-rope is released, lever *e* is carried back to original position, and, in so doing, the triangular surface on end of *f* slides upon inverted triangle *g* (see B), forcing it to one side (see C), until *f* rests in normal position on top of *g*, when it is ready for another pull, as shown in A. In counterbalancing the pendent bell-rope, sufficient weight must be added to bring the levers underground back into position automatically. The counterweight may be located at any convenient place, provided it is above bell-crank *b* to which *a* is attached. In order to prevent the pull necessary for signalling from becoming excessive, it is desirable to reduce friction as much as possible. A force equivalent to a 10-lb. weight should be sufficient for pulling the levers in a well-arranged signalling-system. To accomplish this, it is advisable to bring bell-rope into proper alignment by plumb-lines, and to hold the rope in position by short pieces of old iron pipe 2–3 in. long, instead of by staples. In inclined shafts, the bell-rope should be supported at frequent intervals on some form of roller, such as the spools on which connecting-wire is sold. Arm *k* should be long enough, and knocker *l* heavy enough to cause their instant return to position when the paths of *f* and *g* part company. For this purpose, it is also advisable to put a stop *n* below *h*, so that by no possible chance can it fly beyond its balance-point. By increasing or diminishing the bearing of triangles *f g*, the arc travelled can be changed. It is not advisable to increase this bearing above 1 in. The triangles, as shown in section (A, B, C) should have bases of about $\frac{3}{4}$ in., with perpendiculars of $\frac{3}{4}$–1 in.; their pivots should be good, so that there will be no lateral motion, or they may fail to engage properly. By moving the position of *o* with respect to pivot *p*, the pull necessary to signal can be regulated. Spring *h* must be quite stiff and of good material; but the stiffer the spring, the heavier must be the weight *r*, and, in consequence, the greater the effort required to pull the levers; nevertheless, *r* must be heavy enough to bring back wire *a* and to overcome the resistance of spring *h*. If arm *e* is made sufficiently heavy, a counterweight may be dispensed with, although it is preferable to arrange as shown. If *r* is too heavy, arm *e* has a tendency to pound against stop *s*, but its weight must not be cut down to such an extent as to impair the instant return to position of arm *e*. The whole apparatus may be mounted on a board and placed at some convenient spot in the engine-room.

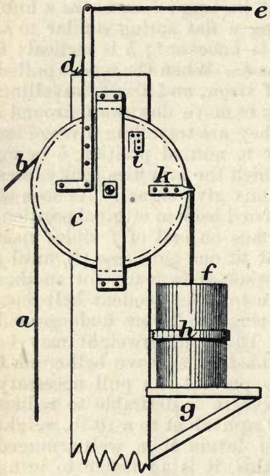


FIG. 216.—SIGNAL FOR 1000-FT. SHAFT.

Another mechanical system common in Mexico and California, especially in shafts 500–1000 ft. deep, whether vertical or incline, is seen in Fig. 216. Bell-rope, galvanised iron-wire rope, ¼ in. diam., from shaft, whether vertical *a*, or inclined *b*, winds round wooden drum *c*, with sides bolted on; L-shaped lever *d* is bolted to one side of drum, at one end of which is rope *e* to engine-room. Counterweight *f* (iron cylinder or small oil-drum filled with scrap-iron) just counter-balances weight of bell-rope, making it possible to pull bell-rope with very slight exertion; it rests on bracket *g*, and usually guard *h* prevents its tipping over. Over-straining is prevented by lug *i*, placed according to length of pull necessary to ring properly. The bell-rope may be let out as sinking progresses, by first detaching lever *d* and lug *i*, both of which are bolted to the drum, and then loosening clamp *k*.

The bell and rope system becomes utterly inapplicable at great depths, because the elasticity of the rope and the tension required are such that there is no certainty in ringing.

Pneumatic signals worked by compressing a piston and so blowing a whistle, have been tried; good enough for short distances, they are not trustworthy for great depths.

Electric signalling is much simpler and more rapid, besides entailing no strain or stress on apparatus. It possesses several important features which place it immeasurably ahead of all mechanical methods; messages can be conveyed simultaneously to every level; signalling can be done to and from a moving cage; driver can send as well as receive a message: and the bell can be supplemented or replaced by a flashlight in those cases (more numerous every day) where shafts are lit by electricity.

Where a plat-man or cage-tender is employed, and it is necessary to call him, and the cage from one level to another, or where balanced skips require signals to be arranged for them, 3 wires are run down the shaft, 2 being connected to the poles of a battery, and the third or middle wire being used to connect all bells in series and ring them simultaneously when a circuit is made through any push on the system. All bells are placed between the middle wire and the same pole wire, and the pushes between the middle and the other pole wire. A separate signal must be arranged from the pit-mouth to the engine-room and vice-versa, for the driver cannot otherwise tell whence the signal comes, whether from the top or the lower set of skips. A bell at the mouth keeps the banksman informed as to what the skips are doing. Communication with the signals below must only be possible by the engine-driver's apparatus. The chief disadvantage is that the driver has no means of knowing the places of the skips except by the correctness of the signal given; but the bells sound simultaneously through the mine when rung from any one place, and men at any level know where the skip is and what it is doing.

In another system, a separate wire is run to each level, the two battery wires being run to each level also: when a signal is sounded, it rings only the bell at one place and a bell or indicator at the mouth and in the engine-room. This system simplifies the code of signals; the driver knows what he is doing, but cost and complexity are increased. The first system is recommended by Clements (Trans. S. Af. S. E. Engs.). Wires may be run in iron pipes, or in tarred wooden conduits along roof timbers in an incline; they are No. 18 S.W.G., lead-covered, and held in position by wooden cleats. All bells and pushes are placed in wooden boxes. Though exposed to much acid water, lead-covered wires are not affected. When conveyed in iron pipes, trouble arises from internally condensed moisture, and from grounding of wires in the pipes.

Some Transvaal mines discarded pipes, and installed bare copper wire on double-petticoat insulators. Such wires should be at least 7/18 S.W.G., and should be shackled off at distances of 100 ft., a bight being made in the wire to run off moisture. Below a bight in the shaft, the wire continues not directly underneath the length above, but to one side. Armoured cables should be of pure rubber, insulated with a leaden covering to the conductor, and a steel-wire sheathing. They can be bent round corners without injury to any part. Clements thinks armoured cable best for signalling purposes. He considers that one quadruplex cable is fully as good as a pair of duplex cables, one for the pole wires, and one for the two middle wires. Steel-sheathed cable may carry its own weight in vertical lengths of 300–400 ft., connections between lengths being in watertight boxes.

At pump-stations or landing-stages, cable should be boxed in for 10–12 ft. above landings; and on inclines, cable ought to be removed from risk of having lamps, tools, etc., hung on it, and be generally protected against injury.

There is no entirely satisfactory signal-bell. Pushers, in general, do not give satisfaction. The galvanometer key is better. Pullers depend on springs, which are apt to fail by rust. For batteries, the secondary cell is preferable for cleanliness and compactness, though the Leclanché primary is satisfactory if properly attended; whatever the form, there should always be an auxiliary. The E.M.F. should range from 6 to 16 volts, averaging about 10.

Bells and pulls must be inspected weekly at least, and all contacts be kept bright.

A system in use at Elkton, Colo. (W. B. Wilson, E. and M. Jl.), permits equally effective ringing from the cage in motion as from a station. Wires are conducted down the shaft, one on each side of guide next to pump-way, far enough apart to give ample clearance for safety cams on cage, and securely fastened on every third set (15 ft.) by hollow earthenware insulators, filled with lead, which hold a brass clip for the wire. A hole is bored in the dividing timber, about 3 in. deep, into which the insulator is driven. Wires, which are soft-iron telegraph wire, No. 8 gauge, are securely fastened to strong insulators at shaft top, and tightened from bottom by long turn-buckles; they are connected to electric-light wires in engine-room through a step-down transformer of piano-wire, which reduces voltage from 110 to about 20. A double-switch in engine-room permits instant change from dynamo to 40-cell (Samson) battery, in case of stoppage of dynamo for oiling or repairs. Signalling is done by circuit-closers at each station, as well as from one on cage. Station circuit-closers are very simple; a small rod of $\frac{5}{8}$-in. round steel is run through a hole in the station post, which brings it just behind the guide, and about 2 in. above the divider, where it is secured in two small solid journal-boxes. An insulated handle is put on the end at station, and two curved brass circuit-closers are fastened on the rod in such manner that a half-turn brings them in contact with the wires, a counter-balance on the rod immediately returning them to original position on releasing handle. The circuit-closer on cage is a little more complicated, but on same principle. It is fastened to corner braces of cage and outside the shoes. The arms work on a ball point, ensuring greater certainty of both points being brought in contact with the wires while the cage is in rapid motion. As auxiliaries, in case of breakdown, are a bell-rope and flash-light signals from each station, the latter being used entirely to let the cage-tender know at what level he is next wanted.

In a very complete system largely adopted in Austrian collieries (Dekanovsky, Coll. Guar.), half the path of the current is formed by the winding-rope, and the other by a copper or steel conducting-cable leading from deepest level to winding-pulley. Connection between this second conductor and the induction-apparatus, mounted in cage, is effected by a sliding contact which can be disconnected. The signalling appliance is in duplicate, one for each cage, so that when cages are conveying shifts to or from work, independent signals can be sent from each to the engineer. A copper or steel conducting-wire *a* (Fig. 217) 4 mm. diam., extends from top to bottom of shaft, and is affixed to efficient insulators both at winding-pulley and at the sole. Below the upper point of attachment, is an appliance for tightening up the conductor from time to time. Above-bank, the conductor is connected by insulated copper with one pole of induction-bell in engine-house. Connection of con-

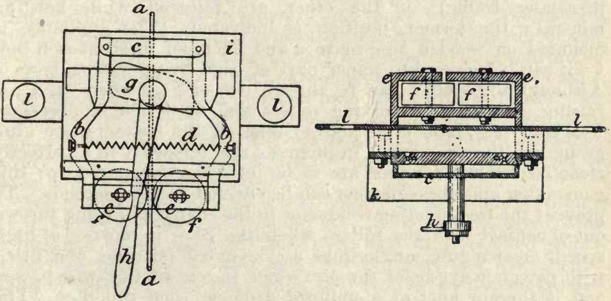


FIG. 217.—ELECTRIC BELL.

ductor with induction-generator screwed on cage is effected by sliding contact, properly insulated from ironwork of cage, and connected with one pole of induction-apparatus. The point of attachment of induction-generator to cage forms at same time connection between second pole and winding-rope (return current); and, since winding-rope is in continual contact with winding-pulley, a particular point on the pulley-bearing, or a point on the headgear when this is of iron, is placed in connection with the second pole of bell by means of insulated copper wire, thus completing circuit; so that, when a current is set up by turning the generator, the signalling-apparatus above-bank will sound the signal determined upon. Thus the entire appliance consists of three principal parts—conductor, contact charged with taking up current and transmitting it to bell, and induction-generator with suitable bells. In considering relative usefulness of copper and steel conducting-cables, the following points arise: Copper or bronze possesses high conductive properties (very important in telephonic communication), and wire can be readily repaired by brazing when broken, even if rupture occur in many places (of great utility when a deeper level is opened up, the new length of conductor being simply brazed to end of old cable), and copper is much more easily cleaned than steel, the latter being very liable to rust, even when coated with zinc, this being soon rubbed off by friction of contact-piece. On the other hand, steel wire is much stronger than copper, and stands wear better. The second part of the signalling apparatus is the sliding contact, which consists of two lever arms *b*, connected at *c*, and pressed together by spring *d*; ends of arms have bearing-bows *e* for carrying contact-rollers *f* between which conducting-wire *a* passes. Eccentric *g*, provided with adjusting lever *h*, and mounted between limbs *b*, can be retained in either of two extreme positions —in one, contact-rollers *f* are pressed against conductor *a* (for passenger traffic); in the other, are released while hoisting mineral: the former position is indicated. The appliance is mounted on wooden base-plate *i*, and enclosed in sheet-iron case *k*; it can be screwed to upper part of cage by projecting bows *l*. Although the contact can be thrown out of gear when cages are raising coal, thus preserving rollers and wire from unnecessary wear and tear, it is preferable to take off the contact-piece altogether while such work is in progress, and replace it in a perfectly clean condition when men are to be let down or brought up; this connection and disconnection can be effected in a few seconds. To prevent the free-hanging conductor in the shaft from being thrown out of contact with the rollers when the cage is moving at high speed, or swinging, angle-irons are mounted on cage, and fitted with porcelain eye-holes, through which the conducting-wire passes, and is thereby kept at a uniform distance from the cage. The third portion of the apparatus consists of generator and signalling appliance. The former is an ordinary alternating-current induction-machine, of same type as used in telephone apparatus, except that it is enclosed in a wooden case impregnated with paraffin, and carries a check-bell, which, when the apparatus is in proper working order, acts concurrently with a signalling bell above-bank. In addition, the case is provided with a small signal-plate, so that the cage-man runs no chance of making a mistake as to his signals. The signalling bells, which are of different tones, are mounted in the engine-house, and marked in accordance with the cages they represent, so that the engineer has no difficulty in recognising whence signals come, and cannot fall into error even if both sound together.

Electric signalling bells have now become very popular, and in some cases as many as 20 primary cells are used on a circuit. To prevent danger from sparking at contacts, Home Office rules limit pressure on signalling circuits to 15 volts in fiery mines. Bells and keys are generally fitted so as to be dust- and damp-tight. On roads where mechanical haulage is employed, two bare wires are stretched overhead at such distance that they can be pinched together by the hand. These wires are so connected to a bell circuit that signals can be transmitted from any point on the roadway to the engineman. Cost of primary batteries in a large colliery amounts to 80*l.* a year for material alone.

Complete sets of bells and station indicators are used on the Rand, bells and pushes being designed so that it is impossible for water to get at vital parts. Indicators are actuated through a rheostat to a voltmeter, the pointer of which shows level or signal. In case of drop or increase in voltage due to the cells, attention of the hauling-engine driver is called by a bell ringing until current is adjusted. The driver, by moving a handle at surface, indicates his reading of the signal to all levels where instruments are fixed.

Telephone apparatus can be substituted for an induction generator on cages, and thus enable verbal communications to be made from cage to bank, in addition to ordinary signals—an advantage of considerable importance in recovery and repair work. Telephones for this purpose are properly protected from moisture by rubber and asbestos. The apparatus acts reliably, and requires no maintenance beyond keeping clean. About once a week, conducting-wires must be tightened up and freed from dirt by rubbing with a cleaning-rag or worn-out emery cloth; contact-pieces must be properly insulated from cage by rubber plates, and not greased. Then wear-and-tear is very slight. Cost complete (excepting telephones) for a pair of cages does not exceed 20–25*l.*

Telephones underground are not common, but where workings are very extensive, they are most useful. The author employed them at Lucknow for communication between foremen from fixed points at extreme ends of the mine, and between mine and office, wires being so arranged that conversations between foremen could be also heard in the office and in the mine-manager's bedroom.

FELLOW AID IN MINING ACCIDENTS.

THE following excellent instructions from the pen of Dr. G. W. King, appeared in E. & M. Jl. All miners should know how to act promptly in sudden emergencies, and to give proper assistance to those requiring help. At the receiving hospital, cases are too often seen where life has been imperilled during transit: simple

FIG. 218.—FRACTURED LEG.

fractures have become compound by the unskilful manner in which they have been jostled about; others in a more critical condition may receive such additional injury that recovery is rendered doubtful if not impossible. The immediate care of the injured, and attentions to be given or withheld before a doctor's services can be obtained, is a form of knowledge that should be acquired by all workmen whose occupation exposes them to many dangers that can neither be foreseen nor guarded against.

The greater number of accidents in mining occurs in places difficult of access to all but those familiar with underground work:

FIG. 219.—CRUDE SPLINT.

hence the need of ready action guided by sufficient knowledge to extemporise and use whatever appliances happen to be at hand, if the case is one that admits of such manipulation. Herein is where ability to judge correctly is of great value. That attempts will be

FIG. 220.—FRACTURED THIGH BONE.

made by those present to handle the wounded is almost a certainty. How it can be safely done is the problem. Ordinarily the first efforts of anxious and excited companions is to force the sufferer into an upright position, forgetting, in their zeal to do something, that his injuries may be such that this would be attended with great risk. The surgeon quietly endeavours first to find out the extent of injuries with the least disturbance of the parts implicated; he is then in a position to give directions for safe removal.

To know what to avoid doing is the first lesson to be learned, and generally the rules to be observed are simple.

FIG. 221.
FRACTURED FORE-ARM.

FIG. 222.
FRACTURED UPPER ARM.

Supposing a case of syncope (fainting) from loss of blood, no position but the horizontal one should be attempted, unless it be in extreme cases, where death is imminent from anæmia (deficient supply of blood in the brain). In the latter condition, lowering the head will assist by gravity in retaining blood enough in the heart and brain to sustain life until skilled assistance arrives. The danger of suddenly raising one thus weakened to an upright position is very great, and would prove fatal. If removal is imperative, and there is no litter available, a common board or

FIG. 223.—LIFTING PATIENT.

FIG. 224.—CARRYING PATIENT.

several pieces of lagging may be utilised. When lagging is chosen for the construction of the litter, the pieces may be fastened together by a rope or cross strips of wood nailed firmly at proper intervals. A litter well adapted for use in mines was described in E. & M. Jl., Nov. 14, 1891. When once the patient is placed upon it, and the fastenings are properly adjusted, there is no danger of subsequent injury, whatever position it may be necessary to assume. Where narrow and tortuous drifts are to be passed through, and ladderways traversed in finding egress to a station, its efficiency becomes apparent.

In transferring the wounded to a litter, the first care should be to protect the injured part during removal. The ways of accomplishing this can be best illustrated by reference to injuries commonly met with.

FIG. 225.—CARRYING PATIENT.

When there is a fracture of the leg below the knee, both bones being broken and more or less violence done to the surrounding tissues, the pain is usually intense; muscles and nerves are bruised and lacerated by sharp-pointed fragments of broken bones, so that the slightest movement of the part without proper support causes torture that is unequalled by more serious injuries of other structures. Rough handling should be avoided. Grasp the limb lightly but firmly, one hand above the other below the point of fracture, making at same time extension and counter-extension (pulling apart); then flex the limb upon the thigh to lessen muscular tension, holding it in this position, and being careful to keep it in line with the body (Fig. 218). As soon as a place of safety is reached, an emergency dressing should be applied. Some form of splint is required: a strip of board 4 in. wide and of proper length; a piece of lagging; or in the absence of either, 2 pick-handles tied together (Fig. 219) make a reliable support; 2 drills may be substituted, if necessary.

When the thigh-bone is broken much more difficulty is experienced in transport. As a rule, the less handling of the part the

FIG. 226.—ASSISTED WALKING.

better. A long splint extending from the axilla (arm-pit) should be applied upon the outer side of the limb; a long-handled spade or shovel can be made to serve (Fig. 220). The litter should then be brought alongside the patient, and slid underneath him while the parts are well supported.

Fractures of upper extremities are more easily managed, as they seldom interfere with walking or otherwise helping oneself. Some support will be required, as in fracture of both bones of forearm (between elbow and wrist), when a piece of board or lagging may be bound to the underside of the injured part, while the palm of the hand is held upward (Fig. 221). A sling can be extemporised from a portion of the wearer's garments turned up and fastened to the breast. When the fracture is between shoulder and elbow, fasten the splint to the outer side of the arm; then by a turn around the body, the elbow is bound closely to the side. The hand should rest in a sling (Fig. 222).

FIG. 227.—MOUNTING LADDER.

Many other injuries are so disabling that assistance is required to effect removal from scene of accident. The ease with which this can be accomplished depends upon whether sufficient bearers can be obtained. Often but a single one is within call, and the task is then one of extreme difficulty, but not impossible in the majority of cases, if the proper steps to be observed are known. When 3 are present, they can carry the disabled very comfortably by adopting the following method: let two slide their arms underneath the patient's body, so as to include hips and shoulders, grasping each other's hands firmly, while the third supports the limbs. If they are careful to all rise from the kneeling position at the same moment, but little shock will be felt (Figs. 223, 224). If bearers are limited to two, they should pass their arms in the same manner, spreading them sufficiently to include the thighs; in this way, they will be able to support the weight evenly, and with least discomfort to themselves. When the extremities are uninjured, shoulders and limbs may be grasped (Fig. 225).

Degree of disability is measured by severity and extent of injuries received, ranging from those who are able to walk with assistance to those who are entirely helpless and insensible. Assistance in walking is given in the manner indicated by Fig.

FIG. 228.—ARRESTING HÆMORRHAGE.

226. If the patient is able to sustain himself by the arms, he can be carried upon the bearer's back, and if necessary up a ladderway (Fig. 227).

Arrest of hæmorrhage (bleeding) is a subject of universal interest. Familiarity with most effective methods requires but little application.

Hæmorrhage from wounds of extremities can usually be arrested by pressure aided by position. When a large vessel is wounded, and the open end of it is in view, pressure with the finger introduced into the wound and placed directly upon the bleeding point is the readiest and most certain way of stopping flow of blood. This method is not to be recommended, however, on account of the danger of poisoning the wound. It is permissible only in cases when it is impossible to employ other means in time to save life. Encircling the part between the wound and body with a handkerchief or a piece of rope, and tying it tight enough to control circulation, is the proper course. The "Spanish windlass," Fig. 228, is applied by means of a loop or rope twisted tightly by means of a stick. Undue force should not be used, for fear of producing strangulation of the limb. Wounds of internal organs

FIG. 229.—ARRESTING HÆMORRHAGE.

accompanied by excessive hæmorrhage are best treated by perfect quiet and horizontal position. Flexion and elevation will frequently be effective when an artery in the foot is wounded. The same method applies in wounds of the palm of the hand; a small pad placed in the flexures of the joints renders the pressure more reliable (Figs. 229, 230).

Handling of wounds should above all things be avoided. Packing them with an ordinary handkerchief or other material that happens to be at hand is dangerous, as septic infection (blood poisoning) may thus be introduced which no subsequent skill and care can cure. A moderate amount of bleeding is less to be feared than a poisoned wound.

FIG. 230.—ARRESTING HÆMORRHAGE.

FIG. 231.—HEAD WOUNDS.

Scalp wounds bleed freely, and prompt action is required in injuries of this region. Fortunately the conditions are favourable for the use of pressure, which may be applied by tying a band around the head, without special choice as to material used. A suspender makes an admirable tourniquet, Fig. 231.

Among the dangers to which miners are exposed is that of gases generated by explosives. It is very insidious: a feeling of drowsiness steals upon the individual, and before he is aware of the necessity for retreat, his limbs fail and he falls helpless, to die if assistance is not at hand to immediately remove him and apply the proper restorative. Removal should be accomplished quickly, without forcing him into a position that would interfere with

FIG. 232.—ARTIFICIAL RESPIRATION.

breathing, for that function is liable to be already arrested by inhalation of poisonous gas. If consciousness is not entirely lost, dashing cold water upon the face and chest will frequently excite a renewed effort at respiration. Should the asphyxia or suffocation be extreme, and breathing cease, air must be at once forced into the lungs by mechanical assistance, known as artificial respiration. This is done by placing the patient upon his back in as comfortable a position as possible, loosening the clothing about neck and chest. First open the mouth; if the tongue has fallen backward sufficiently to obstruct the entrance of air into the lungs, it should be pulled forward and brought out at the angle of the mouth. The arms should then be grasped at the elbows, drawn above the head, and kept there for 2 or 3 sec. (Fig. 232). The manœuvre is then reversed for the same length of time, the arms being pressed against the chest (Fig. 233). This to and fro movement is repeated 15–16 times per min., until the patient breathes naturally. Of all means available in treatment of asphyxia, artificial respiration takes precedence. Without admission of air into the lungs, life cannot be

FIG. 233.—ARTIFICIAL RESPIRATION.

sustained. It is useless to waste time in resorting to measures that do not assist in meeting this positive indication. It must be remembered that violent efforts are not calculated to restore a life that is nearly extinct; on the contrary, it may destroy every hope of success, and hence the utmost gentleness should be observed in all that is done.

MINERALS, ORES,

In the following tables are embraced all ordinary minerals, whether metalliferous or not, especially those possessing any economic importance. They are arranged under the heads of their principal or most valuable element; this facilitates comparison between members of the same group, and introduction of commercial information. The crystallographic classification is that of Groth.

Aluminium.	Colour.	Streak.	Hardness.	Sp. Gr.
Alumstone (alunite, aluminite)	white, greyish or reddish	white	3·5–4	2·58–2·75
Bauxite	white, grey, yellow, brown, red	..	..	2·55
Bole	brownish yellow or red	shining	1–2	2–2·5
Corundum	colourless, white, reddish, bluish	..	9	3·9–4·16
Cryolite	brownish or blackish	white, brittle	2·5	3
Emerald	green	..	9	3·9–4·16
Emery	blackish	..	9	3·9–4·16
Fullers' earth ..	greenish, bluish, brownish	..	..	..
Kaolin	dirty-white	..	1–2·5	2·4–2·6
Kyanite	blue-white	white	5–7	3·6–3·7
Labradorite ..	grey, brown, greenish, reddish	white	6	2·67–2·76
Lithomarge ..	white, yellow, grey, red	..	2–2·5	2·3–2·6

AND METALS.

It must be borne in mind that the characteristics quoted against each mineral are those observed in the pure fresh state. Effects of weathering and other corrosive agencies all tend to obliterate these marked features, and, in many instances, the chemical composition is thereby materially changed. Some minerals are so prone to decomposition that definite formulæ cannot be given them.

Chemical Composition.	Structure.	Remarks.
indefinite: alumina sulphate and silica	trigonal, 21	Roman alum
$Al_2O_3.2H_2O$	amorphous, earthy	a limonite in which the iron has been replaced by aluminium; 40% Al
indefinite: hydrous silicate of alumina	amorphous, earthy	falls to pieces in water with crackling noise
Al_2O_3	trigonal, 21	..
$Al_2F_3.3NaF$	monoclinic, 5	23% Al
Al_2O_3	trigonal, 21	gemstone
Al_2O_3	trigonal, 21	contains $\frac{1}{4}$ to $\frac{1}{3}$ iron oxide as an impurity
indefinite: hydrous silicate of alumina	soft, earthy	feels soapy, adheres to the tongue
indefinite: hydrous silicate of alumina	clay-like	china-clay
$Al_2O_3.SiO_2$	triclinic, 2	..
$Al_2O_3.CaO.3SiO_2$	triclinic, 2	..
indefinite: hydrous silicate of alumina	clay-like	feels greasy, adheres to the tongue

Aluminium (*contd.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Ruby	red	..	9	3·9–4·16
Sapphire	blue	..	9	3·9–4·16
Topaz	yellow, pinkish, bluish	colourless	8	3·4–3·6
Turquoise	blue, bluish, green	white	6	2·6–2·8
Wavellite	white, yellow, brown	..	3·5–4	2·33
Websterite	dull, earthy, white	..	1–2	1·66

Alumstone is mined on a small scale, calcined and lixiviated as a source of alum, containing 17½–40% alumina; lime carbonate and iron oxide reduce its value very much. Alum shale (11–19%) is more important, occurring in beds 200–300 ft. thick, and mined to the extent of 4000–6000 t. per ann. in England, and 500–3000 t. in Germany. It is submitted to very slow pile roasting. Value about 16*s.* per ton. *Bauxite* and *Cryolite* are the present main sources of aluminium. Abundance of bauxite exists in Ireland, France, and the United States. France produces about 100,000 t. a year; America, 50,000 t.; England, 10,000 t. The Arkansas beds are 40 ft. thick, in Tertiary sandstone; the Alabama deposits are 60 ft. thick, in Silurian. Cryolite is quarried in Greenland (10,000 t. per ann.). Minerals worked as a source of the metal should be as free as possible from iron, and as rich as possible in alumina. Cryolite, consisting when pure of 13% aluminium combined with 54 of fluorine and 33 of sodium, is always more or less contaminated with iron and silicon compounds, which give much trouble: when total impurities reach 20%, the mineral is commercially worthless. Bauxite, affording about half its weight of alumina, is liable to the same faults. Average samples will show 55–58% of alumina, 3–20 silica, 2–25 iron oxide, 11–30 water; the maximum figures of silica and iron lower the value greatly. The same may be said of kaolin as a source of the metal. Bauxite fetches about 16*s.* a ton as mined; cryolite, about 70–80*s.* Bauxite has lately come into use for making refractory bricks: it is washed to remove free silica, calcined at 2500° F. min., and bonded with 4% fireclay. *Corundum* and *emery* are alike used for abrasive purposes: corundum is much the harder, with sharper edges, and cuts more deeply and rapidly, but it is more brittle and therefore less durable; it is worth about 20*l.* a ton, emery only 2–4*l.* Corundum occurs

Chemical Composition.	Structure.	Remarks.
Al_2O_3	trigonal, 21	gemstone
Al_2O_3	trigonal, 21	gemstone
$5(Al_2O_3SiO_2) + Al_2F_6SiF_4$	rhombic, 8	gemstone
$Al_2O_3.P_2O_5 + 5H_2O$	..	gemstone
$3Al_2O_3.2P_2O_5 + 12H_2O$	rhombic, 8	yields phosphorus
$Al_2SO_6 + 9H_2O$	monoclinic, ii.	..

principally in Canada and the United States, notably in beds in altered olivine rock. Canada is now by far the largest producer. The ore is crushed, sieved, mulled, and washed repeatedly to remove impurities, dried by furnace or steam heat, and finally screened to various sizes; sometimes magnetic separation is also applied to remove iron, the eliminated portion having still some value for polishing. Emery is chiefly mined in Naxos (Greece) and Asia Minor, occurring in bands or lenses in metamorphic rocks, but following crystalline limestones. Mining in Naxos is generally done in a crude way by fire-setting for 24–30 hr. and quenching, only shallow depths being attainable; in Asia Minor, drilling and blasting are resorted to, but much of the product is highly micaceous, or full of magnetic iron, and is neglected, not bearing the cost of camel-transport. Yearly production is 4000–5000 t. corundum and 10,000 t. emery. *Garnet* often occurs as an impurity with corundum, lessening its value, though itself possessing much abrasive power. The American output is 3000–4000 t. per ann., value about 6*l*. a ton. Fine specimens are used as gems (carbuncle, almandine). *Fullers' earth* beds occur in Oolitic and Lower Greensand formations, at shallow depths, and are usually worked by stripping and quarrying. The impure mineral is crushed, levigated and dried in long shallow troughs (" maggies "). The United States produce 15,000–20,000 t. a year and import 10,000–20,000 t. more, for use in refining lard and cotton-seed oil. Value, about 25–30*s*. a ton. *Aluminous gemstones* are numerous, and, when unfit for jewellery, possess value as abrasives. The principal occurrence of emeralds is in geodes in Cretaceous limestone in Colombia. Lapis lazuli is got by fire-setting in a bed of limestone in Badakshan. Rubies and sapphires are mined in Burma, Siam, and Ceylon, mostly by shallow pits in certain gravels, the gem-stratum being sieved and washed. The topaz is found in gold-bearing (Brazil and Urals) and tin-bearing (Tasmania) gravels derived from granites. Turquoise occurs abundantly in N.E. Persia and New Mexico. *Bole* and *lithomarge* are among the widely-distributed siennas or "metallic paints" employed as pigments they are dug in open pits, weathered, and sometimes calcined. Value, about 8*l*. a ton. *Kaolin* or China clay, of which

Ammonium.	Colour.	Streak.	Hardness.	Sp. Gr.
Mascagnine ..	yellowish grey	..	..	..
Sal ammoniac ..	white, yellow, grey	white	1·5–2	1·52
Antimony.				
Antimonite	lead-grey	grey	2	4·5–4·6
Antimony (native)	tin-white	white	3–3·5	6·6–6·7
Berthierite ..	grey	grey	3	3·5–4·4
Jamesonite ..	grey	..	2–3	5·5–5·8
Kermesite	red	red	1–1·5	4·5
Senarmontite ..	grey, brown, yellow	..	2·5–4	2·5–2·6
Valentinite ..	white, yellow, grey, red, brown	..	2·5–3	5·6

Ores of antimony (oxide and sulphide) are sold on a basis of 45% of metal, the market value per ton being of course liable to fluctuate in sympathy with current prices of metal. These fluctuations are apt to be sudden and considerable. Each unit above 45% is worth so much per ton extra, say 8–9*s*. per unit when the standard is worth 25*l*. a ton; while ore carrying less than 45% is subject to a discount at the same ratio, so that with a poor ore, a limit is soon reached, when the price becomes a negative quantity. Other conditions, such as size and impurities, further affect market value. Thus, in dressing ore, it is essential that it should not be reduced in size much below that of hazel nuts. Of impurities, especially in sulphide, lead is worst: 1% will often make an otherwise good ore unsaleable. Copper, arsenic, and zinc are also objectionable, and reduce the value. Silver adds value if in appreciable quantity. The usual basis of sale is the assay value of metal computed on dry ore. Nevertheless, smelters often prefer to base their offer for a some 12–16 million tons yearly are raised in Cornwall and Devon, is quarried by strong steel-pointed shovels, weathered, washed and filter-pressed, for use by potters; it is worth about 25*s.* a ton. *Phosphorus* is now prepared from wavellite at Holly Springs, Penn., several hundred tons of ore annually being used; elsewhere the mineral is too scanty.

Chemical Composition.	Structure.	Remarks.
$(NH_4)_2SO_4$ ..	mealy	pungent bitter taste
NH_4Cl	cubic	..
Sb_2S_3	rhombic, 8	stibnite; 71% Sb
Sb	rhombohedral, 21	often carries gold and silver
$Sb_2S_3.FeS$	..	57% Sb
$Sb_2S_3.2PbS$	rhombic, 8	31% Sb
$2Sb_2S_3.Sb_2O_3$	monoclinic, 5	75% Sb
Sb_2O_3	cubic, 32	carries gold, silver, and nickel; 83% Sb
Sb_2O_3	rhombic, 8	83% Sb

parcel on the result of actual smelting a sample of 2–3 cwt., which enables them to judge of the behaviour of the ore in the furnace, whether it yields its metal readily, and does not suffer much loss in the slag. Antimonite (stibnite) is the principal commercial ore; kermesite and senarmontite also contribute. Smelters usually pay nothing for precious metal contents. France and Algeria afford 10,000–12,000 t., Italy 2000–8000 t., Mexico 2000–10,000 t., and Australia 2000–3000 t. antimony ore yearly. Silurian shales in Portugal carry many auriferous antimony lodes, but are ineffective as producers; Bohemian, Hungarian, Turkish and Californian deposits are in granite; Styrian in dolomite; Australian and New Brunswick in Cambro-Silurian rocks; Borneo and Bolivia are also producers. Antimony is chiefly used for hardening alloys; the trade is in few hands, and the market is erratic. The Australian (Hillgrove, N.S.W.) and Hungarian ores are nearly always highly auriferous.

Arsenic.	Colour.	Streak.	Hardness.	Sp. Gr.
Arsenic (native)	tin-white to dark grey	tin-white	3·5	5·93
Arsenolite	white, yellow	as colour	1·5	3·7–3·72
Mispickel	tin-white, steel-grey	dark grey	5–5·6	6·3
Orpiment	lemon-yellow	yellow	1·5–2	3·4–3·5
Realgar	orange, red	as colour	1·5–2	3·4–3·5

Mispickel is the common source of arsenious acid or white arsenic, which has a value of about 12–22*l*. a ton; it also often carries very much greater value in such impurities or associates as gold, silver, copper, and tin. The bulk of the (lode) tin and copper ores,

Barium.				
Barytes (barite) ..	white tinted	white	2·5–3·5	4·3–4·7
Barytocalcite ..	white, grey, yellow	white	4	3·6
Witherite	white, grey, yellow	white	3–3·75	4·29–4·35

These heavy white minerals, much employed as pigments ("permanent white"), are valued in the crude state at about 24*s*. a ton; smaller quantities are consumed in sugar-refining, plate-glass manufacture, and water-softening. Derbyshire and Shropshire produce 25,000–35000 t. per ann., and Missouri, Tennessee,

Beryllium.				
Beryl	green, blue, yellow	white	7·5–8	2·63–2·75
Chrysoberyl ..	green	colourless	8·5	3·5–3·8

Classed as precious stones and known as emerald (green), aqua-

Bismuth.				
Bismuth (native)	silver white, faint red tinge	as colour	2–2·5	9·7–9·8
Bismuthine ..	lead grey, orange tarnish	..	2	6·4–6·5
Bismutite	white, grey, green, yellow	..	4–4·5	6·8–6·9
Bismuth ochre ..	grey, green, yellow	..	soft	4·3–4·7

Chemical Composition.	Structure.	Remarks.
As	rhombohedral, 21	often traces of antimony, gold, iron, and silver
As_2O_3	cubic, 32	white arsenic
$FeAs_2FeS_2$	rhombic, 8	arsenical pyrites
As_2S_3	rhombic, 5	in limited demand as pigments
AsS	monoclinic, 5	in limited demand as pigments

mined primarily for those metals, yield, on roasting, considerable arsenic, which is condensed in cooling chambers. Some 3000–7000 t. are produced annually in France, 2000–3000 t. in Germany, 1000–4000 t. in England, and about 1000 t. in Spain and Portugal.

$BaSO_4$	rhombic, 8	heavy spar
$BaCO_3 . CaCO_3$	monoclinic, 5	..
$BaCO_3$	rhombic, 8	..

Virginia and N. Carolina, 40,000–65,000 t. The mineral is found in limestones; it is calcined, ground, washed with weak sulphuric acid (to remove iron) and then with water, and levigated. It is best when free from quartz grains and iron stains.

$3BeSiO_2$ $Al_2O_3 . 3SiO_2$	hexagonal, 27	gemstone
$BeO . Al_2O_3$	rhombic, 8	gemstone

marine (blue), etc.

Bi	rhombohedral, 21	traces of arsenic, sulphur, and tellurium
Bi_2S_3	rhombic, 8	traces of copper and iron; 68% Bi
$Bi_2O_3 . CO_2$	indefinite	82% Bi
Bi_2O_3	indefinite	with hydrous iron and arsenic; 89% Bi

Bismuth is of common occurrence in gold and silver ores, and much impedes extraction of the precious metals. The production of bismuth is very small, and the trade is a monopoly, so that prices are arbitrary; its uses are in alloys, painting on glass and porce-

Boron.	Colour.	Streak.	Hardness.	Sp. Gr.
Axinite	brown, blue, grey	colourless	6·5–7	3·27–3·5
Boracite	colourless, white, grey, yellow, green	white	4·5	2·97
Borax	white, bluish, grey, greenish,	white	2–2·5	1·71–1·74
Pandermite ..	snow-white	..	3	2·26–2·48
Sassolite	white, grey, yellowish	white	1	1·48
Tourmaline ..	black, blue, brown, green, red	colourless	7–7·5	2·9–3·3
Ulexite	snow-white	..	1	1·65–1·8

All except tourmaline (which is sold as a spurious gem) are valued as sources of boracic acid; the consumption is large (as borax, for fluxing and in laundries), and the value is about 10*s.* per unit of anhydrous boracic acid. The boracic salts occur as efflorescences over large areas in Thibet and California, and are scraped up and lixiviated. Thibet produces 500–1000 t. yearly, mostly

Cadmium.				
Greenockite ..	citron-yellow	orange to red	3–3·5	4·8–4·9

Though rich, probably more of the metal is obtained commercially as an impurity in zinc ores. Very limited use in photo-

Calcium.				
Apatite	greenish, yellowish, brownish	white	4–5	2·9–3·2
Aragonite	white, grey, yellow	colourless	3·5–4	2·9–2·95

lain, and in medicine. During the past 10 years, America has imported 60–90 t. per ann., value 6–8*s.* per lb. Principal European sources are Austrian gold and silver mines—8000–10,000 t. of ore yearly, giving 1200–1400 lb. metal.

Chemical Composition.	Structure.	Remarks.
$B_2O_3.CaO.Al_2O_3$ SiO_2	triclinic, 2	not common
$Mg_6B_{16}O_{30}MgCl_2$	dimorphous, 31, iii.	stassfurtite; 60% boracic acid
$Na_2B_4O_7.10H_2O$	monoclinic	tincal; 36% boracic acid
$Ca_2B_6O_{11}.3H_2O$	massive, friable	priceite; 49 to 55% boracic acid
$B_2O_3.3H_2O$	triclinic, 2	boracic acid
$B_2O_3.AlFeMnMg$ SiO_2	trigonal, 20	spurious emerald and sapphire
$Na_2O.2CaO.5B_2O_3$ $14H_2O$	globular, reniform	boronatrocalcite; 49% boracic acid.

conveyed to market by pack-sheep. The Californian boraciferous "salines" or marshes now dominate the world's supplies, yielding 6000–46,000 t. yearly of boraciferous earth containing 5–35% B_2O_3; but Chili affords 12,000–18,000 t. yearly of a soda-lime borate containing 18–24% B_2O_3; Asia Minor, a quantity of lime borate (pandermite); and the Tuscan lagoons about 2500 t. of boracic acid.

CdS	hemimorphic, 20	77% Cd

graphy, pyrotechny, and painting.

$3Ca_3P_2O_8.$ $CaCl_2.CaF_2.$	hexagonal, 23	coprolite, phosphorite
$CaCO_3$	rhombic, 8	..

Calcium (*cont.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Calcite	white, tinted	grey-white	2·5–3	2·6–2·72
Diopside	green, grey, yellow	..	5–6	3·2–3·38
Epidote	brown, red, yellow, grey, green	colourless	6–7	3–3·5
Fluorspar	purplish, bluish, greenish, yellowish	white	4	3–3·25
Gypsum	white, greenish	white	1·5–2	2·33
Prehnite	greenish	colourless	6–6·5	2·8–2·9
Wollastonite ..	white, grey, yellow, brown	white	4·5–5	2·7–2·9

The *carbonates* are burned to produce quicklime (1 t. requires 1¾ t. limestone and 4–8 cwt. coal), employed as a source of carbonic acid for mineral waters, and the harder kinds are useful for fluxing. *Fluorspar* is a valued flux, and worth about 16–32*s.* a ton. It is coming into great demand for reducing aluminium from bauxite; and is much employed in "bonding" emery wheels, and to increase the lighting efficiency of carbon electrodes. America produces 20,000–40,000 t. yearly, England 4000–40,000 t., Germany 15,000–30,000 t., and France 2000–4000 t. *Gypsum* is largely consumed as a manure, for making cement and plaster, and in adulterations; its value is about 7*s.* to 9*s.* a ton. England raises over 200,000 t. per ann., America about 1,000,000 t., France nearly 2,000,000 t., and Canada 300,000–400,000 t. The numerous *phosphates* of lime have a widely extended use (1½–2 million tons

Carbon.				
Amber	yellow, brown	white	2–2·5	1·06–1·08
Bitumen	black	..	soft to 3	·8–1·23
Coal	black	..	·5–2·5	1–1·8
Diamond	colourless, white, red, yellow, blackish	cuts	10	3·52
Graphite	black, grey	..	1–2	2

Chemical Composition.	Structure.	Remarks.
$CaCO_3$	trigonal, 21	chalk, limestone, Iceland spar, satin spar, etc.
$(CaMg)O.SiO_2$	monoclinic, 5	augite, cocolite, diallage, sahlite, etc.
$4CaO.3Al_2O_3.6SiO_2.H_2O$	monoclinic, 5	
CaF_2	cubic, 32	blue-john
$CaSO_4.2H_2O$	monoclinic, 5	alabaster, anhydrite, selenite, etc.
$2CaO.Al_2O_3 3SiO_2.H_2O$	rhombic, 7	
$CaO.SiO_2$	monoclinic, 5	

yearly) for making artificial manures, and are valued at 4–5*d*. per unit of tribasic calcic phosphate, with deductions for iron and alumina oxides over 3%, and for fluorine. Florida affords over 1 million tons yearly, Tennessee ½ million, and Carolina ¼ million; Tunis and France each 400,000–500,000 t., and Belgium 200,000 t. English production of coprolites has almost ceased. The Tennessee deposits are said to still contain 40 million tons. The American product is easily won by open-cast mining in dry weather; it is dried (first by airing and then in kilns), at a cost of 1*s*. a ton, and is mined and hauled for 7*s*. Much of the Florida rock is dredged from the streams which intersect the phosphate country. *Phosphorus* has occasionally been extracted from phosphorite and apatite since electric furnaces have been improved.

COH	massive	succinite
CH_2	indefinite	asphalt, elaterite, etc.
C	indefinite	with oxygen, hydrogen, and nitrogen
C	cubic, 31 dimorphous, 21	bort, carbonado
C	hexagonal, 31 dimorphous, 21	black-lead, plumbago

Carbon (*cont.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Jet	black	..	..	..
Ozokerite	white, yellow, brown	greasy	wax-like	·85–·9
Petroleum	opalescent	..	..	..

Of *amber* some 150–200 t. are produced yearly; the choicest pieces are used for ornament, but the bulk is consumed in varnish-making; it is worth about 12*s.* 6*d.* per lb. all through; copal and kauri belong to the same class—mineralised resins. Bituminous rocks or *asphalt* are a large group, in which the basis is generally calcareous matter, the bitumen being an impregnation, and ranging from about 7 to 40%; values depend on this percentage, and are about 5–50*s.* a ton; a very pure bitumen found in Utah (called uintite or uintahite) readily brings 10*l.* a ton. European sources are France (200,000 t.), Germany (40,000–50,000 t.), and Italy (30,000–40,000 t.); Trinidad affords 100,000 t. yearly. California is also a large producer. All *coals*, from anthracite to lignite and peat, are more or less pure hydrocarbons; their value is primarily controlled by their heat-giving or gas-yielding constituents, but also by their ability to produce a hard coke (for smelting purposes), by absence of a tendency to "slack" or crumble under exposure and to undergo spontaneous combustion when piled or stowed, and by proportion of ash (arising from earthy impurities). *Diamonds* which, from imperfections of crystallisation or colour, are valueless as gems, such as the borts of S. Africa and the carbonados of Brazil, are esteemed for abrasive purposes; prices fluctuate from 3*l.* to 8*l.* per carat. The De Beers mines yearly treat about 5 million loads (of 16 cub. ft. or 1600 lb.) of "blue ground" from shafts 1500–2000 ft. deep, at a cost of 4*s.* 11*d.* for mining and 2*s.* 7*d.* for washing, the yield varying from ·26 to ·46 carat per load, the value from 35 to 70*s.* per carat, and the return from 10*s.* 6*d.* to 24*s.* 3*d.* per load. The weathered ground is puddled, jigged, and trommelled, and the diamonds are largely caught by inclined

	Colour.	Streak.	Hardness.	Sp. Gr.
Cerium.				
Cerite	red-grey	metallic	4–5·5	4·9–5
Monazite	see *Thorium*			
Chromium.				
Chromite	brown-black	brown	5·5	4·3–4·5

Chemical Composition.	Structure.	Remarks.
C	indefinite	wood-like grain, horizontal cleavage, not easily burned or fused
CH_2	rhombic?	mineral wax
CH_2	liquid	kerosene, rock-oil, naphtha

greasy tables and a modification of the Elmore oil concentration. *Graphite* is essentially carbon, with varying kinds and proportions of impurity; its commercial value is not in the least governed by its purity, but depends almost entirely on physical peculiarities, and ranges from about 9*l.* to 40*l.* a ton (for crucibles, lining steel furnaces, facing castings, lubricants, polishes, etc.) up to 5000*l.* a ton (for special pencil-making). Ceylon graphite, which forms the world's main supply, is graded into large lump, ordinary lump (price 18–20*l.* a ton), chip (15*l.*), and dust (12*l.*). Bavaria, Bohemia, Silesia, Styria, Spain, Portugal, and Siberia all contribute to European needs, especially the two first-named, and Japan exports to America. The United States produce annually about 2000 t. worth 13*l.* a ton (mostly in New York State), and import 15,000–25,000 t. *Jet* is employed for beads and trinkets; the "hard" quality is worth 4–21*s.* a lb.; the "soft" brings only 5–30*s.* per stone (14 lb.). About 1000 lb. per ann. is produced in England. *Ozokerite* is a very hard and pure hydrocarbon used for making superior candles; it is worth about 18–30*l.* a ton. Galicia affords 5000–10,000 t. per ann., and the United States (chiefly Utah) 25–100 t. The Galician deposits are in Miocene clay-slates and at shallow depths, mining methods being crude. Of the world's total output of *Petroleum* (24–28 million t. per ann.), the United Slates produce 11–17, Russia 6–11, the Dutch East Indies $\frac{3}{4}$–$1\frac{1}{4}$; smaller contributions ($\frac{1}{4}$–1) come from Galicia, Roumania, India, etc. The oil-shales of Scotland are not nearly utilised as they should be, but the $2\frac{1}{2}$ million tons raised yearly afford over 2,000,000*l.* worth of oil.

$2(CeLaDi)O.SiO_2.H_2O$	rhombic, 8	30% Ce
$FeCr_2O_4$	octahedral, 32	magnesia, iron and alumina always present

Chromite contains 68% chromium sesqui-oxide when pure, and the market rejects anything below 50%; it is therefore usual to dress it to about 52%, and thus allow for differences in sampling and assaying. Its ordinary impurities are silica, magnesia and alumina, the first-named being highly detrimental. It is used in steel alloys, and for lining copper furnaces, but the principal consumption is still in dyeing and tanning. It is always found in asso-

Cobalt.	Colour.	Streak.	Hardness.	Sp. Gr.
Asbolane	black	shining black	earthy	3·18–3·29
Cobaltine	silver-white, reddish	greyish black	5·5–6	6–7
Erythrine	crimson, peach-red, greyish, greenish	as colour	1·5–2·5	2·95–3
Smaltine	tin-white to steel-grey	greyish black	5·5–6	6·4–7·2

The above ores are comparatively rich in cobalt, but quite as important supplies are also obtained from much poorer sources, notably a cobaltiferous wad (decomposed manganese) in New Caledonia (3000–9000 t. yearly of 3–5% ore); and a similar mineral worked in Wales, containing only 1% cobalt (with ½–1% nickel),

Copper.				
Atacamite	deep to blackish green	apple-green	3–3·5	4–4·3
Azurite	blue	light blue	3·5–4·25	3·5–3·8
Bornite	red, blue, brown	grey-black	3	4·4–5·5
Chalcocite	black-grey	as colour	2·5–3	5·5–5·8
Chalcopyrite ..	brass-yellow, iridescent tarnish	greenish-black	3·5–4	4·1–4·3
Chrysocolla ..	blue-green	white to bluish	2–4	2–2·3
Covellite	blue	lead-grey	1·5–2	3·8–3·9
Cuprite	red shades, blue	brown-red	3·5–4	5·7–6·15
Domeykite	tin-white to steel-grey	..	3–3·5	7·2–7·75
Enargite	blackish	as colour	3	4·43–4·45
Libethenite ..	green	..	4	3·6–3·8
Malachite	green	paler green	3·5–4	3·7–4

ciation with serpentine. The world's chief supplies come from New Caledonia (40,000–50,000 t. per ann.) and mainly from one group of detrital deposits easily worked, and giving ore of 55% just as raised, selling locally at 35–40*s.* per ton. Russia affords 7000–20,000 t., Greece 500–15,000 t., Canada 1000–10,000 t. (valued at 45*s.*), New South Wales 50–5000 t., and California about 100 t. (locally bought at 75*s.*).

Chemical Composition.	Structure.	Remarks.
$(CoO.CuO)2MnO_2.4H_2O$	indefinite	earthy cobalt; 14% Co
$CoS_2.CoAs_2$	cubic, 28	cobalt glance; 35% Co
$3CoO.As_2O_5.8H_2O$	monoclinic, 5	cobalt bloom; 29% Co
$(CoFeNi)As_2$	octahedral, 28	tin-white cobalt; 18% Co

brings 6*l.* a ton. Buyers insist on 3% min., and allow an advance for each ·1% additional, so that the ore is often dressed up to 4–5%, and is worth about 5*l.* per ton. Almost all the cobalt produced is converted into black oxide, and used as a pigment (9*s.* to 12*s.* a lb.).

$3CuO.CuCl_2.3H_2O$	rhombic, 8	59% Cu
$2CuCO_3.CuH_2O_2$	monoclinic	chessylite; 55% Cu
$3Cu_2S.Fe_2S_3$	cubic, 32	peacock ore, erubescite; 61% Cu
Cu_2S	orthorhombic, 13	copper glance; 79% Cu
$Cu_2S.Fe_2S_3$	tetragonal, 11	copper pyrites; 34% Cu
$CuO.SiO_2.2H_2O$	botryoidal, 17	36% Cu
CuS	..	indigo copper; 66% Cu
Cu_2O	octahedral, 29	tile ore, red copper ore; 71% Cu
$CuAs$	..	[88% Cu
Cu_3AsS_4	..	48% Cu
$5CuO.P_2O_5.H_2O$	..	53% Cu
$2CuO.CO_2.H_2O$	monoclinic, 5	57% Cu

Copper (*cont.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Melaconite	grey, brown	..	3	6·25
Olivenite	green-brown, yellow	..	3	4·1–4·4
Stromeyerite ..	see *Silver*	..	..	..
Tennantite	grey, brown	..	3·5–4	4·3–4·5
Tetrahedrite ..	steel-grey to iron-black	as colour	3–4·5	4·5–5·1

The above are only a few of the many minerals which carry copper: there are approximately one hundred which afford the metal commercially, and a very large number besides, which are not utilised. While the rich ores are valuable in themselves they afford but a small proportion of the world's output of copper, the bulk of the metal being obtained from rock masses with cupreous impregnations ranging from ½ to 5%, and rarely exceeding 3–3½%. Though a large proportion of the copper and copper ores now brought to market are sold by assay, the antiquated and cumbersome system of "ticketing" still survives. "Ticketings," as held in Cornwall and Wales, are periodical auctions, at which buyers make bids (written on tickets) for such parcels of ore as they may have previously sampled and assayed. The value of the parcel is worked out as follows. The "standard" is the actual value per ton (21 cwt.) of the fine copper contained in the ore, and is made up of the price paid for it added to the "returning charges," or cost incurred in extracting it from the ore. Then the market value of the parcel of ore is the amount arrived at by reckoning the "settled produce," or fine copper yielded by it at standard, and deducting the returning charges. These latter vary. In Cornwall they are fixed at 55*s.* per ton of ore, whether rich or poor. In Swansea, they vary with the character of the ore, and consist of two items, one being a fixed rate of 12*s.* 2*d.* per ton of ore, and the other a charge of 3*s.* 9*d.* per unit of metal in the ore. The following examples calculated out will make the matter clearer:—

A. *Finding Standard*—(Cornwall).

328 tons ore gave 21 tons of fine copper, or about 6·4%.

		£	*s.*	*d.*
328 tons × 55*s.* returning charges ..	=	902	0	0
Ticket offer for parcel	=	820	0	0
21 tons of fine copper into		1722	0	0
Gives a standard of		£82	0	0 per t.

Chemical Composition.	Structure.	Remarks.
CuO	..	tenorite; 79% Cu
$Cu.AsO_4.CuOH$	rhombic, 8	44% Cu
..	..	31% Cu
$4Cu_2S.As_2S_3$	cubic	51% Cu
$4Cu_2S.Sb_2S_3$	cubic, 31	fahlerz; 36% Cu

B. *Finding "Value" of Parcel*—(Cornwall).

76 tons ore at 4·55%	=	3·45 tons fine copper.
Multiplied by		82*l.* per ton.
		£282 18 0
Less returning charges at 55*s.* on 76 tons	=	209 0 0
Value of parcel	=	£73 18 0

C. *Finding "Returning Charges"*—(Swansea).

Ore at 5·75%.

		£	*s.*	*d.*	
Fixed charge	=	0	12	2	per ton.
Sliding charge at 3*s.* 9*d.* per unit on 5·75%	=	1	1	6	„
Total returning charges	=	£1	13	8	„

The presence of ·25% arsenic, or of ·01 to ·1% antimony, bismuth, or sulphur, or of 1 to 2% lead, is injurious. For electrical purposes, lead renders it unsaleable; but for many purposes a small amount of arsenic does not affect the value.

The American Smelting and Refining Co.'s rates are as follows —copper, 5*s.* per unit dry up to 5%, 6*s.* up to 10%, 7*s.* over 10%, 4*s.* when lead is paid for; gold, 79*s.* per oz. if ·05–2 oz., 81*s.* if over 2 oz. per ton; silver, 95% of New York quotation if over 2 oz.; zinc, 10% limit, 2*s.* per unit penalty; treatment charges, 16*s.* when value is under 58*s.*, 20*s.* under 4*l.*, 23*s.* under 5*l.*, 25*s.* under 6*l.*, 27*s.* under 7*l.*, 29*s.* under 8*l.*, 31*s.* under 9*l.*, 33*s.* under 10*l.*, 37*s.* 6*d.* under 15*l.*, 41*s.* 6*d.* under 20*l.*, 46*s.* over 20*l.*; on concentrates, silica limit is 10%, penalty 5*d.* per unit; zinc limit, 5%, penalty 1*s.* 3*d.* per unit.

About 70% of the world's output of copper is produced in N. America. The following statistics of four representative Lake mines are interesting: tons of ore raised annually—1 million, 1 million, ¾ million, ¼ million; tons of copper produced—9500, 9500, 8000, 6000; cost of copper per lb.—5·34*d.*, 5·43*d.*, 6·68*d.*, 3·14*d.* The average cost of actual production by the chief American contributors is said to be 11·75–12 c. per lb., or about 50*l.* a ton. Many mines all over the world, with a large aggregate production, come into action when copper reaches 80*l.* a ton and upwards, but cannot live at 60–65*l.*

Gold.	Colour.	Streak.	Hardness.	Sp. Gr.
Electrum	white-yellow	..	2·5–3	13–15·5
Gold (native) ..	yellow	..	2·5–3	15·6–19·5
Maldonite	pinkish-white	..	..	..
Porpezite	pale	..	..	..
Tellurides	see *Tellurium*			

The richly auriferous ores, such as the tellurides, are very valuable, but exceedingly refractory in treatment. The bulk of the metal is obtained (in the "free" state) from quartz (chiefly) and other vein matters, or the gravels and sands produced by their attrition, or is extracted (as an accessory) by solution or smelting

Iridium.				
Iridium (native)	steel-white	..	6	22·38–22·6
Iridosmine	grey-white	..	6–7	19–21·12

Rare and valuable; extremely durable and unchangeable; trade

Iron.				
Chalybite	reddish or yellowish brown	white	3·5–4·5	3·7–3·9
Cleveland ore ..	greenish-blue or blackish	..	..	..
Copperas	greenish	colourless	2	1·8–1·9
Göthite	yellowish or reddish brown	ochre yellow	5–5·5	4–4·4
Hæmatite	red-black	red-brown	5·5–6·5	4·5–5·3
Leucopyrite ..	greyish	..	5–5·5	6·8–8·7
Lievrite	black	green-brown	5·5–6	3·7–4·2

The annual production of metallic copper is now 500,000–700,000 t., of which the United States yield 250,000–400,000 t. (mostly from less than 1% ore); Spain, 50,000 t.; Japan and Chili, each 25,000–30,000 t.; Australasia, 25,000–35,000 t.; Germany, 20,000 t.; Mexico, 15,000–70,000 t.; Brit. N. America, 10,000–20,000 t.; Russia, 8000–10,000 t.; S. Africa, 7000 t. In the last ten years prices have ranged from about 50*l.* to 100*l.* per ton.

Chemical Composition.	Structure.	Remarks.
AuAg	..	native alloy; 64% Au
Au	..	traces of silver, copper, lead, iron, antimony, bismuth, arsenic, etc.
Au + Sb	..	native alloy; 61% Au
Au + Pa	..	native alloy; 64% Au

from the various sulphides of the common metals, notably mispickel, pyrite, chalcopyrite, and galena. The world's gold output is 15–20 million oz. (S. Africa, 6; Australasia, 4; United States, 5; Russia, 1; Canada, $\frac{1}{2}$; Mexico, $\frac{3}{4}$; India, $\frac{1}{2}$).

Ir	..	traces of osmium, platinum and copper
IrOs	..	traces of rhodium, iron and palladium; 50% Ir

monopolised, principal application being for pointing pens.

$FeCO_3$	rhombohedral, 21	siderite, spathic irons; 48% Fe
indefinite: $FeCO_3$, clay, carbonates of Ca and Mg, and ·65–·74% P	imperfect oolitic	up to 50% Fe, generally 29–33%
$FeSO_4 . 7H_2O$	monoclinic, 5	melanterite
$Fe_2O_3 . H_2O$	rhombic, 8	
Fe_2O_3	reniform	specular iron; 70% Fe
Fe_2As_3	rhombic	
$Fe_6H_2Ca_2Si_4O_{18}$	rhombic	ilvaite, yenite

Iron (*cont.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Limonite	brown	yellow-brown	5–5·5	3·6-4
Magnetite	iron-black	black	5·5–6·5	4·9–5·2
Marcasite	grey-yellow	..	6–6·5	4·6–4·8
Pyrite	yellow	greenish black	6–6·5	4·8–5·2
Pyrrhotite	bronze-red	grey-black	3·5–4·5	4·4–4·65
Vivianite	bluish, greenish	bluish	1·5–2	2·58–2·68

The principal industrial ores of iron are chalybite, hæmatite, limonite, magnetite and pyrite. An impure form of chalybite—the Cleveland ironstone—is the present chief source of the English metal. All iron ores are apt to contain more or less silica, alumina, lime, magnesia, carbonic acid, and water; also sulphur, phosphorus, and titanium. Phosphorus is the most harmful impurity, and next ranks sulphur. An appreciable percentage of either will condemn an ore. Titanium, if in any quantity, renders the ore useless for the blast-furnace. Mechanical condition is also of importance, iron

Lead.				
Anglesite	blue, green, yellow, grey tints	colourless	2·7–3	6–6·39
Boulangerite ..	dark metallic grey	as colour	2·5–4	5–6
Bournonite	steel-grey to black	as colour	2·5–3	5·7–5·9
Cerussite	grey, blue, green, brown tints	colourless	3–3·5	6·4–6·48
Crocoite	red	orange-yellow	2·5–3	5·9–6·1
Galena	lead-grey	as colour	2·5–2·7	7·2–7·7
Mimetite	yellowish, brownish	whitish	3·5	7–7·25
Minium	red	orange-yellow	2–3	4·6
Pyromorphite ..	green, yellow, brown	yellowish	3·5–4	6·5–7·1

The chief commercial ores are anglesite, cerussite, galena, and pyromorphite. The annual output of the metal is $\frac{3}{4}$–1 million tons: United States, 200,000–300,000 t.; Spain, 200,000 t.; Germany, 140,000 t.; Mexico, 75,000–100,000 t.; Australasia, 70,000–140,000 t. Value in last 10 years, 10*l.* to 16*l.* per ton. Antimony, silver, and

Chemical Composition.	Structure.	Remarks.
$2Fe_2O_3.3H_2O$	radiating, fibrous	bog iron ore; 60% Fe
Fe_3O_4	cubic, 32	72% Fe
FeS_2	rhombic	iron pyrites, mundic; 46% Fe
FeS_2	cubic, 28	
Fe_7S_8	hexagonal, 20	magnetic pyrites
$3FeO.2P_2O_5.8H_2O$	monoclinic, 5	

sands and such ores as crumble readily commanding a lower price, because they can only be added in small quantities to other ores for smelting. Modern improvements in concentrating machinery, especially by the aid of electro-magnetism, are responsible for rendering many otherwise valueless ores quite useful to the smelter. The yearly product of metal is some 35–40 million tons of iron, and 25–30 million tons of steel; of this total the United States account for about 12 and 10 million tons respectively; Great Britain, 10 and 5; Germany, 8 and 6; France, 3 and 1½; Russia, 2½ and 1.

$PbSO_4$	rhombic, 8	lead vitriol; 67% Pb
$3PbS.Sb_2S_3$	rhombic, 8	58% Pb
$2PbS.Cu_2S.Sb_2S_3$	rhombic, 8	endellionite; 42% Pb
$PbCO_3$	rhombic, 8	77% Pb
$PbCrO_4$	monoclinic, 5	64% Pb
PbS	cubic, 32	86 % Pb
$3(Pb_3As_2O_8).PbCl_2$	hexagonal, 23	69% Pb
Pb_3O_4	pulverulent	red lead; 90% Pb
$3(Pb_3P_2O_8).PbCl_2$	hexagonal, 23	75% Pb

zinc are all detrimental. For pigment making, the presence of zinc does not so much matter, as the salt is white. For sheet-lead for chemical manufacturers' purposes, the impurities named must be absent. Zinc not only reduces the value of the lead produced, but it also causes loss of metal by volatilisation and formation of fume.

Antimony and arsenical pyrites are even more troublesome in this respect. Barytes interferes with smelting operations, and increases consumption of fuel. The ore is valued for its lead contents per ton of 21 cwt. dry. Many lead ores contain silver. This does not add to the value of the sample unless it exceeds 5 oz. per ton. Against the assay value of ore is deducted a "returning charge," which varies according to cost of fuel, rates of carriage, etc., and generally ranges between 2*l.* and 3*l.* a ton. Upward of 90% of the lead produced from home-mined ores in the United States is refined and sold by three interests, chief being the American Smelting and Refining Co. The market for pig lead is largely controlled by this company. Demands for silver-lead ores vary considerably, and, with them, charges made for treatment. Copper is paid for at current best selected unitage rates, less returning charges applicable to copper ores. Lead payment is based on Spanish quotations at 90% of fire assay for 30–55% ores, 92½% for 56-64%, and 95% for 65% and upwards; silver, at fine quotations, if under 100 oz. per ton, and at standard, if over; gold, at 98% of assay, at 82*s.* per oz.

Lithium.	Colour.	Streak.	Hardness.	Sp. Gr.
Lepidolite	violet-grey, yellowish	..	2·5–4	2·84–3
Petalite	rose-tinted white	colourless	6–6·5	2·4–3
Spodumene ..	greenish, greyish	colourless	6·5–7	3·1–3·2
Triphylline ..	greenish, bluish, brownish	greyish	5	3·5–3·6
Magnesium.				
Actinolite	dark-green	white	5·6	3–3·16
Ankerite	pinkish	white	5	3·6
Anthophyllite ..	grey-brown	paler	5·5	3·18–3·22
Asbestos	white to green	..	5	3·02–3·1
Bronzite	bronze-like	white	5–5·5	3·12–3·3
Brucite	greyish, greenish	white	2·5	2·35
Chlorite	green	..	1–1·5	2·6–2·8
Dolomite	grey, green, red, brown, black	..	3·5–4	2·8–2·9
Epsomite	white	white	2–2·5	1·7
Hornblende ..	green-black	paler	5–6	2·9–3·4
Hypersthene ..	green-brown	brown-grey	5–6	3·3–3·4

Smelting and refining charges are 45–55*s*. per ton. Penalties are: zinc, all over 10%, at 1*s*. per unit %; sulphur, all over 15%, at 6*d*. per unit; bismuth, over $\frac{1}{4}$% special contract. *Ex.*—Sample assaying 45% lead, 125 oz. silver, ·04 oz. gold, 9% zinc. Quotations: Spanish lead 11*l*. 10*s*. per long ton = 9*l*. 16*s*. 5*d*. per ton; silver, 26$\frac{3}{4}$*d*. per oz. fine, 24$\frac{3}{4}$*d*. per oz. standard. Then—

		£	*s.*	*d.*
Lead, 45% less 10% = 40·5% at 9*l*. 16*s*. 5*d*.	=	3	19	4
Silver, 125 oz. at 24$\frac{3}{4}$*d*. per oz.	=	12	18	11
Gold, ·04 oz.	=		nil	
Total		16	18	3
Less smelting charges, say		2	10	0
Net value per ton		£14	8	3

Canadian smelters penalise zinc above 8%.

Chemical Composition.	Structure.	Remarks.
$(LiKAl)Fl.SiO_2$	monoclinic, 5	4$\frac{1}{2}$% Li
$(Li_2Na_2Ca)O.Al_2O_3.SiO_2$	monoclinic, 5	5% Li
$3(Li_2Na_2Ca)O.4Al_2O_3.6SiO_2$	monoclinic, 5	3$\frac{1}{2}$% Li
$Li_3PO_4.Fe_3P_2O_8$	orthorhombic, 8	4% Li
$(MgCaFe)SiO_3$	monoclinic, 5	
$(MgCaFe)CO_3$	..	
$(MgFe)SiO_3$	rhombic, 8	
$(MgCa)SiO_3$	fibrous	amianthus
$MgSiO_3$	rhombic, 8	enstatite
MgH_2O_2	rhombohedral, 21	
$8MgO.Al_2O_3.5SiO_2.7H_2O$	monoclinic, 5	
$MgO.CaO.2CO_2$	rhombohedral, 17	magnesian limestone
$MgSO_4.7H_2O$	rhombic, 6	
$(MgCaFe)SiO_3$	monoclinic, 5	amphibole
$(MgFe)SiO_3$	rhombic, 8	

Magnesium (*cont.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Kieserite	white, grey, yellow	..	3	2·51–2·57
Magnesite	greyish, yellowish, brownish	white	3·5–4·5	2·9–3·1
Meerschaum ..	greyish, yellowish	scratched by finger-nail	2–2·5	·98–1·2
Olivine	greenish	colourless	6–7	3·3–3·5
Periclase	dark-green	..	6	3·67
Schiller-spar ..	green-brown	metallic, pearly	3·5–4	2·5–2·8
Serpentine	green, brown, yellow	white	2·5–5	2·5–2·8
Spinel	red, brown, black, blue, green	..	8	3·5–4·1
Talc	white, grey, green	greasy	1–1·5	2·5–2·8
Tremolite	dark-grey crystals	..	5–6·5	2·9–3·1
Wagnerite	green, yellow, brown	..	5–5·5	3

Of this large group, the only useful members are the silicates asbestos, meerschaum, and talc, the carbonates dolomite and magnesite, the sulphate kieserite, and the aluminous compounds embraced in the name spinel. *Asbestos* varies greatly in chemical composition and in physical structure, and the range of value is very wide; length and strength of fibre are the most important factors, while the main chemical constituents of best brands are approximately 40% silica, 41–43% magnesia, 1–2% each alumina and iron oxide, and 13–14% water. Chrysotile asbestos is in growing demand for making cloth and cordage, owing to its tensile strength. The amphibole variety, though occurring in much longer fibres (up to 2 ft.), and in larger masses (so that its mining is cheapened), cannot be used alone for spinning, but is limited to making board, moulded articles, and lagging for fireproofing and insulating. The average price of amphibole or true asbestos is 50*s*. a ton, while crude chrysotile sells for 20*l*. or over. Italy supplies most of the true asbestos, and Canada most of the amphibole; all the latter, having fibre ¾ in. long and upwards, is used for spinning. Canada produces 6000–9000 t. yearly; Italy, 3000–4000 t. Rock yielding 1% asbestos is generally profitable. *Meerschaum* is of but trifling commercial importance. *Talc* in its massive form, known as steatite or soapstone, is esteemed for furnace linings. hearthstones, sinks, etc., and, when ground is employed for paper-making, for foundry facings, as a lubricant, for dressing skins and

Chemical Composition.	Structure.	Remarks.
$MgSO_4.H_2O$	..	
$MgCO_3$	rhombohedral, 21	
$2MgO.3SiO_2.2H_2O$	..	sepiolite
$2MgO.SiO_2$	rhombic, 8	chrysolite
MgO	octahedral	
$3MgO.2SiO_2.2H_2O$	indefinite	decomposed bronzite
$3MgO.2SiO_2.2(3)H_2O$	doubtful	antigorite, chrysotile
$MgO.Al_2O_3$	cubic, 32	various gemstones
$3MgO.4SiO_2.H_2O$	rhombic (?)	steatite, soapstone, agalite, rensselaerite
$MgCaSiO_3$	blade-like	amphibole
$2MgPO_4.MgF_2$	monoclinic, 5	

leather, etc. By far the largest portion of the product of the United States is the fibrous mineral which is mined in New York, and used chiefly for paper filling, in connection with the wood pulp industry, to a less extent for the manufacture of wall plaster, cheap soap, waterproof paints, non-conducting pipe-covering and plaster, face and foot powders, and adulterated drugs. Value depends on colour, and the fibrous quality, so that physical and not chemical tests are used for grading. The better grades are white with a bluish tinge; the poorer have a yellowish cast. To the naked eye, product ground to 100 mesh appears an impalpable powder; but the microscope reveals varying fibrosity. The productive power of the New York deposits is almost unlimited, the annual output being 60,000–70,000 t., value about 30*s*. *Magnesite* has assumed considerable importance of late; Greece is the chief producer (about 20,000 t. yearly), Silesia, Styria, Hungary and California contributing; the mineral is calcined (the carbonic acid gas thus freed being utilised), and converted into fire-bricks, values being about 16*s*. a ton for raw mineral, 50*s*. a ton for calcined, and 30*l*. per 1000 for bricks. *Dolomite* and *kieserite* are used as manures and in chemical manufactures. *Spinel* in its various hues forms a group of semi-precious stones, among which are the spinel ruby (red), balas ruby (rose), almandine (violet), rubicelle (orange) and pleonaste (black).

Manganese.	Colour.	Streak.	Hardness.	Sp. Gr.
Alabandite	blackish grey	white	3·5–5	3·9–5
Braunite	brownish black	as colour	6–6·5	4·7–4·9
Hausmannite ..	brownish black	brown	5–5·5	4·7–4·9
Manganite	iron-black to grey	dark brown	4	4·2–4·4
Polyanite	grey	..	6·5–7	4·8–5
Psilomelane ..	grey, black	..	5–6	3·7–4·7
Pyrolusite	dark grey to iron-black	bluish black	2–2·5	4·8–5
Rhodochrosite ..	reddish, brownish, yellowish	white	3·5–4·5	3·4–3·7
Rhodonite	reddish, brownish	white	5·5–6·5	3·4–3·7
Wad	reddish, brownish	..	indefinite	3–4·2

To the 600,000–800,000 t. of manganese ores yearly produced (about ⅓ in the Caucasus) all the oxides and the carbonate contribute, most valued being rhodochrosite, pyrolusite, braunite, manganite, and psilomelane, while material supplies (for fluxing and steel-making) exist in the "clinker" left after extracting zinc from franklinite. There are two distinct markets for manganiferous ores, one being chemical and the other metallurgical. For the former, value depends upon the capacity of the ore to assist in generation of chlorine for manufacture of bleaching powder, and its quality is gauged by comparison with pure manganese dioxide (MnO_2). The market basis adopted for Spanish and similar ores is 70% MnO_2 in ore dried at 212° F.; which means that 100 parts by weight of ore liberate as much chlorine as 70 parts of pure MnO_2 would do. The price of such ore fluctuates generally around 4*l.* per ton, with an addition or deduction of 2*s.* 6*d.* per unit for higher or lower quality, the minimum being usually 65%. In the case of German ores, the normal strength is 60%, the minimum 57%, and the price per unit 2*s.* up or down. Of impurities, the most injurious are carbonates (of lime, etc.), as they not only consume hydrochloric acid, but also evolve carbonic acid, which has a most deleterious effect in bleaching-powder manufacture. Physical character of the ore is of some consequence, soft varieties being most easily soluble in acid, and therefore preferred. Some high-grade ores are so hard as to consume excess of acid and steam, which greatly lowers their market value. Much manganese ore is more properly speaking manganiferous iron ore, and is largely used for steel alloys. The standard grade is a minimum of 40% metallic manganese, but the demand for 50% is keener. The market for manganese ores is now practically controlled by the iron and steel industries, other consumption

Chemical Composition.	Structure.	Remarks.
MnS	cubic, 31	glance or blende; 63% Mn
Mn_2O_3	tetragonal, 15	69% Mn
Mn_3O_4	tetragonal, 11	72% Mn
$Mn_2O_3 . H_2O$	rhombic, 8	62% Mn
MnO_2	tetragonal, 15	63% Mn
indefinite	indefinite	MnO_2 with K, Ba, etc.
MnO_2	massive, reniform	63% Mn
$MnCO_3$	rhombohedral, 21	diallogite; sometimes contains lime; 48% Mn
$MnSiO_3$	triclinic (?), 2	42% Mn
indefinite	indefinite	mainly consists of MnO_2

having fallen into comparative unimportance, while employing also only the purest and richest ores. Large supplies of manganese ore are available in readily accessible places, those in the Caucasus alone, it is estimated, being capable of meeting for another century the world's demands, at the present rate of consumption—1,000,000 t. a year. Prices are based on manganese contents, with penalties for impurities. The United States basis is fixed by the Carnegie Steel Co.—ore must not contain more than ·1% phosphorus or 8% silica; for each 1% silica, 7½*d.* is deducted, and for each ·02% phosphorus, ½*d.* is deducted. The price per unit of manganese is 14*d.* per ton for ore with over 49% Mn, 13½*d.* for 46–49%, 13*d.* for 43–46%, and 12½*d.* for 40–43%. Russian (Caucasus) ore of ordinary grades, during 5 years to 1902, averaged about 48*s.* per ton; recently, however, prices of high grade ore have advanced strongly. The basis is 50% Mn, with P not to exceed ·17%, nor SiO_2 9%. Such ore sells at European ports for 8–14*d.* per unit of Mn; 2½–5*d.* is deducted for each unit of silica. On Turkish ore, the basis is 45% Mn, with limits of ·03 for phosphorus and 11 for silica. Japanese brown ore sells at Hamburg at from 50*s.* per ton for 65% MnO_2 ore (41% Mn) to 115*s.* for 87% MnO_2 ore (55% Mn). For German ore, the price is calculated on 50% MnO_2 at 20*s.* per ton, with an increase of 1*s.* per unit of dioxide above 50; but much of the Hessian ore is too pulverulent. French ore, calcined, with 35–40% Mn in 1904 brought 1*s.* 3*d.* per unit. Spanish and Brazilian ores carry only ·01–·02% P, as against Caucasian with ·15–·17%. Indian ore is also low in P. Principal yearly outputs in tons of metal are: Russia, 400,000–700,000; Brazil, 150,000–200,000; India, 100,000–200,000; Spain 25,000–100,000; Germany, 50,000; Turkey, 50,000; France, 25,000; Chili, 20,000; Cuba, 20,000; Japan, 15,000.

Mercury.	Colour.	Streak.	Hardness.	Sp. Gr.
Amalgam (native)	silver-white	as colour	3–3·5	10·5–14·1
Calomel	white, grey, brown	white to pale yellow	1–2	6·5
Cinnabar	red, grey, brown	scarlet	2–2·5	8·99–9

Practically cinnabar is the only source of the metal, though a black sulphide (metacinnabarite) and a sulpho-selenide (guadalcazarite) are encountered in California and Mexico. Cinnabar occurs chiefly as an impregnation of sandstones and limestones,

Molybdenum.				
Molybdenite ..	opaque lead-grey	as colour	1–1·5	4·4–4·8
Molybdite	straw-yellow	..	1–2	4·5
Wulfenite	greyish, yellowish, brownish	white	2·75–3	6·03–7·01

Affords a blue pigment used in ceramic ware; latterly it is in demand for steel alloys up to 4%, and ores carrying 50–55% Mo are worth about 20–40*l.* a ton, while 95% metal brings 5*s.* per lb.

Nickel.				
Annabergite ..	greenish white	..	2–2·5	3–3·1
Breithauptite ..	copper-red	..	5–5·5	7·5–7·6
Chloanthite ..	tin-white	dark-grey	5·5–6	6·4–6·7
Garnierite	apple-green to white	paler	2–4	2·3–2·8
Gersdorffite ..	iron-grey	greyish black	5·5	5·6–6·9
Millerite	yellow	bright	3–3·5	4·6–5·6
Niccolite	copper-red	brownish	5–5·5	7·3–7·6
Zaratite	emerald green	paler	3–3·25	2·5–2·69

The rich ores have but little commercial importance; of the 10,000–20,000 t. metal yearly produced, nearly ½ (Canada) is recovered from nickeliferous pyrrhotite (1½–3% Ni when picked), and about the same quantity from garnierite (New Caledonia) dressed

Platinum.—Properly speaking there is perhaps no ore of platinum, the metal always occurring in the native state, though almost invariably alloyed more or less with one or more of the following rare metals—iridium, osmium, palladium, and rhodium; and these

Chemical Composition.	Structure.	Remarks.
Hg_2Ag	rhombic dodecahedron, 32	79% Hg
Hg_2Cl_2	tetragonal, 15	85% Hg
HgS	rhombohedral, 18	86% Hg

ranging from ¾ to 5%. The market value of mercury is about 2*s.* per lb.; output, some 4000 t. yearly, about ½ from Spain, ¼ United States, ⅛ Austria, ⅛ Russia.

MoS_2	scales, foliated, granular	60% Mo
MoO_3	earthy powder	molybdic ochre; 66% Mo
$MoPbO_4$	tetragonal, 10	26% Mo

Queensland exported 21 t. molybdenite, value 2673*l.*, in 1906. Molybdenite is very like graphite in appearance, but differs in being soluble in acids, and giving a greenish streak on damp paper.

$3NiO.As_2O_5.8H_2O$	coating on niccolite	29% Ni
NiSb	trigonal, 20	often with some iron, arsenic, and lead; 32% Ni
NiAs	cubic, 28	43% Ni
NiMg silicate	indefinite	genthite, noumeaite; 3 to 26% Ni
$NiS_2.NiAs_2$	cubic, 28	32 %Ni
NiS	trigonal, 20	nickel pyrites; 64% Ni
NiAs	trigonal, 20	copper-nickel; 43% Ni
$NiCO_3.6H_2O$	stalactite	emerald-nickel; 25 % Ni

to 7–8%. The trade is controlled by powerful smelting corporations; the metal costs 1*s.* 6*d.*–2*s.* 6*d.* per lb. The presence of copper is highly injurious, as it cannot be separated, except by wet process. Nevertheless, scarcely any ores are entirely free from it.

are collected indiscriminately from sands which are generally also auriferous. But it has been found in intimate association with the the rare mineral laurite (ruthenium sulphide) in Borneo, and the nickeliferous pyrrhotite of Canada is said to contain an arsenide of platinum ($PtAs_2$), called sperrylite, which may one day be utilised. The trade is a monopoly, and the refined metal is worth 2–5*l.* per oz. The associated metals named above are similarly controlled.

Potassium.	Colour.	Streak.	Hardness.	Sp. Gr.
Adularia	colourless	colourless	6–6·5	2·4–2·69
Agalmatolite ..	greenish-grey	white	1–3	2·5–2·9
Apophyllite ..	reddish, greyish, greenish	colourless	4·5–5	2·3–2·4
Carnallite	whitish to reddish	..	1	1·618
Kainite	greyish, yellowish	..	2·5	2·13–2·2
Nitre	white	white	1·5–2	1·93–2·3
Sylvite	white to colourless	white	2	1·9–2
Syngenite	colourless	..	2·5	2·6

Carnallite (27% potassium chloride), kainite (23% potassium sulphate), and kieserite (see Magnesium) are mined together, the two former being known commonly as *potash salts*; they are used in the crude state as fertilisers, and form the basis of considerable chemical industry, being worth about 35*s.* per ton, on a basis of

Silica.				
Quartz..	white and tinted	..	7	2·5–2·8

A great many varieties are distinguished, mainly due to coloration by ferric oxide, etc., thus—agate, amethyst, aventurine, bloodstone, cairngorm, carnelian, cat's-eye, chalcedony, chert, chrysoprase, flint, floatstone, heliotrope, hyalite, hydrophane, jasper, menitate, milky quartz, moss-agate, onyx, opal, plasma, prase, rock-crystal, rose-quartz, sard, sardonyx, sinter, tiger's-eye, tridymite, wood-opal, etc.; some are used in jewellery and decoration.

The mineral known as *diatomite* or *kieselguhr* is essentially silica (75–81%), with alumina (3½–10), iron (3), lime (½–2½), and some other impurities; it is used for polishing, and as an absorbent of nitro-glycerine in explosive compounds, and is worth about 30*s.* a ton. When pure, it is both fire-proof and acid-proof, and strongly

Silver.				
Argentite[1]	dark grey	as colour	2–2·5	7·19–7·36
Bromyrite[2]	yellow, greenish	..	2–3	5·8–6
Castillite[2]	an impure bornite	..	..	..
Cosalite[2]	grey	black	2·5–3	6·39–6·75

Commercial supplies are almost entirely from Russia (100,000–200,000 oz. per ann.), Colombia contributing 2000–10,000 oz.

Chemical Composition.	Structure.	Remarks.
$K_2O.Al_2O_3.6SiO_2$	monoclinic, 5	
$K_2O.3Al_2O_3.9SiO_2.3H_2O$	indefinite	
$K_2O.8CaO.16SiO_2.16H_2O$	tetragonal, 15	
$KCl.MgCl_2.6H_2O$	rhombic, 8	
$KCl.MgSO_4.3H_2O$	..	
KNO_3	rhombic, 8	saltpetre
KCl	cubic, vii.	
$K_2SO_4.CaSO_4.H_2O$	..	kaluszite

12% potash. The production of kainite is about 600,000 t. per ann., and of other potash salts, 800,000–900,000 t., all from Stassfurth, Prussia. *Saltpetre* comes almost exclusively from India (15,000–30,000 t. yearly), and is worth 12–15*l.* per t. Its principal use is for making gunpowder and nitric acid.

SiO_2	hexagonal	

non-conductive of heat, so that it is useful in boiler lagging; wood pulp now replaces it largely in explosives. It is mainly produced in Germany. *Pumice* contains 57-73% silica, and 9–20 alumina; it is used for polishing, and varies in value from 25*s.* to 40*l.* a ton. It is principally obtained from Lipari (6000 t. per ann.). *Fire-clay* is virtually 97% silica; iron (over 6%), and magnesia, lime, soda and potash (above 3% total) are fatal objections. *Mica*, of various kinds, consists of complex silicates of alumina, with potash (muscovite or white mica), magnesia and iron (biotite or amber mica). The United States produce 2000–5000 t., and India 1000 t. per ann. Prices range from 6*d.* to 6*s.* a lb. By far its greatest use is for electrical insulation.

Ag_2S	cubic, 32	87% Ag
$AgBr$	..	57% Ag
$(AgCu)_2S$	massive	4½% Ag
$2(CuPbZnFe)S$		
$2(Ag_2Pb)S.Bi_2S_3$	massive	2½–16% Ag

Silver (*cont.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Dyscrasite[2]	white	white	3·5–4	9·44–9·85
Embolite[1]	greenish	..	1–1·5	5·31–5·43
Freibergite[1] ..	..	..	..	..
Freieslebenite[2] ..	grey	..	2–2·5	6–6·4
Iodyrite[2]	yellow, greenish	..	soft	5·6–5·7
Kerargyrite[1] ..	colourless to pale grey	shining	1–1·5	5·5
Polybasite[1]	iron-black	..	2–3	6·21
Proustite[1]	cochineal red	as colour	2–2·5	5·42–5·56
Pyrargyrite[1] ..	greyish red	red	2–2·5	5·7–5·9
Silver (native)[1] ..	white, tarnishing	white	2·5–3	10·1–11·1
Stephanite[1] ..	iron-black	as colour	2–2·5	6·2–6·3
Stromeyerite[2] ..	grey, black	steel-grey	2·5–3	6·2–6·3

This metal is remarkable for the number of its natural compounds, which are not nearly exhausted in the above list. The group marked [1] are most important; those which contribute less are marked [2]; all are of commercial significance. Very much silver is also recovered as a by-product from gold-bearing, copper, lead, and zinc ores; a mixed sulphide called cylindrite, containing sulphides of tin (5–7% Sn), antimony, arsenic, zinc, and lead, affords 150 oz. silver per ton. The world's output is about 200 million oz. yearly, of which Mexico yields 65–70 million, the United States 55, Australia 14, Germany 12, Bolivia 7, Canada 5–8, Peru 5, Spain 4, Japan 3; value, about 2*s.* to 2*s.* 6*d.* per oz. The silver market is intimately connected with banking transactions, large exports being made from London to the Far East in settlement of balances; hence the silver market of the world is determined by London silver brokers. The quotation is for "standard" silver, i.e. ·925 fine. Silver ores contain a wide range of impurities, and the customs of smelters in purchasing them vary considerably, but the following conditions are pretty general:—

Sodium.				
Albite	white with many tints	colourless	6–7	2·5–2·64
Analcime	do	white	5–5·55	2·25–2·29
Glauberite	red, grey, yellow	..	2·5–3	2·64–2·85
Glauber-salt ..	colourless	..	1·5–2	1·4–1·5
Natrolite	white, reddish, yellowish	white	5–5·5	2·17–2·2

Chemical Composition.	Structure.	Remarks.
Ag_3Sb	..	72% Ag
$Ag(ClBr)$	..	61–73% Ag
indefinite	..	richly argentiferous fahlerz
$5(Ag_2Pb)S.2Sb_2S_3$	monoclinic, 5	22–24% Ag
AgI	..	46% Ag
$AgCl$	..	horn silver; 75% Ag
$9(Ag_2Cu)S.(SbAs)_2S_3$	monoclinic, 5	64–72% Ag
$3Ag_2S.As_2S_3$	rhombohedral, 20	65% Ag
$3Ag_2S.Sb_2S_3$	rhombohedral, 20	60% Ag
Ag	octohedral, 32	often contains copper, gold, etc.
$5Ag_2S.Sb_2S_3$	rhombic, 7	68% Ag
$Ag_2S.Cu_2S$	rhombic	61% Ag

Silver contents are paid for up to 98–100% of assay.

Gold contents are paid for if not less than ·1 oz. (2 dwt.) per ton.

Lead contents are paid for up to 90% of assay, if at least 5%.

Iron and manganese (added) in excess of silica contents are paid for at a scale per unit.

Deductions are made at scales per unit—

For each unit of zinc, if over 8%.

For each unit of antimony, arsenic and baryta (added), if over 3%.

For each unit of sulphur, if over 3%.

For each unit of silica in excess of iron and manganese (added).

The Broken Hill group of mines work lead ores carrying about 40 oz. silver per ton; the Bolivian ores are essentially silver, and yield 100–5000 oz. per ton; the Colorado ores are pyritic iron, with 20–50 oz.; some of the Californian and Canadian ores carry much cobalt-nickel pyrites, and up to 1000 oz. Ag per ton.

$Na_2O.Al_2O_3.6SiO_2$	triclinic, 2	
$Na_2O.Al_2O_3.4SiO_2.2H_2O$	cubic, 32	
$Na_2SO_4.CaSO_4$	..	
$Na_2SO_4.10H_2O$	..	mirabilite
$Na_2O.Al_2O_3.3SiO_2.2H_2O$	rhombic, 5, 8	

Sodium (*cont.*)	Colour.	Streak.	Hardness.	Sp. Gr.
Natron	white, grey, yellow	..	1–1·5	1·42–1·46
Salt	white with many tints	white	2–2·5	2·1–2·25
Sodalite	grey, green, red, yellow, brown	..	5·5–6	2·1–2·4
Soda-nitre	greyish, yellowish, reddish	..	1·5–2	2·1–2·3
Thenardite ..	colourless	..	2·5	2·73
Thermonatrite ..	white, grey, yellow	..	1–1·5	1·5–1·6
Trona	greyish, yellowish	..	2·5–3	2·11

Commercially useful members are common *salt* and *soda-nitre*: the former is worth about 10*s*. a ton; the latter, employed as a ferti-

Strontium.				
Celestine	very pale reddish, bluish	white	3–3·5	3·9–3·96
Strontianite ..	geeenish, yellow, grey, brown,	white	3·5–4	3·6–3·7

Strontium salts are employed in sugar-refining, and have a

Sulphur.				
Brimstone	yellow	..	1·5–2·5	2·072
Pyrites	(described under *Arsenic*, *Copper*, and *Iron*)			

Native *sulphur* comes chiefly from Sicily and Italy (some 500,000 t. yearly), and from Louisiana, U.S.A. (see p. 375), which of late years has swamped the market. Japan produces 15,000–20,000 t., and exports about 75% of it. In Sicily, 20% ore is workable; in Japan, ore under 38% is rejected, and much of it gives 50%. Prices have fallen from 5–6*l*. a ton to 3–4*l*. The consumption is mainly as a source of SO_2 for sulphuric acid manufacture, and in this capacity it is very largely replaced by pyrites. Pyrites are valued on their sulphur contents primarily, but this estimate is discounted by carbonaceous impurities (as in "coal-brasses"), excess of arsenic

Tantalum.				
Columbite	black	..	just scratched with good knife	5–7

Chemical Composition.	Structure.	Remarks.
$Na_2CO_3 . 10H_2O$	..	halite
$NaCl$	cubic, vii.	
$3Na_2O . 2NaCl . 3Al_2O_3 . 6SiO_2$	..	
$NaNO_3$	rhombic	Chili saltpetre
Na_2SO_4	rhombic	
$Na_2CO_3 . H_2O$	..	
$Na_2CO_3 . HNaCO_3$	..	

liser and in chemical manufactures, comes almost exclusively from Chili (about 1,500,000 t. yearly), and is worth approximately 9*l.* a ton.

$SrSO_4$	rhombic, 8	often with barium, calcium, and iron salts; 47% Sr
$SrCO_3$	rhombic, 8	lime, manganese, and iron impurities, 59% Sr

limited use in pyrotechny.

S	rhombic	native sulphur
..	..	30 to 52% S

(though practically all contain some), and a pulverulent character. Main sources of English supplies are local brasses (30% S), Irish pyrites (30–35%), Norwegian (45%, but generally 1½% As), Spanish (ditto, but carrying copper, for which the cinders are subsequently treated). Newfoundland affords a 52% ore, burning well and free from As. The United States, chiefly Mass., produce 100,000–250,000 t. per ann., classified as "lump" (clean natural ore), "spall" (lump broken to 2½-in. ring, and screened), both worth 4¾–5¼*d.* per unit for 38–45% S; and "fines" (dust below ⅝-in. screen, washed and jigged), value 4¼*d.*–4¾*d.*

..	..	tantalite 22–50% Ta_2O_5

This rare ore may be distinguished from wolfram by comparative absence of cleavage, from magnetite by lack of magnetism, and from tourmaline by much greater density. Several tantalum-yielding minerals (ferric, manganic, and antimonic) have been widely found, all containing more or less niobium. American ores carry 25–40% Ta_2O_5, Norwegian up to 75%, N.W. Australian 45–70% Ta_2O_5, 6–25% Nb_2O_5, and 16–20% Mn_2O_3. They occur with

Tellurium.	Colour.	Streak.	Hardness.	Sp. Gr.
Calaverite	bronze-yellow	yellowish-grey	2·5	9·043
Coloradoite ..	black to grey	..	3	8·627
Hessite	grey	..	2–3·5	8·13–8·6
Krennerite	white to brassy	..	..	8·35
Nagyagite	dark grey	..	1–1·5	6·8–7·2
Petzite	grey	..	2–5·3	8·7–9·02
Sylvanite	grey, white, yellow	as colour	1·5–2	5·73–8·3
Tellurium (native)	tin-white	as colour	2–2·5	6·1–6·3
Tetradymite ..	grey-white	..	1·5–2	7·2–7·9

The demand for tellurium is trifling, but the auriferous tellurides are highly valuable for their gold contents, which they yield with

Thorium, etc.				
Monazite	brown, red, yellow	..	5–5·5	4·9–5·2
Orangite	orange-yellow	..	..	4·88–5·39
Pyrochlore	red-brown	..	5–5·5	4·3
Samarskite	brown	..	5·5–6	5·6–5·75
Thorite	yellow, brown, black	..	4·5–5	5–5·4

Several of these complex minerals, and perhaps others, are known in trade as *monazite* sand. The regions in which workable deposits occur are few in number and small in extent—N. and S. Carolina, the Brazilian coast, the Samarka river (Russia), with stream tin in Australia and Tasmania, and in similar company throughout the Malay Peninsula. Magnetic concentration (Wetherill machine) is sometimes applied in N. Carolina, to raise the tin and tungsten in granitic rocks, and with tin and zinc in France. The metal has remarkable properties—insolubility in all mineral acids (except hydrofluoric), and in alkalies, exceeding hardness, and great brilliance as an electric lamp filament. Ore carrying 22% Ta_2O_5 is quoted in N. York at 5*l.* per unit, or 88*l.* per ton. In 1905, about 70 t. exported from W. Australia fetched 8925*l.*, but the only market (Germany) is now said to be overstocked.

Chemical Composition.	Structure.	Remarks.
$(AuAg)Te_2$	..	55½% Te, 44½ Au
HgTe	..	38½% Te
Ag_2Te	..	35–44% Te
$(AuAg)Te_2$	..	45% Te, 35½ Au
$AuTe.PbTe$	rhombic, 8	15–30% Te, 6–12 Au
$(AgAu)_2Te$	..	33% Te, 25 Au, 42 Ag
$(AuAg)Te_3$	monoclinic, 5	56% Te, 28 Au, 16 Ag
Te	trigonal, 21	traces of gold and iron
$2BiTe_3.Bi_2S_3$	trigonal, 21	48% Te

difficulty. Their occurrence is much wider than is commonly supposed.

$3(ThCeLaDi)_2PO_4$	..	
..	..	
$ThO_2Nb_2O_5.TiO_2.$ $CaO.CeO.FeO.UO_2.$ $HgO.Na_2O.F.H_2O$	..	
$ThO_2.UO_3.Cb_2O_5.$ $Ta_2O_5.WO_3.SnO_2.$ $ZrO_2FeO.CuO.$ $MgO.CeO.CaO.YO$	..	
$ThSiO_4.2H_2O$	..	

contents of thoria (ThO_2) to 5%, in accordance with buyers' demands. Brazil produces about 4000 t. per ann., and the United States 400 t. The market (Germany) is controlled by a ring, and prices are arbitrary. Thoria (present as an impurity in monazite sands) is used in preparing incandescent gas mantles. A new mineral (thorianite) found in Ceylon is so much richer in thoria that it threatens to displace monazite entirely.

Tin.	Colour.	Streak.	Hardness.	Sp. Gr.
Cassiterite	black, brown, yellow	white to brown	6–7	6·4–7·1
Stannite	grey to blackish	blackish	4	4·3–4·5

The confusing and old-fashioned method of computing the value of tin ore, by which the ignorant seller was robbed of at least one-third of the worth of his parcel, is now replaced by an accurate assay, the percentage of pure tin being worked out as tin oxide (black tin). The presence of titanium does not affect the value to any extent beyond reducing the percentage of tin; but tungsten lowers the value. When a market can be found for the product, ore may be freed from tungsten by treating with soda sulphate, and washing out the soda tungstate.

By far the most important ore is cassiterite, which is almost always associated more or less closely with granitic rocks. The world's output of tin is about 90,000–100,000 t. per ann., of which, Malaya affords 70,000–75,000 t.; Bolivia, 10,000–15,000 t.; Australia, 3000–6000 t.; and Cornwall, 4000–5000 t. A very large proportion of the Malayan product is the result of Chinese manual labour in alluvial deposits, and is closely dependent on selling prices: the majority of the Chinese gravel mines would close on

Titanium.				
Anatase . ..	brownish or bluish black	colourless	5·5–6	3·8–3·9
Brookite	brown, red, black, grey	..	5·5–6	4
Ilmenite	black, brownish, reddish	sub-metallic	5–6	4·5–5
Rutile	reddish, brownish, yellowish, blackish	pale-brown	6–6·5	4·18–4·25
Sphene	grey, green, brown, yellow, black	white	5–5·5	3·4–3·56

At present, none of these ores has any commercial value, either

Tungsten.				
Hubnerite	..	..	..	..
Scheelite	yellow, brown, green, red	white	4·5–5	5·9–6·1
Tungstite	yellow, green	..	soft	..
Wolfram	grey-black	reddish-brown	5–5·5	7·1–7·9

Chemical Composition.	Structure.	Remarks.
SnO_2	tetragonal, 15	79 % Sn; sometimes 9 % Fe_2O_3
$SnS_2.Cu_2S.FeS$	cubic, tetrahedral, 31	27% Sn; bell-metal ore

tin falling below 150–155*l.* per ton for any length of time; ordinary returns are 2–4 lb. up to 30–40 lb. per cub. yd. The Bolivian lode mineral is rich, up to 10–20%, but the mines lie at such altitudes (15,000 ft. upwards) that labour is ineffective and costly, and transport extremely difficult; improved railway facilities, however, are partially remedying this. The Tasmanian product is partly alluvial and partly lode; and the lode mineral is sometimes cassiterite (often closely associated with bismuth and wolfram), and sometimes stannite (with argentiferous galena). Much of the Queensland lode tin is intimately mixed with bismuth, ores carrying 3–20% tin-bismuth; concentrated to 57% Sn and 3½% Bi they are shipped to Germany, no other smelters accepting them. An average yield for Cornish ores is 1½–3% Sn; the black tin (SnO_2) contains 78·8% Sn; "crop" tin, 70·2%; slimes, 60·5%. Most of the slimes carry ½–2% Cu, and the average all round is about ⅓% Cu; much of the "crop" shows ¼–¾% As, and in several cases there is 1–5% WO_3. All contain 1½–12 (av. 5%) SiO_2, and ½–12 (av. 4½%) Fe.

TiO_2	tetragonal, 15	octahedrite; 60% Ti
TiO_2	rhombic, 8	arkansite; 60% Ti
$TiO_2.FeO$	rhombohedral, 17	titanic iron; 31% Ti
TiO_2	tetragonal, 15	60% Ti
$TiO_2.CaO.SiO_2$	monoclinic, 5	titanite; 24% Ti

for its iron or for its titanium.

$MnWO_4$	..	60·7% W
$CaWO_4$	tetragonal, 13	64% W
WO_3	earthy, pulverulent	tungstic ochre; 80% W
$(FeMn)WO_4$	monoclinic, 5	51% W

The order of commercial importance is wolfram, scheelite, hubnerite.

Tungsten finds its greatest field in the manufacture of "high-speed" or "self-hardening" steel for special purposes. Alloyed (either alone or in connection with molybdenum and chromium) with steel it imparts great toughness and ability to resist shock. Hence for high-speed tool steel, armour plate, projectiles, car-springs, etc., it occupies a prominent position. It is also alloyed with aluminium, copper and many other metals. Filaments of tungsten have been adopted in electric lamps. Sodium tungstate

Uranium.	Colour.	Streak.	Hardness.	Sp. Gr.
Autunite	yellow	yellowish	2–2·5	3–3·2
Chalcolite	green	paler	2–2·5	3·4–3·6
Pitchblende ..	black, grey, brown	as colour	3–6	4·8–8
Uraconite	yellow	..	..	3·78–3·97

The extraordinary interest in radium, which is essentially associated with uranium, gives the latter a new importance, and should stimulate prospecting, especially in pegmatite (coarse granite) country. The present chief source of uranium is a single occurrence of pitchblende in Cornwall, the vein carrying 18–29% metal. The principal use (as a soda salt) is for staining glass and porce-

Vanadium.				
Dechenite	red	..	3·4	5·6–5·81
Descloizite	green, brown	..	3·5	5·8
Mottramite	black	yellow	3	5·89
Psittacinite ..	green	..	..	..
Pucherite	red, brown	..	6	6·25
Roscoelite	brownish, greenish	..	soft	2·92–2·94
Vanadinite	yellow, brown, red	..	2·7–3	6·6–7·2
Volborthite	green	..	3–3·5	3·5

The great consumption of specially hardened steel in the automobile industry has caused an exceptional demand for vanadium, which is used in alloys up to 10–20%. Ores containing 3–5% min. are sold at 20*s*. per ton. Vanadium salts are employed in photography, and as pigments. For a long time the only available supply is used as a mordant in dyeing, and for rendering textile materials non-inflammable. For this last purpose, scheelite is objected to. The world's production of tungsten concentrates is 1500–2000 t., of which about half is raised in Colorado (U.S.), and 200–300 t. each in Cornwall, Portugal, and New S. Wales. A new find in Brazil is likely to materially influence supplies in the near future. Market prices fluctuate seriously, and have ranged from 14*s.* to 40*s.* per unit in very short periods. "First class" concentrates must not contain more than ·25% P, nor more than ·01% S, and should carry at least 60% WO_3.

Chemical Composition.	Structure.	Remarks.
$U_3O_4.(FeCuCa)O.H_2SO_4$	rhombic, 8	uranocalcite; 68% U
$2U_2O_3.CuO.P_2O_5.8H_2O$	tetragonal, 15	uran-mica, uranite, torbernite; 67% U
U_3O_4(with Pb, Fe, Ag, Ca, Mg, Bi, SiO_2, etc.)	cubic, vii.	uraninite; about 80% U
U_2O_3	earthy, pulverulent	uran-ochre; 90% U

lain, but a new demand has lately grown for adding to thoria for making incandescent gas mantles. Clean pitchblende is worth 8–18*s.* a lb.; a uraniferous sandstone (carnotite) carrying 3–5% uranium oxide brings 20*s.* per unit, and is sometimes concentrated to 40–50%.

PbV_2O_6	..	
$4(PbZn)V_2O_6H_2O$	rhombic	27% V
..	..	these may be identical; 17–19% V_2O_5
$3Pb_3V_2O_8.Cu_3V_2O_8.6CuH_2O.12H_2O$	..	
$Bi_2V_2O_8$	rhombic	15% V
indefinite	..	20–30% V_2O_3
$3Pb_3V_2O_8.PbCl_2$	..	10% V
$(CuCa)V_2O_5.H_2O$	...	33% V

was from the Spanish lead ores, carrying 4–5% vanadic oxide (as vanadinite), but probably the near future will see markets completely upset by an immense deposit of vanadium sulphide (40%) discovered in Peru.

Yttrium.	Colour.	Streak.	Hardness.	Sp. Gr.
Gadolinite	green-black	colourless	6·5–7	4·2–4·35
Yttrocerite	grey, yellow, red	..	4–5	3·45
Yttrotantalite ..	black, brown	..	5–5·5	5·4–5·9
Yttrotitanite ..	brown, black	..	6–7	3·5–3·7
Zinc.				
Blende	black, brown, yellow, red	brownish	3·5–4	3·9–4·2
Calamine	grey, green, brown white	white	4·5–5	3·1–3·9
Goslarite	white	white	2–2·5	1·9–2·1
Smithsonite ..	green, grey, brown	..	5	4–4·5
Willemite	yellow, brown	..	5–5·5	4–4·1
Zincite	red, orange	orange	4–4·5	5·4–5·7

Practically all zinc ores are found in a very impure state, largely mixed with argentiferous galena, and with iron and manganese, considerable supplies of metal being got from the ferro-zinc sulphides christophite and marmatite (30–40% Zn), and from franklinite ($5\frac{1}{2}$% Zn), a mixture of zinc, iron, and manganese oxides. The yearly output of metal is 500,000–700,000 t., and the price fluctuates considerably between 15*l.* and 30*l.* per ton. The chief deleterious ingredient of zinc ores is lead. The only ores absolutely free from lead are those from Lehigh. All others contain some small proportion, say ·01%. Antimony, arsenic, cadmium, copper, iron, and lead are injurious, both in the roasting of blende and in the subsequent distilling of the oxide. The smelter who buys blende or calamine bases his estimate of the value of a parcel of ore to him in somewhat the following way. He takes the market price of spelter, which may be assumed at 20*l.* a ton: from this he deducts 6*l.* per ton as the cost of smelting. Then from the zinc contents of the ore by assay, say 45%, he deducts 15% if blende, or 10% if calamine, as being the proportion of metal which will be lost in the slags and vapours, so that he has 30 or 35% of metal which he can reckon on extracting. Thus the market value of the zinc product of the ore is arrived at by a rule of three sum, e.g.—

As $\frac{\text{Spelter}}{100\%}$ is worth 14*l.* a ton, then $\frac{\text{Blende}}{30\%}$ is worth 4*l.* 4*s.*

Zirconium.				
Zircon	red, brown, yellow, green	colourless	7·5	4–4·75

Used in

Chemical Composition.	Structure.	Remarks.
$3(YLaFeBe)SiO_3$	monoclinic, 5	
$2YF_2.2(9CaF_2).CeF_2.3H_2O$	..	
$(YCaFe)_2.Ta_2O_7$	rhombic	
$YO.SiO_2.TiO_2.CaO.Al_2O_3.Fe_2O_3.CeO$	..	
ZnS	cubic, 31	black-jack, sphalerite; 67% Zn
$2ZnO.SiO_2.H_2O$	rhombic	54% Zn
$ZnSO_4.7H_2O$	rhombic, 8	white vitriol; 22% Zn
$ZnCO_3$	rhombohedral, 17	52% Zn; dry-bone
$2ZnO.SiO_2$	hexagonal, 17	58½% Zn
ZnO	hexagonal, 20	80% Zn

But as the smelter must make a profit, he offers such a figure below 4*l.* 4*s.* as will leave him the margin he desires. When the ore is very impure, more than 15% will be lost in the slags, etc. The ultimate value of any zinc ore depends upon (*a*) percentage content of metallic zinc, (*b*) whether the residue left after smelting for spelter or for zinc oxide can be profitably treated for gold, silver or copper, or, as in the case of franklinite for manganese; (*c*) percentage content of elements which deteriorate the product or increase expense of reduction. Blende is deducted as an objectionable element in calamine ores, and calamines when found in blende are not paid for. Sulphur, which composes about one-third the weight of pure blende, is not considered of value, because the cost of converting it into sulphuric acid (a necessity at some works) leaves little, if any, profit.

Of the world's output of metallic zinc (in 1000 t.), Germany affords 150–200, Belgium 120–150, United States 100–200, England 30–50, Holland 7–15, Austria 7–10, Russia 6–10, and Spain 5–6. Australia produces 15,000–35,000 t. of metal per ann. in the ore exported, practically all from the Broken Hill mines, N.S.W., where the mineral is in the nature of a bye-product, and will, in the immediate future, be turned out in much larger quantities.

$ZrSiO_3$	tetragonal, 15	hyacinth, etc.

jewellery.

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GLOSSARY.

Abattis (Leicester), cross packing of branches or rough wood, to keep roads open for ventilation.
Abbruch (Germ.), ore broken off the vein.
Abendort (Germ.), western end of a mine.
Abendschicht (Germ.), afternoon shift.
Abendstoss (Germ.), western end of a mine
Abflanherd (Germ.), buddle.
Abfüllen (Germ.), to win a good body of ore.
Abkomniss (Germ.), junction of main lode and tributary.
Abra (Span.), fissure in a lode.
Abronziado (Span.), copper sulphides.
Abstrich (Germ.), the mass of black litharge appearing on the bath of work-lead early in cupelling.
Abtheilung (Germ.), a defined district of a mine under care of a deputy.
Abzug (Germ.), the first scum appearing (before *abstrich*) on molten lead.
Accompt (Corn.), settling day or place.
Achicar (Span.), to decrease water in a mine.
Acreage rent (Eng.), royalty or rent for working minerals.
Addlings (N. Eng.), earnings.
Ademar (Span.), to timber.
Ademador (Span.), mine carpenter.
Adit, a slightly rising tunnel driven into a mine from daylight, and used for access, transport, and drainage.
Adobe (Span.), sun-dried brick.
Adventurers, original prospectors.
After-damp, poisonous gas resulting from explosions of fire-damp chiefly carbonic acid.
Agitator (W. Amer.), settler.
Ahondar (Span.), to sink.
Air-box, wooden tubes, 9–15 ft. long, for ventilating headings or sinkings.
Air-course, ventilation channels.
Air-crossing, a bridge carrying one air-course across another
Air-end way, ventilation levels run parallel with main level.
Air-gate (Midlands), ventilation ways.
Air-head (Staff.), ventilation ways.
Airless end, unventilated extremity of a stall in long-wall workings.
Air-level, an old level used for ventilating.
Air-slit (Yorks.), a short head between other air-heads.
Air-stack, ventilating chimney.

Aitch-piece, parts of a pump in which the valves are fixed.
Albañil (Span.), mason.
Albayalde (Span.), white lead.
Alberti furnace, a continuous reverberatory for mercury ores.
Alcam (Wales), tin.
Alive (Corn.), productive.
Alligator (Amer.), rock breaker.
Alloy, compound metal.
Alluvium, gravel, sand, and mud deposited by streams.
Almadeneta (Span.), stamp-head.
Almagre (Span.), red ochre.
Aludel (Span.), earthen condenser for mercury.
Amalgamation, absorption of gold and silver by mercury.
Anemometer, wind measurer.
Anthracite, hard, very pure coal.
Apex (Amer.), end or edge of vein nearest surface.
Apolvillados (Span.), superior ores.
Appolt oven, a coke oven.
Aprons (Amer.), battery copper plates.
Arch (Corn.), portion of lode left standing to support hanging wall, or because too poor.
Arenaceous, sandy.
Arends tap, an inverted siphon for drawing molten lead from crucible or furnace.
Arenillos (Span.), refuse earth.
Argentiferous, silver-bearing.
Argillaceous, clayey.
Arian (Wales), silver.
Arm, the inclined leg of a set of timber.
Arrage (N. Eng.), sharp corner.
Arrastra (Span.), an amalgamating mill.
—— *de cuchara* (—), spoon arrastra.
—— *de marca* (—), large arrastra.
—— *de mula* (—), mule-power arrastra.
Arrastrar (Span.), union of veins.
Aspirail (Fr.), opening for ventilation.
Assessment work (W. Amer.), the annual work necessary to hold a claim.
Astel, overhead boarding in a gallery.
Astyllen (Corn.), small dam in an adit; partition between ore and deads on grass.
Atierres (Span.), refuse rock.
Atizador (Span.), furnace man.
Attle (Corn.), refuse rock.
Auger-stem, bar to which a drilling bit is attached.
Auget, priming tube.
Aur (Wales), gold.
Auriferous, gold-bearing.
Ausscharen (Germ.), junction of lodes.

Auszimmern (Germ.), timbering.
Average produce (Corn.), percentage of fine copper in ore.
Average standard (Corn.), price of pure copper in ore.
Aviador (Span.), he who provides the capital to work a mine.
Azogue (Span.), mercury.
Azogueria (Span.), amalgamating works.
Azoguero (Span.), amalgamator.
Azogues (Span.), inferior ores.

Back (Corn.), lode lying between adit and surface.
Back-casing, temporary shaft-lining of dry-laid bricks.
Back-end (N. Eng.), the last portion of a judd.
Backing, timbers let into notches in the rock across the top of a level.
Backing deals (N. Eng.), vertical planks behind the curbs in a shaft.
Back-shift, afternoon shift.
Back-skin (N. Eng.), a leather jacket for wet workings.
Bait, provisions.
Bal (Corn.), a mine.
Balance-bob, a heavy counterpoise to pump-rods.
Balk (N. Eng.), a hitch causing a nip.
Balland (Derb.), pulverulent lead ore.
Bancos (Span.), rocks disturbing a lode.
Band (N. Eng.), stone interstratified with coal.
Bank, (1) surface at pit's mouth; (2) deposit worked above water level; (3) coal face.
Bank claim (Aust.), mining right on bank of stream.
Bank right (Aust.), right to divert water to bank claim.
Baño (Span.), excess of mercury used in torta.
Bar, (1) ridge crossing lode or stream; (2) drilling-rod.
Bar diggings (W. Amer.), auriferous claims on shallow streams.
Bargain, portion of mine worked by a gang on contract.
Barilla (Span.), grains of native copper disseminated through ores.
Barmaster (Derb.), mine manager, agent, and engineer.
Barmote (Derb.), mining court.
Barrel-work (N. Amer.), native copper that can be hand-sorted ready for smelting.
Barrow (Corn.), a dump.
Basque, crucible or furnace lining.
Bass (Derb.), indurated clay.
Basset, an outcrop.
Batea (Span.), a bowl for separating metal from refuse.
Batt (Derb.), indurated clay.
Beans (N. Eng.), small coals.
Bean-shot, copper granulated by pouring into hot water.
Beat (Corn.), to cut away a lode.
Bed-claim (Aust.), a mining-claim on bed of stream.
Bed-rock, solid rock underlying alluvial.
Bede, miners' pickaxe.

Belly-helve, forge hammer lifted by cam.
Ben (Corn.), productive.
Benching up (N. Eng.), working on top of coal.
Bend (Derb.), indurated clay.
Benheyl (Corn.), flowing tin stream.
Biche, a hollow-ended tool for recovering boring rods.
Bind (Derb.), indurated clay.
Binder (Corn.), mine carpenter.
Bing (N. Eng.), 8 cwt. of ore.
Bing-hole (Derb.), an ore-shoot.
Bing-ore (Derb.), lead ore in lumps.
Bing-tale (N. Eng.), ore given to the miner for his labour.
Black band, an earthy carbonate of iron found in coal-beds.
Black copper, impure smelted copper.
Black damp, carbonic acid gas.
Black ends, refuse coke.
Black flux, charcoal and potassium carbonate.
Black jack, zinc blende.
Black lead, graphite.
Black sand (Aust.), dark minerals found with alluvial gold.
Black tin (Corn.), dressed tin ore.
Blanch, lead ore mixed with other minerals.
Blanched copper, copper alloyed with arsenic.
Blanket strake (Aust.), sloping tables or sluices lined with baize for catching gold.
Blick (Germ.), iridescence on gold and silver at end of cupelling.
Blind creek (Aust.), dry watercourse.
Blind level, (1) an incomplete level; (2) a drainage level.
Bloat, a hammer swelled at the eye.
Block claim (Aust.), a square mining claim.
Block tin, cast tin.
Blocking out (Aust.), washing gold gravel in sections.
Bloomary, a forge for making wrought iron.
Blossom, the decomposed outcrop of a vein or coal-bed.
Blower (N. Eng.), an outrush of gas.
Blue-billy, residue of copper pyrites after roasting with salt.
Blue elvan (Corn.), greenstone.
Blue-john (Derb.), fluorspar.
Blue lead (W. Amer.), a blue-stained stratum of gravel of great extent and richness.
Blue peach (Corn.), a slate-blue fine-grained schorl.
Blue stone, copper sulphate.
Bob (Corn.), triangular frame transmitting power from engine to pump-rods.
Boca (Span.), mine mouth.
Bocamma (Span.), mine mouth.
Bog iron ore, loose earthy brown hematite recently formed in swampy ground.
Boliche (Span.), concentrating bowl.

Bollos (Span.), triangular blocks of amalgam.
Bolsa (Span.), small bunch of ore.
Bonanza (Span.), body of rich ore.
Bone, slaty matter in coal seams.
Bonnet, the roof of a cage.
Bonney (Corn.), an isolated body of ore.
Bonze, undressed lead ore.
Booming, removing gravel by sudden outlets of pent-up water.
Borrasca (Span.), unprofitable ore.
Bort, amorphous dark diamond.
Bottoms, impure copper alloy below the matt in smelting.
Bounds (Corn.), a tract of tin ground.
Bout (Derb.), 24 dishes of lead ore.
Bowke (Staff.), small wooden box for hauling ironstone underground.
Bowse (Derb.), lead ore as cut from the lode.
Box-bill, tool for recovering boring-rods.
Brace (Corn.), buildings at pit mouth.
Braize (Amer.), charcoal dust.
Brake-sieve, hand jigger.
Brances, iron pyrites in coal.
Branch, small vein shooting off from main lode.
Brasses, iron pyrites in coal.
Brat, a thin bed of coal mixed with pyrites or limestone.
Brattice, a lining or partition.
Brazil (N. Eng.), iron pyrites in coal.
Breeze, small coke.
Brettis (Derb.), a timber crib filled with slack.
Broaching bit, a tool for re-opening bore-hole which has partially closed by swelling of the walls.
Brob, a spike to prevent timber slipping.
Broil (Corn.), traces of a vein in loose matter.
Brooch (Corn.), mixed ores.
Brood (Corn.), heavy waste from tin and copper ores.
Brownspar, ferruginous dolomite.
Browse, imperfectly smelted ore mixed with cinder and clay.
Bryle (Corn.), traces of a vein in loose matter.
Bucking, breaking down ore with a very broad hammer ready for jigging.
Buddle, an inclined stationary or revolving platform, on which ores are dressed.
Buitron (Span.), a silver furnace.
Bulkhead, (1) a tight partition, or stopping ; (2) the end of a flume carrying water for hydraulicing.
Bulldog, furnace lining.
Bully, a miners' hammer.
Bunch, a small rich ore body.
Bunding, a staging in a level for carrying débris.
Bunney, a nest of ore not lying in a regular vein.
Burden, earth overlying a bed of useful mineral.

Burr, solid rock.
Burrow, refuse heap.
Buscones (Span.), prospectors, fossickers, tribute workers.
Butt, coal surface exposed at right angles to the face.
Butty (Mid.), a contract miner.

Cabezuela (Span.), rich gold and silver concentrates.
Cal (Corn.), wolfram.
Cala (Span.), prospecting pit.
Caliche (Span.), felspar.
Callys (Corn.), stratified rocks traversed by lodes.
Canch, stone quarry.
Cancha (Span.), space for drying slimes.
Cand (Corn.), fluorspar.
Cank (Derb.), whinstone.
Caple (Corn.), hard rock lining tin lodes.
Carbona (Corn.), an irregular deposit of tin ore.
Case, a fissure admitting water into a mine.
Cata (Span.), a mine denounced but not worked.
Cauf (N. Eng.), a coal bucket or basket.
Caunter (Corn.), a vein crossing diagonally.
Cawk, baryta sulphate.
Cazeador (Span.), amalgamator.
Chats (N. Eng.), small pieces of stone with ore.
Chilian mill, a mortar mill.
Chimney, an ore shoot.
Choke-damp, carbonic acid gas.
Chuza (Span.), a washer.
Claggy (N. Eng.), when coal is tightly joined to the roof.
Claim, a portion of mining ground held under one grant.
Clean-up, collecting the product of a period of work with battery or sluice.
Cleat, (1) a joint in rock; (2) a wedge.
Clod, soft shale or slate roof to coal.
Coal-pipes (N. Eng.) very thin irregular coal beds.
Cob (Corn.), to break up ore for sorting.
Cobre, Cuban copper ores.
Cockle (Corn.), black tourmaline, often mistaken for tin.
Cod (N. Eng.), the bearing of an axle.
Cofer (Derb.), to caulk a shaft by ramming clay behind the lining.
Coffer, the iron box in which stamps work.
Coffin (Corn.), an old pit.
Coil drag, a tool for picking pebbles, &c., from drill holes.
Colas (Span.), brown sulphides.
Colorados (Span.), decomposed ores stained with iron.
Colours, particles of gold found in panning a sample.
Colrake, a shovel for stirring lead ores while washing.
Cope (Derb.), lead mining on contract.
Copela (Span.), dry amalgam.

Copelilla (Span.), zinc-bende.
Corbond, an irregular mass from a lode.
Corf, a mine wagon or tub.
Coro-coro (S. Amer.), grains of native copper mixed with pyrite, chalco-pyrite, mispickel, &c.
Corve, a mining wagon or tub.
Cost-book (Corn.), mining accounts.
Costean (Corn.), to prospect a lode by sinking pits on its supposed course.
Country, the formation traversed by a lode.
Cow, a self-acting brake.
Coyoting (W. Amer.), irregular mining by small pits.
Crab-hole (Aust.), water-worn holes in bed rock.
Cradle, a wooden trough for washing gold sands.
Cramp, a pillar left for support in a mine.
Cranch, part of a vein left by previous workers.
Creaze (Corn.), tin ore collected in the middle of the buddle.
Creep, movement of walls or floor of a mine.
Crib, a timber frame.
Cribble, a sieve.
Crop (Corn.), the richest portion of dressed tin ore.
Cross-course, a cross vein.
Cross-cut, a level driven across a vein.
Crosses and holes (Derb.), made in the ground by the discoverer of a lode to temporarily secure possession.
Crow-foot, a tool for drawing broken boring rods.
Culm (Eng.), anthracite; (N. Amer.), fine waste coal and dirt.
Curb, a timber frame.

Dam, a barrier for water or gases.
Damp, poisonous gas.
Dan (N. Eng.), a truck without wheels.
Dant (N. Eng.), soft inferior coal.
Dead, (1) unproductive; (2) unventilated.
Dead men's graves (Aust.), grave-like mounds in the basalt underlying auriferous gravels.
Dead riches (N. Amer.), lead carrying much bullion.
Dead roasting, roasting till all sulphur is driven off.
Deads, rubbish.
Dead work, unproductive work.
Dean (Corn.), the end of a level.
Desaguador (Span.) a water pipe or drain.
Desecho (Span.), floured mercury.
Desmontes (Span.), poor ores.
Despoblado (Span.), ore with much gangue.
Dessue (Corn.), to cut away the ground beside a thin vein so as to remove the latter whole.
Dialling, surveying.
Dillueing (Corn.), dressing tin slimes in a fine sieve.

Dippa (Corn.), a small catch-water pit.
Dish (Corn.), an ore measure; in lead mines a trough 28 in. long, 4 in. deep, and 6 in. broad; sometimes 1 gal., sometimes 14-16 pints.
Dizzue (Corn.), see Dessue.
Dolly, (1) a perforated board for breaking up clay in puddling; (2) a primitive quartz stamp.
Donk (N. Eng.), soft mineral found in cross veins.
Dradge (Corn.), inferior ore separated from the prill.
Dresser (Staff.), a large coal pick.
Drift, (1) a tunnel following the vein; (2) alluvial deposits.
Dropper, a branch leaving the vein on the footwall side.
Druggon (Staff.), a vessel for carrying fresh water into a mine.
Dumb'd, choked—of a sieve or grating.
Dump, a heap of ore or refuse.
Durn (Corn.), a timber frame.
Dürr (Germ.), barren ground.
Dzhu (Corn.), see Dessue.

Efydd (Wales), copper.
Elvan (Corn.), a belt of felspathic or porphyritic rock.
Estano (Span.), tin.

Fahlband (Germ.), a course impregnated with metallic sulphides.
Faiscador (Span.), a gold-washer.
False bottom, a bed of rock under alluvial which has other alluvial below it.
Famp (N. Eng.), thin beds of soft tough shale.
Fang (Derb.), an air course.
Fast (Corn.), bedrock.
Feigh (N. Eng.), ore refuse.
Ferro blanco (Span.), arsenopyrite.
Flang (Corn.), a double-pointed pick.
Flat wall (Corn.), footwall.
Float, detached fragments of a quartz reef.
Floran (Corn.), very fine tin.
Flouring, the breaking-up and contamination of mercury, rendering it useless for amalgamating.
Flucan (Corn.), clayey matter in a lode.
Fluke, a rod for cleaning out drill-holes.
Flume, a water conduit.
Fluthwerk (Germ.), river prospecting.
Fodder (N. Eng.), 21 cwt. of lead.
Foot (Corn.), 2 gal. or 60 lb. black tin.
Force piece, diagonal timbering to secure the ground.
Fork (Corn.), bottom of sump; (Derb.), prop for soft ground.
Fossicking, casual and unsystematic mining.
Fother (N. Eng.), $\frac{1}{3}$ chaldron.
Free milling, ores requiring no roasting or chemical treatment.

Gad, a wedge.
Gal (Corn.), hard gossan.
Gale, a grant of mining ground.
Galemador (Span.), a silver furnace.
Galiage, royalty.
Gangue, refuse associated with ore.
Ganister, furnace lining composed of fire clay and ground quartz.
Gatches (Corn.), final sludge from tin dressing.
Glist (Corn.), micaceous iron ore.
Goaf, worked-out ground, and the refuse with which it is filled.
Gob, see Goaf.
Gossan (Corn.), ferruginous quartz or calcspar filling a lode.
Goths (Staff.), sudden burstings of coal from the face owing to tension caused by unequal pressure.
Gouge (N. Amer.), soft clay lying between the ore body and sides of the lode.
Grass, surface.
Grassero (Span.), slag-heap.
Grena (Span.), undressed ore.
Grizzly (W. Amer.), a grating to throw out large stones from hydraulic gold sluices.
Groove (Derb.), a mine.
Grouan (Corn.), granite.
Grundy, granulated pig iron.
Guag (Corn.), worked-out ground.
Gubbin, ironstone.
Guija (Span.), quartz.
Gunnies (Corn.), 3 ft.
Gurt (Corn.), water runnel from dressing-floor.

Haiarn (Wales), iron.
Halvans (Corn.), inferior copper ore.
Hard-head, residue from tin refining; contains much iron and arsenic.
Hazle (N. Eng.), sandstone mixed with shale.
Hechado (Span.), dip.
Hilo (Span.), a thin metalliferous vein.
Hulk (Corn.), to pick out the soft portions of a lode.
Hungry, worthless looking.
Hushing, prospecting by laying ground bare by sudden discharges of pent-up water.
Hutch (Corn.), an ore-washing box.
Hydraulicing (W. Amer.), working auriferous gravel beds by hydraulic power.

Inch, Miners', see p. 23.
Irestone (Corn.), any hard tough stone.
Iron hat, decomposed ferruginous mineral capping a lode.
Itabirite (Braz.), micaceous iron ore.

Jacotinga (Braz.), ferruginous ores associated with gold.
Jales (Span.), tailings.
Jigging, separating heavy from light particles by agitation in water.
Judge (Derb. and N. Eng.), a measuring staff.

Kann (Corn.), fluorspar.
Kazen (Corn.), a sieve.
Keckle-meckle, poorest lead ore.
Kerned (Corn.), pyrites hardened by exposure.
Kevil (Derb.), a calcspar found in lead veins.
Kibble, a mining bucket.
Kieve, tossing-tub.
Killas (Corn.), clay slate.
Kirving (N. Eng.), the cutting made beneath the coal seam.

Lama (Span.), slimes.
Lappior (Corn.), an ore dresser.
Launder, water trough.
Lazyback (Staff.), a coal stack.
Leat, water-course.
Leavings (Corn.), halvans.
Limadura de plata (Span.), dry silver amalgam.
Linnets (Derb.), oxidised lead ores.
Lista (Span.), tail of impure mercury.
Long-tom, a gold-washing trough.
Loob (Corn.), sludge from tin dressing.

Macizo (Span.), unworked lode.
Magistral (Span.), roasted copper pyrites, copper sulphate, &c., used to reduce silver ores.
Makings (N. Eng.), small coal produced in kirving.
Manga (Span.), canvas bag for straining amalgam.
Maquilla (Span.), a custom mill.
Marmajas (Span.), concentrated sulphides.
Maza (Span.), stamp head.
Mear (Derb.), 32 yd. along the vein.
Miners' inch, see p. 23.
Mistress (N. Eng.), a miners' lamp.
Mock-lead (Corn.), zinc-blende.
Moil (Corn.), a wedge-pointed drill.
Mucks (Staff.), bad earthy coal.
Mundic, pyrites.

Negrillo (Span.), black sulphide of silver.
Nittings, refuse of good ore.
Noria (Span.), an endless chain of buckets.
Nuts, coal of a certain size.

Oro (Span.), gold.
Oroche (Span.), retorted bullion.
Overburden, soil overlying a bed of useful mineral.

Packing (Corn.), final dressing of tin or copper ore.
Pacos (Span.), ferruginous silver ores.
Panning, washing earth or crushed rock in a shallow dish (see p. 325).
Pasilla (Span.), dry silver amalgam.
Peach stone (Corn.), chlorite schist.
Pee (Derb.), a fragment of lead ore.
Pepenado (Span.), dressed ore.
Pilch (Corn.), portion of lode worked by tributers.
Pillion (Corn.), metal remaining in slag.
Piping, hydraulicing.
Placer, an alluvial mine.
Plata (Span.), silver.
Plata cornea amarillia (Span.), iodyrite.
Plata cornea blanca (Span.), cerargyrite.
Plata cornea verde (Span.), embolite.
Plata mixta (Span.), gold and silver alloy.
Plata negra (Span.), argentite.
Plata pasta (Span.), spongy silver bars after retorting.
Plata piña (Span.), silver after retorting.
Plata verde (Span.), bromyrite.
Plate (N. Eng.), scaly shale in limestone beds.
Plomb d'œuvre (Fr.), dressed galena.
Plomo (Span.), lead, galena.
Plush copper, chalcotrichite.
Plwm (Wales), lead.
Poch-erz (Germ.), ore requiring to be crushed and dressed.
Podár (Corn.), copper pyrites.
Polroz (Corn.), water-wheel pit.
Polvillos (Span.), rich ores.
Polvoulla (Span.), black silver.
Pot growan (Corn.), decomposed granite.
Prian (Corn.), soft white clay.
Prill (Corn.). the best ore after cobbing.
Pryan (Corn.), see Prian.
Pulp, crushed ore, wet or dry.

Quajado (Span.), dull lead ore.
Quetschwerk (Germ.), ore requiring to be crushed and dressed.
Quick, (1) productive; (2) mercury.
Quillato (Span.), carat.

Rabban (Corn.), yellow dry gossan.
Rack (Corn.), a stationary buddle.
Raffain (Corn.), poor ore.

Rag burning (Corn.), the first roasting of tin-witts.
Ragging (Corn.), rough cobbing.
Rake (Corn.), a vein ; (Derb.), fissure vein crossing strata.
Ramble (N. Eng.), shale bed overlying coal.
Red rab (Corn.) red slaty rock.
Reliz (Span.), wall of lode.
Rick (N. Amer.), open heap in which coal is coked.
Riffle, a groove or check on the floor of a sluice to catch gold.
Rimrock, bed rock forming a boundary to gravel deposit.
Rosiclara (Span.), ruby silver ore.
Roughs (Corn.), second quality tin sands.

Scovan (Corn.), a tin lode showing no gossan at surface.
Scove (Corn.), purest tin ore.
Scrin (Derb.), a small vein.
Scrowl (Corn.), loose ore where a vein is crossed.
Seam (Corn.), a horse-load of ore.
Shadd (Corn.), rounded fragments of ore overlying a vein.
Shet (Staff.), fallen roof of coal mine.
Shoad (Corn.), see Shadd.
Shoading (Corn.), prospecting.
Sickening, see Flouring.
Siddle, inclination.
Skimpings (Corn.), the poorest ore skimmed off the jigger.
Skip, a box for raising ore.
Skit (Corn.), a pump.
Slack, small coal.
Sleck (Derb.), mud in a mine.
Sleeping-table (Corn.), a buddle.
Slickensides, polished surfaces of vein walls.
Slimes, most finely crushed ore.
Sludge, see Slimes.
Sluice, a long channel in rock or of timber, with checks to catch gold.
Slurry (N. Wales), half smelted ore.
Smeddum, lead ore dust.
Smut (Staff.), soft bad coal.
Sollar (Corn.), platform, landing.
Spall, to break ore for dressing.
Speiss (Germ.), impure arsenides produced in copper and lead smelting.
Spiegeleisen (Germ.), manganiferous white cast iron.
Spills (Corn.), a temporary lagging driven ahead on levels in loose ground.
Squat (Corn.), tin ore mixed with spar.
Standage, pump reservoir.
Stannary, tin works.
Stope, to excavate mineral in a series of steps.
Stowce, (1) windlass; (2) landmarks.

Strake, an inclined table or trough for separating metal from refuse.
Studdles, timber props.
Stull, platform to carry miners or waste.
Stup, powdered coke or coal mixed with clay.
Sturt, a tribute bargain profitable to the miner.
Stythe (N. Eng.), choke-damp.
Sweeping-table, a stationary buddle.
Swither (N. Amer.), a crevice branching from a main lead lode.

Tailings, the refuse flowing from the tail or lowest end of the apparatus.
Teem, to pour or tip.
Tepetate (Span.), rubbish.
Thill (N. Eng.), floor of coal mine.
Thurl (Staff.), to cut through from one working into another.
Ticketing (Corn.), purchasing ore by tender on tickets.
Tierras (Span.), earth impregnated with mercury ore.
Tin-witts (Corn.), product of first dressing of tin ores, containing also wolfram and sulphides.
Tossing, shaking powdered ore in water to effect separation of heavy and light particles.
Trapiche (Span.), a primitive grinding mill.
Treloobing (Corn.), stirring tin slimes in water.
Tributers, miners paid by results.
Turbary, a peat bog.
Tut-work, work paid for by the piece, not by results.
Tye, a strake.

Van, to dress or concentrate ore by hand or machine.
Vend (N. Eng.), total sales of coal from a mine.
Vestry (N. Eng.), refuse.
Vinney, copper ore with green efflorescence.
Vugh (Corn.), a cavity.

Wale (N. Eng.), hand-dressing coal.
Wash dirt, auriferous gravel.
Weeldon, old ironstone workings.
Whim, a winding drum.
Whip, a winding pulley.
Whits, see Tin-witts.
Wild lead, zinc blende.
Winze, interior shaft connecting levels.
Wythern (Wales), lode.

Yellow ore (Corn.), chalco-pyrite.

ADDENDA TO GLOSSARY.

Amang (Malaya), titaniferous sands caught in tin sluices.

Biji (Malaya), alluvial tinstone.

Cawk (Derb.), massive barytes.

Dogger (Yorks.), (1) seams of shale containing ferruginous nodules; (2) the nodules themselves.

Ground-sluicing (Klondike), when gravel loosened and moved by hydraulic jet needs second handling to bring it into the sluice-box.

Hydraulicing, working auriferous gravel beds by hydraulic power, the same jet breaking down and carrying gravel into sluice-box.

Karang (Malaya), tin pay-dirt.

Lampan (Malaya), a ground-sluice.
Lombong (Malaya), open-cast tin mine.

Maggie (Som.), long drying-trough for preparing fullers' earth.

Scar (Yorks), large masses of iron ore fused together by over-firing.

Tiff (Amer.), barytes.
Tirring (Scot.), overburden of limestone beds.

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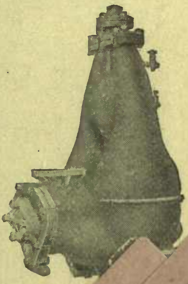

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