THE BLASTING OF ROCK

IN

MINES, QUARRIES, TUNNELS

ETC.

A. W. DAW & Z. W. DAW

ROCK BLASTING

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MINES, QUARRIES, TUNNELS

ETC.

A SCIENTIFIC AND PRACTICAL TREATISE FOR

THE USE OF ENGINEERS AND OTHERS ENGAGED IN MINING, QUARRYING, TUNNELLING, &c., AND FOR MINING AND ENGINEERING STUDENTS

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THE PRINCIPLES OF ROCK BLASTING AND THEIR GENERAL APPLICATION

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PREFACE

TO

THE SECOND EDITION.

In this Edition there has been added miscellaneous matter in the Addenda relating to the following subjects: Crater Forms and the Influence of Flexure thereon; Prof. Höfer's Theory of Blasting and Military Mining; Relation of Explosive Force to Resistance of Rock; The Best or Most Economical Length of Borehole Charge; Safety Explosives; and a Special Article to demonstrate the wrong premises on which Prof. Höfer's Theory of Blasting is based.

THE AUTHORS.

It is a remarkable fact that the theories of rock blasting which have been generally adopted, do not take into account the influence of the form of the chamber employed, seeing that the initial force of the blast, and the resistance of the rock will largely depend thereon for any direction of the line of resistance.

The methods by which the rules and formulæ were deduced are fully explained, and examples of all the more important calculations are given to assist the engineer to deal with any question that may arise in practice, especial attention being directed to how the greatest economy may be attained in the boring of holes and consumption of explosive.

The subject may be appropriately divided into two parts, viz. (1) the principles and their general application, and (2) appliances for drilling the shot-holes, and methods of blasting in mines, quarries, tunnels and subaqueous operations. We have therefore treated it under these headings, this volume comprising the first part, whereas the second part will be published in a second volume which is in preparation.

Valuable practical information is given on the most useful and economical explosives, and on detonators, electric fuses and electric exploders, for which we have much pleasure in acknowledging our obligations to Messrs. Nobel's Explosives Co., Limited; Messrs. The Cotton Powder Co.; Messrs. Curtis and Harvey; and Messrs. Siemens Bros. and Co., Limited. We are also greatly indebted to Messrs. Bickford, Smith and Co. for information respecting their fuses.

The information on electric blasting agrees with the results obtained in our experience, and will, we believe, be found very useful.

A number of Tables are added to facilitate the calculations.

The present volume, though only the first instalment of the work, is complete in itself, and we believe it will be found to give the essential information for carrying out economical and systematic blasting operations. To render it more useful an Index is added which has been carefully prepared.

It will be a source of great pleasure and satisfaction to us if we have accomplished the purposes for which this work was undertaken, namely, to give the Engineer, Miner and Quarryman a correct theory of rock blasting as well as a useful counsellor in questions of application; to the teacher of the science a serviceable text-book for instruction; and to the student of mining and quarrying a welcome aid in the study of blasting.

Albert W. Daw.

Zacharias W. Daw.

CONTENTS.

CHAPTER I.

PRELIMINARY REMARKS.

CHAPTER II.

ON THE RESISTANCES IN ROCK BLASTING.

CHAPTER III.

FORCE DEVELOPED BY A BLAST.

CHAPTER IV.

WEIGHT OF CHARGE REQUIRED TO EJECT ROCK AFTER RUPTURE.

CHAPTER V.

RELATIONS OF THE DIAMETERS OF BOREHOLES AND SPHERICAL CHAMBERS TO LINES OF RESISTANCE.

CHAPTER VI.

ON THE MAXIMUM DISTANCE APART THAT SIMILAR SHOTHOLES, WHEN IN LINE PARALLEL TO A FREE FACE, WILL DISLODGE THE WHOLE OF THE ROCK BETWEEN THEM, WHEN FIRED SIMULTANEOUSLY, THE LINE OF RESISTANCE FOR EACH HOLE BEING THE SAME AS IF IT WERE TO BE FIRED INDEPENDENTLY, AND THE LINE OF RESISTANCE FOR TWO OR MORE SHOTHOLES SUPPORTING EACH OTHER.

CHAPTER VII.

QUANTITY OF ROCK WHICH WILL BE LOOSENED UNDER THE USUAL CONDITIONS OF BLASTING OPERATIONS WHEN THERE ARE NO JOINTS OR FISSURES.

CHAPTER VIII.

THE LENGTH OF CHARGES IN BOREHOLES FOR RUPTURE BY SHEARING.

CHAPTER IX.

THE BEST POSITION FOR A CHAMBER OR CHARGE, WHEN THERE ARE TWO OR MORE FREE FACES AT RIGHT ANGLES TO EACH OTHER.

CHAPTER X.

BOREHOLE CHARGES.

CHAPTER XI.

THE INFLUENCE OF FISSURES, JOINTS AND BEDDING PLANES IN DETERMINING THE CHARGE.

CHAPTER XII.

BLASTING IN CUTTINGS, STOPES OR QUARRIES.

CHAPTER XIII.

THE PLACING OF SHOTHOLES WHEN THERE IS ONLY A SINGLE EXPOSED SURFACE FOR ATTACK, AND NUMBER OF SHOTHOLES REQUIRED FOR A HEADING OR SHAFT.

CHAPTER XIV.

HOW TO FIND THE COEFFICIENT OF ROCK C_a AND CHARGING COEFFICIENT C_v BY TRIAL SHOTS.

CHAPTER XV.

THE TAMPING OR STEMMING OF SHOTHOLES.

CHAPTER XVI.

ON THE DIFFERENT METHODS OF ARRANGING BOREHOLES IN DRIVING AND SINKING.

CHAPTER XVII.

SAFETY FUSE.

CHAPTER XVIII.

ELECTRIC SHOT FIRING.

CHAPTER XIX.

ON EXPLOSIVES AND THEIR SELECTION FOR ROCK BLASTING.

CHAPTER XX.

INSTRUCTIONS FOR THE USE OF EXPLOSIVES.

CHAPTER XXI.

CHAPTER XXII.

EXAMPLES OF ALL THE MORE IMPORTANT CALCULATIONS THAT ARE LIKELY TO OCCUR IN THE DAILY PRACTICE OF ROCK BLASTING, AND OF THE USE OF THE TABLES FOR FACILITATING THE CALCULATIONS.

TABLES.

a

THE

PRINCIPLES OF ROCK BLASTING

Errata.

Page 8, end of line 16, *for* "points" *read* "point."
,, 22, line 14, *for* "$1\frac{1}{2}$ sin *b*" *read* "$1\frac{1}{2}$, sin *b*."
,, 58, last line, *for* "$EFGH \times 2m + 6d + 4k$" *read* "$EFGH = 2m + 6d + 4k$."
,, 59, line 14, *for* "$NS = N(2qd + 2qk = 2qm - 2qk$" *read* "$NS = N(2qd + 2qk) + 2qm - 2qk$."
,, 64, line 6, *for* "$V = W^3 + \frac{m}{2} W$" *read* "$V = W^3 + \frac{m}{2} W^2$."
,, 85, line 20, *for* "$A = C_a$, S W" *omit comma and read* "$A = C_a$ S W."
,, 100, first line, *for* "axes" *read* "axis."
,, 165, lines 15 and 16, *omit* "And fuses in position."
,, 182, line 17, *for* "By filling the shotholes in (*c*) with water" *read* "By filling the shotholes in (*b*) with water."
,, 202, line 17, *omit comma between words* "ammonia, class."
,, 215, ,, 19, *for* "whole" *read* "hole."
,, 223, ,, 20, *omit the word* "chamber."
,, 223, ,, 26, *for* "C" *read* "C_a."
,, 224, ,, 10, *for* "C" *read* "C_v."
,, 224, fourth line from bottom, *for* "moreover that the charging coefficient will" *read* "moreover that the charging coefficient for the volume of rock blasted will."
,, 225, second line from bottom, *for* "$M = T - D$" *read* "$M = D - T$."
,, 228, last line, *for* "6 feet apart" *read* "6 inches apart."
,, 229, line 13, *for* "1·00 feet" *read* "1·00 foot."
,, 229, ,, 14, *for* "6 feet apart" *read* "6 inches apart."
,, 229, third line from bottom, *for* "or 20·9 less boring" *read* "20·9% less boring."
,, 230, line 5, *for* "chamber" *read* "rock."
,, 232, ,, 9, *omit the word* "chamber"
,, 233, ,, 10, *omit the word* "chamber."
,, 235, ,, 7, *for* "or as this may be put" *read* "or as *e* W may be put."
,, 246, under coefficients C_a, *for* "24" *read* "·024."

quantity of powder to be used in blasting, direction of the holes, &c., which theoretically are all very

THE PRINCIPLES OF ROCK BLASTING AND *THEIR GENERAL APPLICATION.*

CHAPTER I.

PRELIMINARY REMARKS.

1. *Rock Blasting* is the science of splitting or loosening rock by means of explosives applied in holes or chambers in the rock.

2. *Failure of previous Rules for Rock Blasting.* Many books have been written on the science for the guidance of the practical man, but it may be said that they all fail to give the most essential information for the determination of the size and position of the chambers and weight of charge. With regard thereto, Mr. George F. Harris, F.G.S., in his work on 'Granite and our Granite Industries,' says:

" In reading works on the subject one frequently sees a great deal about rules for determining the quantity of powder to be used in blasting, direction of the holes, &c., which theoretically are all very well; but the most of them in practice will not work. They nearly all assume that the rock to be blasted is a firm solid body without any cracks, &c., whilst the peculiar conditions in which holes would have to be bored to follow out the rules would waste too much time, and cost too much money to be of real advantage. As a matter of fact, before a hole is bored for the blast, a good quarrymaster looks at the block to be removed, and endeavours to find all cracks and joints. He then sees whether the mass has to be blown up the bed or against it, and observes the manner in which it may be wedged in by other blocks. After mature consideration he instructs the men under him to bore the hole, and estimates the quantity of powder to be used, not by the depth of the hole, but by judging the amount of force required to move the block. Experience has taught him how to do this."

Now, the quarryman's experience is the ascertained result of series of trials and experiments; that is to say, he knows approximately the best position and size of hole, and the quantity of explosive required for a given blast, from the results of similar shots, without, perhaps, much knowledge of the fundamental principles governing the different conditions which obtain in blasting, but which, if understood by him, should enable him to attain still greater economy in his work.

Such, then, being the result of experience, it becomes most important to study the principles involved more carefully in order to ascertain wherein the cause of the erroneous results consists. In the present work, therefore, rules or formulæ are worked out on well-known mechanical principles so as to be applicable to all the varied conditions of the rock and charge, and by means of which any question as to the size and position of chamber and quantity of charge to be employed may be answered; whilst the causes of the failure of the rules given in works on the subject are amply demonstrated. A great deal of useful information is also given regarding explosives and fuses, and the operations of rock blasting.

3. *The Operations of Rock Blasting* consist (1) in boring, mining or excavating suitable holes or chambers in the rock to be blasted; (2) in inserting a charge of some explosive compound therein; (3) in filling up the whole or part of the remaining portion of the hole with suitable material; (4) in igniting or detonating the charge to cause its explosion. The second operation is called charging, the third tamping, and the fourth firing.

4. *Effect of a Blast.*—By the explosion of the charge there is a sudden development of gas of high tension, which exerts a great pressure or shock upon the walls of the chamber, and if such pressure or shock is greater than the resistance of the rock to rupture, the rock is loosened or projected according to the strength of the charge.

5. *Conditions that influence a Blast.*—From experience and theoretical considerations we learn that the effect of a blast may be influenced by—

(*a*) The shape in which the rock is presented, or the size and number of free faces.
(*b*) The tenacity or cohesive strength of the rock.
(*c*) The structure of the rock as to whether it is laminated, stratified or fissured, and the position, direction and number of the joints.
(*d*) The strength and nature of the explosive compound.
(*e*) The size and form of the chamber.
(*f*) The character of the fuse, and tamping.
(*g*) The thermal conductivity of the rock.
(*h*) Whether the blast is to act alone or simultaneously with or following others.
(*i*) The angle of the line of resistance with the horizon.
(*j*) The specific gravity of the rock.

6. *Form of Cavity produced in Homogeneous Rock.*—In Fig. 1, the crater *a b c* represents the general form of cavity produced by the action of a blast at a point *b* in homogeneous rock when there is only one free face *a c*, the angle *a b c* varying generally between the limits 90°–120°, according to the structure of the rock and strength of charge employed. The form of the cavity, as will be explained further on, depends on the form of the chamber, or

the projection of its pressure surface at right angles to the line of resistance. For instance, if this is circular the crater formed by the blast will be the frustum of a cone; and if a square, the frustum of a pyramid —that is, if not modified by any irregularities (joints or fissures) in the structure of the rock, or there being more than one free face. A free face is the exposed surface of any one side of a mass of rock.

7. *Quarrying of Rock.*—Experience shows that rock is most economically and conveniently mined

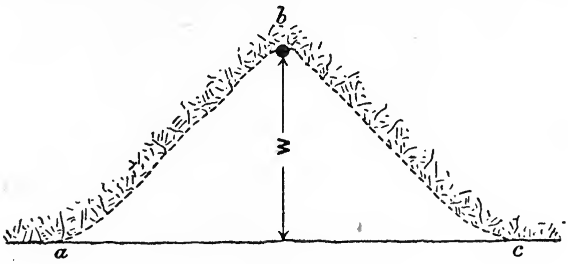

FIG. 1.

or quarried in steps or benches with straight and vertical walls, and that the height of such steps depends on the depth and diameter adopted for the boreholes.

8. *Formula for Determination of Charge.*—According to works on rock blasting, the calculation of the charge required at *b*, Fig. 1, for a line of resistance W should be made according to the following formula:—

$$L = C\,W^3,$$

in which L represents the weight of charge, W the shortest distance from the charge to the free face, and C the charging coefficient. A necessary condition, however, for the application of this formula is that the chamber be properly proportioned to the resistance of the rock.

9. *Previous Theories of Rock Blasting.*—As the basis for the above formula, the following theories are given by Andrée and Guttmann :—

(*a*) "As homogeneous matter varies as the cube of any similar line between them, charges of explosive capable of producing the same effects are to each other as the cubes of the line of least resistance."—*Andrée.*

(*b*) "When a charge is concentrated at a mathematical point in the centre of an unlimited and easily compressible mass its conversion into gas by firing will enlarge the space originally occupied by the explosive into a spherical cavity, and hence it follows that the quantity of a concentrated charge has a direct ratio to the sphere affected by the explosion ; or as spheres vary as the third power of their ratio, and the line of least resistance in rock may be taken as proportional to the radius of explosion, it therefore follows that the charge, under like conditions, will vary as the third power of the line of least resistance.

"Extended charges may be considered as an interrupted series of concentrated charges each of which will have its own sphere of action, and as these spheres intersect and reinforce one another the cavity produced by their continued action will be ellipsoidal, the mutual reinforcement being greatest at the centre of the charge."—*Oscar Guttmann.*

10. *Objections to the above Theories.*—With regard to these theories, there appear at once the following three serious objections to them: First, they do not take into account the cohesive strength or principal resistance of most rocks; secondly, they entirely neglect the influence of the size and form of the chamber on which the initial force of the explosive and the resistance of the rock to rupture depend; and thirdly, they ignore the fact that the resistance due to the mass is affected by the direction of rupture; for if from above downward the weight of the mass will be a force assisting rupture, and the reverse if from below upwards, or when directly opposed to gravity. Excepting, therefore, as stating a single relation of the charge to the resistance, which is true under special conditions, these theories may be said to be totally misleading.

CHAPTER II.

ON THE RESISTANCES IN ROCK BLASTING.

11. There are the following three distinct resistances to a blast:—

(*a*) A resistance due to the cohesion of the rock.

(*b*) A resistance due to the mass or weight of the rock.

(*c*) A resistance due to the hanging of the rock loosened by the blast along the lines of fracture.

12. *Influence of Mode of Application of Force.*—The force required to produce rupture of a given section of rock, as, for instance, the section *a b c* (Fig. 1), will depend on the mode of application of the force, as, according to the laws of mechanics, rupture will take place by shearing when the points of application of the force coincide with the surface of separation, and, on the contrary, by flexure when the arm of the force is of sufficient length to allow of a bending of the mass, and the force is not very suddenly applied.

13. *Force of an Explosion in a Chamber conducive to Rupture by Shearing.*—If we suppose Figs. 2 and 3 to represent the section of a mass of rock with a

free face A B, and to contain a short cylindrical chamber C D, having a charge filling the chamber, and such charge to be exploded, the gases thereby

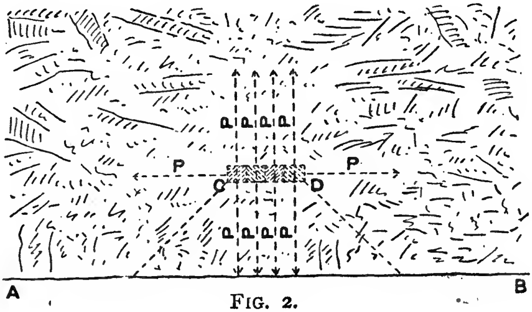

FIG. 2.

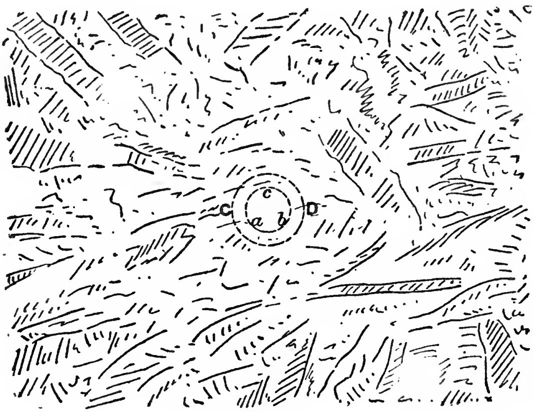

FIG. 3.

generated will develop a number of equal forces P acting on each unit of the surface of the chamber, and each on the side of the free face A B, tending to rupture the rock in the direction of the free face

A B, whereas the pressure of the gases on the other sides are neutralised by the resistance of the rock. The centre of pressure of these forces is the circular line *a b c* (Fig. 3), whose radius is two-thirds of the radius of the chamber C D, one-third of which may be taken as the arm of the force; as experience shows that the line of fracture produced by a blast in homogeneous and compact rock, when there is only one free face, invariably commences at the limit of the surface on which the gaseous pressure is exerted. The arm of the total force acting on the centre of pressure is, therefore, very short for rupture by flexure, and as it is still shorter for any other form of chamber offering the same pressure area to the gases, the conditions are evidently more favourable for rupture by shearing than by flexure, the shearing action of the force being also promoted by its sudden application and the inelastic nature of rock.

14. *Force required to produce Rupture by Shearing.*—According to the laws of mechanics, if rupture takes place by shearing and S denotes the periphery of the chamber, W the line of resistance, and K_1 the modulus of shearing, we can put for the force P required to produce rupture

$$P = S\,W\,K_1.$$

15. *Experiments on Resistance of Ice to Rupture.*—The formula $P = S\,W\,K_1$ might be verified by bursting specimens of homogeneous rock by

mechanical means; but as we may assume that the laws governing the resistance of one inelastic body are the same as for another, it is advantageous to substitute ice for this purpose, as it is homogeneous and easily worked into any desired form. In accordance with this view, we have experimented with ice with the mean results tabulated below. Blocks varying from 4 to 8 inches thick were taken

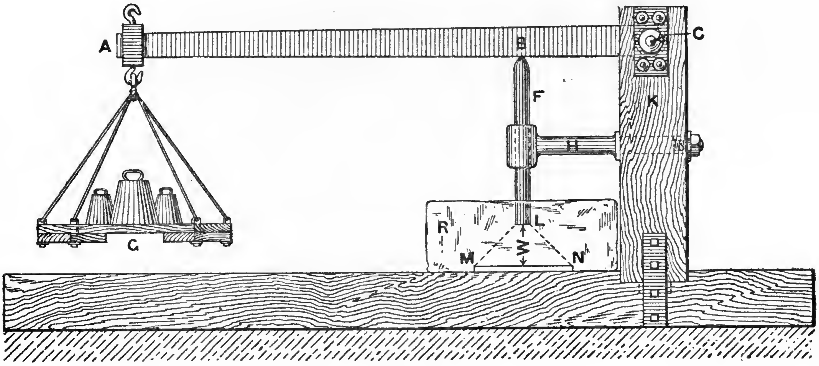

FIG. 4.

for the experiments, which were carried out with the apparatus illustrated in Fig. 4.

In Fig. 4, A B C is a steel lever of the second kind, pivoted at C to the bracket K, and having a scale G, a steel bar F working in the guide H for transmitting pressure to the block of ice R, this arrangement being adopted to ensure a perpendicular pressure being kept on the block at L; M N the free face, and W the thickness of ice ruptured.

The size of the bar at L, thickness of ice ruptured, and size and form of the free face were varied as given in the table. The weight due to the lever was determined by weighing and calculation.

In the experiments very homogeneous ice without any apparent flaws was used, and every care was taken to insure the proper working of the testing apparatus, and that the means of measuring the force exerted in producing rupture was correct. During the experiments the temperature varied between $+ 2°$ and $- 5°$ Celsius, which may have affected the relative accuracy of the results slightly, but we think to a much smaller extent than any difference in the structure of the ice itself.

The very satisfactory agreement between the results obtained for the modulus of rupture K_1, found by dividing the pressure required to produce rupture by the product of the thickness of ice ruptured and periphery of the pressure surface, establishes, we think, beyond question the theory that rupture takes place by shearing, as the slight discrepancies in the results may be taken as due to irregularities in the structure and cohesive strength of the ice used for the experiments, and inaccuracies due to faulty arrangement, friction, &c., of the apparatus. Further experiments are necessary to ascertain the influence of the angle of fracture, which seems to affect the resistance to rupture slightly. In rock, the angle of fracture is affected

TABLE OF EXPERIMENTS ON RESISTANCE OF ICE TO RUPTURE.

Number of Experiments.	Number of Free Faces.	Shape of Free Face.	Diameter or length and breadth of Free Face.	Diameter or section of bar F.	Thickness of ice ruptured W.	Product of thickness of ice ruptured and periphery of bar F. S × W.	Form of Cavity produced.	Mean pressure required to produce rupture P.	Modulus of rupture K_1 obtained by dividing pressure applied by product of thickness of ice ruptured and periphery of bar or pressure surface.
			inches	inches	inches			kg.	
6	1	Circular	3	1	1	3·14	Frustum of cone	167·1	53·22
6	1	,,	3	1	1½	4·71		253·8	53·88
6	1	,,	3	1	2	6·28	,,	338·2	53·85
6	1	,,	3	1	2½	7·85	,,	422·3	53·80
6	1	,,	4	1	1	3·14	,,	157·7	50·22
6	1	,,	4	1	1½	4·71	,,	229·2	48·66
6	1	,,	4	1	2	6·28	,,	315·8	50·29
6	1	,,	4	1	2½	7·85	,,	393·7	50·15
6	1	,,	4	1	3	9·42	,,	473·8	50·30
4	1	,,	6	2	2	12·56	,,	610·6	48·62
4	1	,,	6	2	3	18·84	,,	910·7	48·34
4	1	,,	6½	2½	2	15·70	,,	744·0	47·40
3	1	Rectangular	3 × 6	3 × ½	1½	10·50	Frustum of pyramid	558·0	53·14
3	1	,,	3 × 6	3 × ½	2	14·00		734·0	52·43
3	1	,,	3 × 6	3 × ½	3	21·00	,,	1097·7	52·28

by the greater facility with which it cleaves in one direction than in another. The weight of the ice ruptured is evidently very small, and may be neglected.

Calling the thicknesses of ice burst W, W_1, W_2, W_3, . . . W_n, the bursting pressures P, P_1, P_2, P_3, . . . P_n, and the peripheries of the pressure surfaces, S, S_1, S_2, S_3, . . . S_n, we have, according to the above table, very nearly

$$\frac{P}{S \times W} = \frac{P_1}{S_1 \times W_1} = \frac{P_2}{S_2 \times W_2} = \frac{P_3}{S_3 \times W_3} = \frac{P_n}{S_n \times W_n} = K_1$$

which agrees with the formula for shearing.

16. *Similarity between Cavities produced by Sudden and Gradual Application of Force.*—An important point to note in connection with these experiments is that the form of cavity produced by the application of gradual pressure is similar to that obtained by blasting in rock under similar conditions of free face and pressure surface, it being influenced in both cases by the form of the pressure surface at right angles to the direction of rupture; for instance, in homogeneous rock with one free face, a concentrated charge will produce a conical cavity, and an extended one an elongated trough.

17. *Force required to overcome the Cohesive Resistance of Rock when there is one or more Free Faces.*—In Figs. 5 and 6, plan and section, let M, M_1, N_1, N, represent the surface acted upon by a

blast in homogeneous rock; then, if A B is a free face parallel thereto, M, M_1, N, N_1, E, E_1, F, F_1, will be

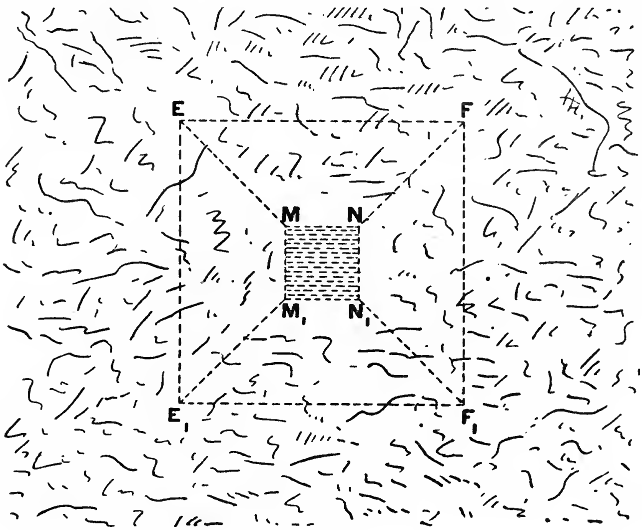

FIG. 5.

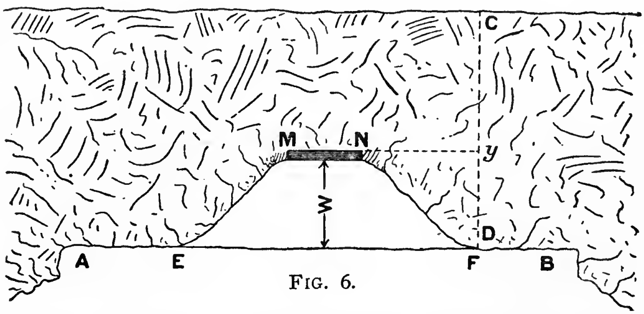

FIG. 6.

the limits of the rock loosened, if the area of the surface M, M_1, N, N_1, and the line of resistance W are properly proportioned to the strength of the explosive. (It should be remarked that the corners at E, E_1, F, F_1, will form a more or less irregular curve instead of right angles, as shown in the plan.) If we denote the periphery of the chamber M, M_1, N, N_1, by S, the forces producing rupture by P, and the distance between the chamber and the free face $A\ B$ by W, it is evident from the foregoing that the relations of these quantities will be expressed by the formula $P = S \times W \times K_1$. If there are two free faces, as in the case of a lateral free face along F, F_1, Fig. 5, perpendicular to the free face $A\ B$, as shown by the dotted line $C\ D$ in Fig. 6, the formula $P = S \times W \times K_1$ will also obtain if we so regulate the distance of the chamber M, M_1, N, N_1, from the free face $C\ D$, that the force required to produce rupture on the side $C\ D$ is approximately equal to the force required to produce rupture on each of the sides E, E_1, E, F, and E_1, F_1.

It may happen that such equilibrium of the resistance of the rock on each side of the chamber exists when the free face $C\ D$ coincides with the limit of fracture F, F_1, on the free face $A\ B$. The influence of the free face $C\ D$ in this case causes the detachment of the section of rock $N\ \gamma\ D$, with the mass $E\ M\ N\ F$ (Fig. 6). If there be free faces along the other limiting lines of fracture $E\ E_1$, $E\ F$, or $E_1\ F_1$, parallel to the line of least resistance, the rock will be similarly fractured on each of these sides, as explained for the case of a free side along the limiting line of fracture $F F_1$. Therefore, for the same chamber in a projecting mass of rock with five free faces as represented in Figs. 7 and 8, if the line of resistance W is proportioned as above described, and the distances $M \gamma_1$, $N \gamma_2$, $N \gamma_1$, and $N_1 \gamma_3$, are not

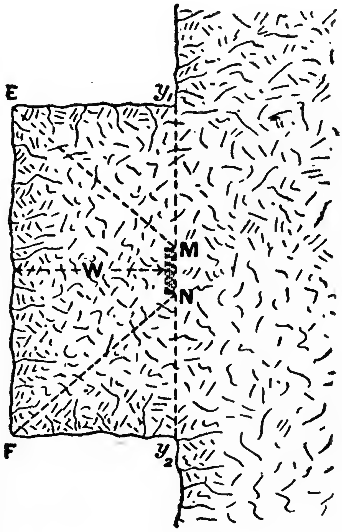

FIG. 7.

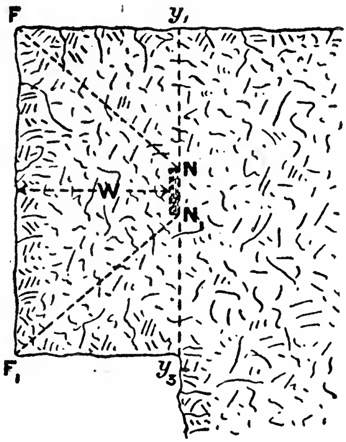

FIG. 8.

greater than W, the mass will be ruptured along the section $\gamma_1 \gamma_2$, instead of the lines E M N F.

Another case is that of a block of rock detached on all sides as in the case of a freestone (Fig. 9), in which the sides $E E_1$, $E_1 F_1$, $F F_1$ and $E F$ correspond to the limiting lines of fracture in the cases before mentioned; hence the product of the line of least resistance and the periphery of the chamber is also a measure of the resistance to rupture when there are six free faces.

The formula $P = S \times W \times K_1$ is therefore applicable to any number of free faces on the principle that the position of the charging chamber should be

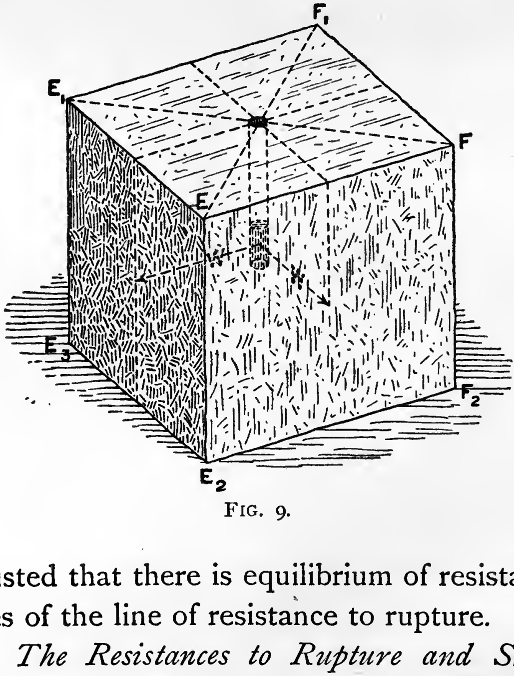

FIG. 9.

so adjusted that there is equilibrium of resistance on all sides of the line of resistance to rupture.

18. *The Resistances to Rupture and Shearing may be equalised.*—It is important to note that, owing to the inelastic nature of rock and the sudden application of the force, equal tension is produced in the rock, parallel to the line of resistance for any section that may be blasted, and that the resistance to rupture of the cross section $\gamma_1 \gamma_2$ (Fig. 7) may

be equal to the resistance to shearing under certain conditions which will be explained in the context. Assuming the rock to be absolutely inelastic, it is evident that we may put

$$R = F K$$

for the resistance of such cross section if F represent the area of the same and K the modulus of rupture of the rock; and that we shall have $F K = S W K_1$ when R is equal to the resistance to shearing.

19. *The Section of Rock which may be ruptured is proportional to Periphery of Chamber for a given Line of Resistance.*—For a given line of resistance W, if the periphery of the chamber M N be S, the force required to produce rupture by shearing is

$$P = S W K_1;$$

and if the periphery of projection of the chamber be S_1, the force P_1 required to produce rupture is

$$P_1 = S_1 W K_1;$$

consequently,

$$\frac{P_1}{P} = \frac{S_1}{S}.$$

But as the force P is capable of breaking a cross section $F = C S W$, and therefore a force P_1 a cross section $F_1 = C S_1 W$, C being a coefficient,

$$\frac{P_1}{P} = \frac{F_1}{F} = \frac{S_1}{S}.$$

20. *Economy of Low Explosives.*—Hence it is clear that in homogeneous rock for any given line of resistance, the section of rock which a blast of any given strength will break is directly proportional to the periphery of projection of the charging chamber at right angles to the line of resistance. It is owing to this condition that the low explosives are sometimes employed in rock having comparatively small cohesive strength, or in the case of a mass of rock bounded laterally by free faces or joints, with greater economy than the high explosives. For instance, if the projecting mass of rock (Figs. 7 and 8) has a short line of resistance for the section $y_1 y_2$, then the proper charge of very strong explosive corresponding to such line of resistance applied at the centre of the mass would only break a conical cavity in the centre of the same; whilst an equally powerful charge of a weaker explosive, requiring a larger chamber and having consequently a longer periphery, would break down the whole mass.

21. *Resistance of the Mass after Rupture.*—After the rock is ruptured by the explosion of a charge applied in a suitable chamber therein, the gases produced will expand and enter the cracks and fissures formed in the rock, and the force of the blast will depend on the new surfaces resisting the free vent of the gases to the atmosphere, and the quantity of gases developed, or the weight of the charge. The resistance, on the contrary, will depend on the volume and specific gravity of the

rock to be moved, as also on the direction of movement; for, accordingly as the mass is moved from above downwards or from below upwards, the weight of the mass will be a force assisting or resisting the action of the blast, as in the one case the weight tends to assist, and in the other to resist the blast, such resistance varying as the sine of the angle of direction of movement with the horizon.

22. *Resistance of the Mass or Weight of Rock blasted at any Angle to the Horizon.*—In case the direction of the blast is upwards, we have to consider not only the resistance of cohesion but also that of the mass or weight of rock to be blasted, which is proportional to the product of its weight and the sine of the angle of movement to the horizon. The weight of the rock being G, and the angle of movement above the horizon a, we have

$$R = G \sin a.$$

If the direction of the angle a is below the horizon, the resistance R is turned into a force P, tending to produce rupture, and $P = G \sin a$.

Therefore, for a weight of rock G, and angles a and b of direction of blast above the horizon, the relation of the corresponding resistances R and R_1 (neglecting the resistance of cohesion) is

$$R : R_1 :: G \sin a : G \sin b.$$

$$\therefore \frac{R_1}{R} = \frac{\sin b}{\sin a}.$$

23. *Resistance of the Friction and Hanging of the Rock along the Line of Rupture.*—If there be friction and hanging of the mass of rock after rupture along the lines of fracture, and these resistances be taken as proportional to the mass of rock to be moved, and represented by a coefficient B, we have

$$R : R_1 :: G\,(B + \sin a) : G\,(B + \sin b),$$

whence

$$\frac{R_1}{R} = \frac{G\,(B + \sin b)}{G\,(B + \sin a)} = \frac{B + \sin b}{B + \sin a}.$$

For instance, if 50 per cent. more explosive were required for a vertical than for a horizontal blast (the quantity will depend on the effect to be produced), we shall have $\frac{R_1}{R} = 1\frac{1}{2}$ sin $b = 1$ and sin $a = 0$, and consequently

$$\frac{B + 1}{B} = 1\tfrac{1}{2}, \text{ whence } B = 2.$$

Substituting this value of B in the above formula we get

$$\frac{R_1}{R} = \frac{2 + \sin b}{2 + \sin a}.$$

If, then, we make R = the resistance for a horizontal blast sin $a = 0$, and

$$\frac{R_1}{R} = \frac{2 + \sin b}{2} = 1 + \frac{\sin b}{2}:$$

Calling the charges which will overcome the resistances R and R_1, L and L_1, we have

$$\frac{L_1}{L} = 1 + \frac{\sin b}{2},$$

or

$$L_1 = \left(1 + \frac{\sin b}{2}\right) L.$$

If only 25 per cent. more explosive were required to give the desired effect we should have

$$L_1 = \left(1 + \frac{\sin b}{4}\right) L.$$

And in general, when $\frac{1}{n}$th more explosive is required for a vertical than a horizontal blast,

$$L_1 = \left(1 + \frac{\sin b}{n}\right) L;$$

or, as $L = C_v W^3$ (Chapter III.), we have in general

$$L_1 = \left(1 + \frac{\sin b}{n}\right) C_v W^3.$$

24. *Combined Resistance of the Weight and Cohesion of Rock.*—When the resistance of the weight of the rock as well as the cohesive resistance of the same has to be considered in calculating the size and form of the chamber which, when filled with explosive, will overcome the total resistance to rupture, this is expressed by

$$R = (S \times W \times K_1) + G \sin a,$$

S being the periphery of the chamber, W the line of resistance, K_1 the modulus of shearing of the rock, G the weight of the mass fractured, and a the angle of direction of the blast to the horizon.

Consequently, for any other values of these quantities we can put

$$R_1 = (S_1 \times W_1 \times K_1) + G_1 \sin a_1.$$

But the resistances R and R_1 are proportional to the areas of projections of the chambers at right angles to the line of resistance, and therefore

$$\frac{R_1}{R} = \frac{A_1}{A} = \frac{(S_1 \times W_1) + \dfrac{G_1 \sin a_1}{K_1}}{(S \times W) + \dfrac{G \sin a}{K_1}}.$$

25. *The Resistance of Cohesion of Rock to Rupture for any one Explosive varies as the Square of the Line of Resistance.*—For a line of resistance W and periphery S of blasting chamber, the resistance R to a blast is

$$R = S \times W \times K_1,$$

and for any other line of resistance W_1,

$$R_1 = S_1 \times W_1 \times K_1,$$

K_1 being the modulus of shearing of the rock.

Since, for boreholes whose diameter is d, the length of charge should be $m = n\,d$ (n being a coefficient of the diameter), $S = (2\,n + 2)\,d$; and for boreholes whose diameter is d_1, $S_1 = (2n + 2) d_1$, we have

$$\frac{S_1}{S} = \frac{(2n + 2) d_1}{(2n + 2) d}.$$

Therefore, as $\dfrac{W_1}{W} = \dfrac{d_1}{d}$ (see Chapter V.),

$$\frac{R_1}{R} = \frac{W_1 \times W_1 \times K_1}{W \times W \times K_1} = \left(\frac{W_1}{W}\right)^2;$$

that is, for the same explosive, the resistance of cohesion of rock to rupture in blasting varies as the square of the line of resistance.

CHAPTER III.

FORCE DEVELOPED BY A BLAST.

26. *Conditions affecting the Force of an Explosion.*—In rock blasting, the forces P, P_1, P_2, P_3, &c., in the formula $\frac{P}{S \times W} = \frac{P_1}{S_1 \times W_1} = \frac{P_2}{S_2 \times W_2}$, &c. $= K_1$, are obtained by the ignition or detonation of explosive compounds in closed chambers formed in the rock, whereby the explosive compound is converted from its solid or liquid state into gases in an inappreciably short space of time, this chemical conversion liberating heat and the gases in consequence highly expanding, and through such expansion exerting a great pressure on the rock. The force which is developed by blasting, therefore, depends on the following conditions :—

(*a*) The absolute quantity of the gases produced.
(*b*) The temperature of the gases.
(*c*) The expansion of the gases due to the temperature resulting from the explosion.
(*d*) The time occupied in obtaining the maximum expansion or pressure.
(*e*) The size and form of the chamber.

(*f*) The thermal conductivity of the surrounding medium.

27. *Of Different Action of Explosives.*—According to their properties explosives may be divided into two classes :—

(*a*) Low, or slow and rending.
(*b*) High, or quick and shattering.

The former are those in which the transformation into gas is comparatively slow, the explosive force being exerted by degrees as the gases are developed. The gases from such explosives being slowly evolved, the pressure upon the containing body cannot be much greater in any part than that which is exerted upon the part which yields. Gunpowder is the best type of such explosives.

The latter, on the contrary, are those in which the transformation of the explosive substance into gas occurs practically instantaneously. The full force of the enlarged volume is at once exerted in all directions, and upon every part of the containing body, because motion requires time ; and as no time is allowed for the less resistant part to yield by moving away before the full pressure of the fluid is developed, it follows that the whole force of the explosion is exerted upon its surroundings. Nitroglycerine and guncotton are the most prominent types of this class of explosives.

Between gunpowder and nitroglycerine as extremes the other explosives range according to their strength, and their applicability to rock blasting will depend on the nature of the rock, and also on whether the rock is to be shattered or broken in large blocks, and whether time is of very great importance in carrying out the work, as for instance in most railway tunnels.

28. *Maximum Pressures Developed by Explosives.*—The experiments of Sarrau, Vielle, Noble and Abel give the following as the approximate maximum pressures developed by mercury fulminate, nitroglycerine, guncotton and blasting-powder at their maximum densities :—

Mercury fulminate,	27,000	kg. per square	centimetre.
Nitroglycerine,	12,000	,,	,,
Guncotton,	10,000	,,	,,
Blasting-powder,	6,000	,,	,,

29. *The Useful Work of Explosives*, which consists partly in shattering the rock and partly in displacing the shattered masses, does not approach their theoretical on account of incomplete combustion, the escape of gas through the holes and fissures caused by the explosion at high pressure, and the thermal conductivity of the surrounding medium ; moreover, energy is absorbed by the heating and cracking of the rock which is not displaced. According to von Rziha's experiments the useful effect is only 13·71 per cent., as given in the following table :—

Explosive.	Work in Metre-Kilogrammes.		Relative working value, Powder = 1.
	Theoretical.	Useful.	
Powder containing 62 per cent. saltpetre	242,335	33,224	1·0
Dynamite containing 75 per cent. nitroglycerine ..	548,250	75,165	2·2
Blasting-gelatine containing 92 per cent. nitroglycerine	766,913	105,144	3·2
Nitroglycerine	794,565	108,935	3·3

30. *The Power of an Explosive* cannot be calculated with precision from the quantity and temperature of the gases developed by the detonation or ignition of any explosive compound, owing to a want of knowledge of the state of dissociation of the gaseous products at the moment of explosion and during the period of cooling.

31. *Relative Force developed by an Explosive.*—We may obtain sufficiently accurate relative values of the maximum forces developed in different sizes and forms of chamber for blasting purposes by the aid of the two important laws of the statics of fluids given below if we assume that each unit of the same explosive compound will develop the same quantity of gases, and attain the same maximum pressure, under like conditions.

The two laws of the statics of fluids above referred to are—

(*a*) That the pressure exerted by a fluid upon the different parts of the walls of the containing chamber are proportional to the areas of those parts.

(*b*) That the pressures exerted by a fluid in any direction upon a surface is proportional to the projection of the surface at right angles to the given direction.

Since rock is invariably a very inelastic body, whose limit of elasticity is reached when it has undergone a very slight extension or change of form, it is evident that by the explosion of a charge in a chamber in the rock there will be no appreciable enlargement of the chamber before rupture takes place. Therefore, if M denotes the maximum pressure or shock per unit of surface in a chamber due to the explosion, and A and A_1 projections of two chambers at right angles to the direction of rupture, we can put for the absolute forces or pressures P and P_1 acting in the direction of rupture,

$$P = MA, \quad \text{and} \quad P_1 = MA_1,$$

whence

$$\frac{P_1}{P} = \frac{A_1}{A};$$

that is, the forces are directly proportional to the area of the projections; hence, for the use of the same explosive compound, the projection of chambers at right angles to their lines of resistance may be taken to represent the relative forces which will be developed by the explosion of charges filling the chambers.

32. *Condition necessary for the Development of the Maximum Pressure of an Explosive.*—By experiments it has been proved that the maximum pressure or effect which any explosive substance can develop is that when detonating in a space entirely filled, viz. in a space equal to its own volume. Hence, to obtain the greatest disruptive effect the charge should entirely fill the chamber.

33. *Influence of the Form of Chamber and the Thermal Conductivity of the Rock on the Charge.*—When the blast has only the resistance of cohesion to overcome, as when the direction of rupture is downwards and there is no friction or hanging of the ruptured rock along the lines of fracture, the quantity of charge required will depend largely on the form of the chamber in which it is applied. For instance, a chamber whose cubical contents are $12 \times 12 \times 1$ will give as great pressure area to a charge, and overcome as great resistance in one direction, as another whose cubical contents are $12 \times 12 \times 2$, but the latter will take double the charge of the former to give the same rupturing effect. In the case given, it would therefore appear to be conducive to economy in the use of explosives to use a flattened form of chamber, or to attenuate the charge as much as possible. There is, however,

a limit to this, irrespective of the difficulty of boring such chambers, as a disproportionately large area of walls of the chamber to the quantity of explosive in the charge would cause a great loss of the force of the blast, owing to the thermal conductivity of the rock, which is proportional to the area of the walls of the chamber. Therefore the minimum width of the chamber should not be less than three-quarters of an inch if it be desired to obtain nearly the full effective pressure of the blast.

CHAPTER IV.

WEIGHT OF CHARGE REQUIRED TO EJECT ROCK AFTER RUPTURE.

34. *Ratio of Charge to Mass of Rock to be Moved.* For any explosive compound of uniform strength, theory (see next article) and experience show that when a charge L will remove a mass M, under like conditions a charge 2 L will move a mass 2 M, and a charge n L a mass n M, the masses being of the same specific gravity. If, therefore, for a given direction of movement the charges required for the volumes V and V_1 of a given kind of rock are L and L_1 we have the following relation of these quantities :—

$$L : L_1 : : V : V_1,$$

and consequently,

$$\frac{L_1}{L} = \frac{V_1}{V}.$$

35. *Ratio of Charge to Line of Resistance for similar Masses of Rock.*—The volumes V and V_1, as they are similar in form, are proportional to the cube of any similar line within them, and, there-

fore, if W and W_1 are the lines of resistance corresponding to the volumes V and V_1 we have

$$\frac{V_1}{V} = \left(\frac{W_1}{W}\right)^3,$$

and substituting this value of $\frac{V_1}{V}$ in the formula $\frac{L_1}{L} = \frac{V_1}{V}$ we get

$$\frac{L_1}{L} = \left(\frac{W_1}{W}\right)^3.$$

The above formula agrees with that usually given in most works on rock blasting for estimating charges if we put the coefficient C_v for $\frac{L_1}{W_1^3}$, as we then obtain

$$L = C_v W^3.$$

36. *Theory of the Action and Force of a Blast after Rupture has taken place.*—The pressures, or tensions, and volumes of gas produced by the explosion of a charge in a chamber on the fractured mass *a b c* (Figs. 10 and 11) by the force due to such pressures may be expressed by the law of Mariotte (or Boyle).

According to this law the density of one and the same quantity of gas is proportional to its tension, or pressure; or, since the space occupied by one and the same mass is inversely proportional to the density of the gas, the volumes of one and the same quantity of gas are inversely proportional to their tensions, or pressures.

Assuming, then, that the explosion of one unit of weight of any explosive having a constant chemical composition (when exploded under like conditions and neglecting the thermal conductivity of the chamber) to give a constant volume of gas at

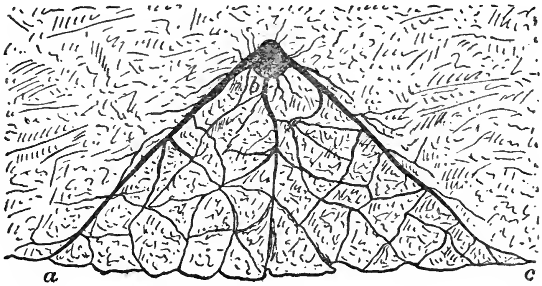

FIG. 10.

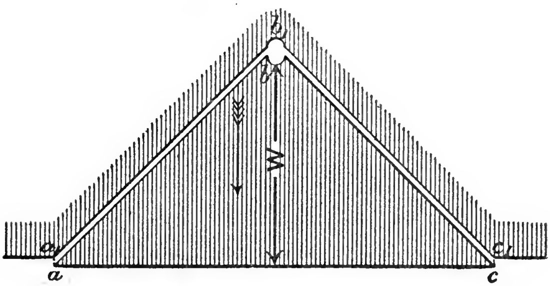

FIG. 11.

a certain tension, two such units to give two such volumes, three such units three such volumes, &c., it is evident, according to the above-mentioned law, that the quantity or volume of gas, or gases, produced by the explosion of a charge is directly proportional to the space occupied by the gas, whereas the pressure on each unit of the walls of the containing chamber is in the inverse ratio of the volume of gas filling the chamber.

The space $V = a\ b\ c\ c_1\ b_1\ a_1$ (Fig. 11), formed by the movement of the fractured mass of rock $a\ b\ c$ in the direction of the line of resistance, is equal to the product of the surface $a\ c = F$ of the mass of rock $a\ b\ c$ and the distance $d = a\ a_1 = c\ c_1$ of such movement, and is as follows:—

For a line of resistance W, θ being a coefficient,

$$V = d\ F = \theta W \times \theta W \times d = \theta^2 \,.\, d \,.\, W^2.$$

For a line of resistance $W_1 = 2\ W$,

$$V_1 = d\ F_1 = 2\,\theta W \times 2\,\theta W \times d = 4\,\theta^2 \,.\, d \,.\, W^2.$$

For a line of resistance $W_n = n\ W$,

$$V_n = d\ F_n = n\,\theta W \times n\,\theta W \times d = n^2\,\theta^2 \,.\, d \,.\, W^2.$$

Consequently,

$$\frac{V_n}{V} = \frac{F_n}{F} = \frac{n^2 \times \theta^2 \times d \times W^2}{\theta^2 \times d \times W^2} = n^2.$$

Or the space V and the surface F increase as the square of the line of resistance W, for the same movement d of the fractured mass $a\ b\ c$.

According to the statics of fluids the pressure exerted by a fluid in any direction is proportional to the projection of the surface at right angles to the given direction; hence the pressure exerted by the gas in the chamber in the direction of the line of resistance W is proportional to the surface $a\ c = F$, or the square of the line of resistance.

Since the volume of gas produced by a blast under the given conditions is proportional to the square of the line of resistance, its pressure is inversely as the square thereof, and for one and the same quantity of charge, supposing the pressure to be unity for a line of resistance = 1, we shall have the following relative pressures per unit of surface for any other lines of resistance.

Line of resistance.				Pressure of gas.
1	..	..	..	1
2	..	..	..	$\frac{1}{4}$
3	..	..	..	$\frac{1}{9}$
4	..	..	..	$\frac{1}{16}$
n	..	..	..	$\frac{1}{n^2}$

But the total pressure of the blast in the direction of the line of resistance is the product of the surface F and the pressure per unit of surface in the chamber, which may be expressed relatively as under :—

Line of resistance.		Relative pressure in direction of blast.
1	..	.. .. $= 1$
2	..	$2 \times 2 \times \frac{1}{4} = 1$
3	..	$3 \times 3 \times \frac{1}{9} = 1$
4	..	$4 \times 4 \times \frac{1}{16} = 1$
n	..	$n \times n \times \frac{1}{n^2} = 1$

Consequently, the total pressure produced by the explosion of one and the same quantity of charge upon the mass of rock *a b c* at any distance from its bed may be taken as a constant quantity until the gases escape to the atmosphere, when the force of the blast is lost and the further projection of the mass is due to the velocity it has attained. The direction of movement of the ruptured mass of rock under the direct pressure of the blast corresponds with the line of resistance, and is indicated by the arrow in Fig. 11.

Since the total pressure developed by a blast upon a mass of rock at any point in its ejection from its bed is a constant quantity for one and the same quantity of charge, for a double quantity of charge at any given distance of the rock from its bed before the gases escape to the atmosphere the total pressure will also be constant, but double that for the single charge; and in like manner for a treble charge it will be trebled, and generally the force of the blast will be proportional to the quantity of charge.

On the contrary, the resistance of a ruptured mass of rock to a blast, when there is no friction or hanging of the same on the sides, is directly proportional to the product of its weight and the sine of the angle of the direction of the blast to the horizon. Therefore, calling the resistance R, the weight of the rock G, and the angle of the blast to the horizon a, we have

$$R = G \sin a.$$

But since we can put $G = C\,W^3$

$$R = C\,W^3 \sin a.$$

Therefore, for any other line of resistance R_1 we have

$$R_1 = C\,W_1^3 \sin a,$$

and consequently

$$\frac{R_1}{R} = \left(\frac{W_1}{W}\right)^3$$

for any given direction of blast.

For R and R_1, substituting the forces P and P_1, required to overcome these resistances, we have

$$\frac{P_1}{P} = \left(\frac{W_1}{W}\right)^3.$$

But it has been demonstrated that the charges must be directly proportional to these forces, and denoting the charges which will develop the forces P and P_1 by L and L_1

$$\frac{P_1}{P} = \frac{L_1}{L}$$

and

$$\frac{L_1}{L} = \left(\frac{W_1}{W}\right)^3.$$

When, therefore, for a given direction of blast L has been found to be the proper charge for a line of resistance W, L_1 will be the proper charge for a line of resistance W_1, conditionally that these charges are inserted in properly proportioned chambers to enable the gaseous pressure developed by their explosion to overcome the cohesive resistance and weight, or inertia, of the rock, and that the masses of rock are similar in form.

37. *Sectional Area of Chamber required at Right Angles to the Line of Resistance.*—The ratio of the resistances R and R_1 to charges in chambers is

$$\frac{R_1}{R} = \frac{S_1 \times W_1 + \dfrac{G_1 \sin a_1}{K_1}}{S \times W + \dfrac{G \sin a}{K_1}}.$$

It is evident that the force of a blast must be made equal to the resistance of the rock, and that we must make

$$\frac{R_1}{R} = \frac{P_1}{P}.$$

Therefore, as $\frac{P_1}{P} = \frac{A_1}{A}$, A and A_1 being the sectional areas of charging chambers, we have

$$\frac{A_1}{A} = \frac{S_1 \times W_1 + \dfrac{G_1 \sin a_1}{K_1}}{S \times W + \dfrac{G \sin a}{K_1}}.$$

The value of $\dfrac{A}{S \times W + \dfrac{G \sin a}{K_1}}$ may be found by trial blasts, and, therefore, if we put the coefficient C_a for this quantity we have, in general,

$$A = C_a (S \times W) + \frac{C_a G \sin a}{K_1}.$$

But as $\frac{C_a G \sin a}{K_1} = 0$ when the direction of blast is horizontal, and it is always a comparatively small quantity, the sectional area of chamber may be calculated from the formula

$$A = C_a (S \times W)$$

in most cases.

The dimensions of the chamber depend also on the volume of charge required and the form of chamber.

38. *Chamber Coefficient.*—From the formula $A = C_a S W$ we have

$$\frac{A}{S} = C_a W = \theta.$$

The value θ may be called the chamber coefficient, as it depends solely on the form of chamber.

For any chamber whose section at right angles to the line of resistance is circular, if the diameter of the section is d, we have

$$A = \cdot 7854\, d^2 \quad \text{and} \quad S = 3 \cdot 1416\, d,$$

whence,

$$\theta = \frac{\cdot 7854\, d^2}{3 \cdot 1416\, d} = \frac{d}{4}$$

and

$$d = 4\,\theta.$$

For any square section of chamber whose side is l,

$$A = l^2 \quad \text{and} \quad S = 4\,l,$$

hence

$$\theta = \frac{l^2}{4\,l} = \frac{l}{4}$$

and

$$l = 4\,\theta.$$

The coefficient for any other form of chamber may be found in a similar manner.

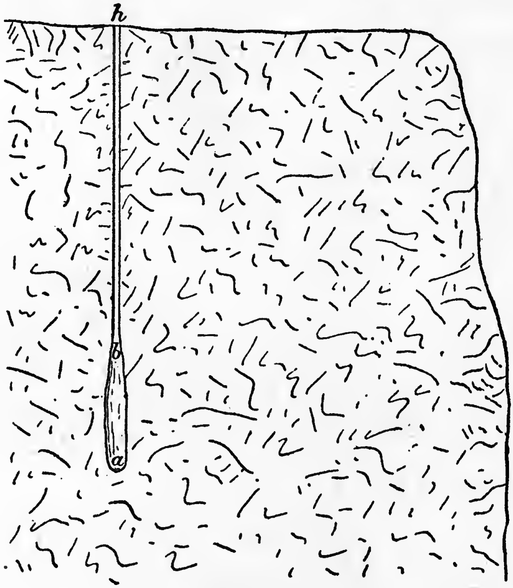

FIG. 12.

By the explosion of a charge in rock the sides of the chamber are corroded by the heat developed, but such corrosion does not affect the chamber coefficient appreciably. Hence, if the diameter of a borehole is too small to enable it to take sufficient explosive to overcome the resistance of the rock to rupture the only effect of the blast will be to corrode the walls of the hole in immediate contact with the charge, so that the diameter of the hole will be enlarged, as illustrated in Fig. 12. A chamber *a b* is therefore produced, which may be further enlarged by repeating the process, until it will take sufficient explosive to rupture the rock. The best results are obtained with the strongest explosive and the use of only a little paper as tamping. If the hole is inclined below the horizon the size of the chamber may be ascertained by measuring the quantity of water required to fill the same. This operation is termed chambering, and if conducted by an experienced man will in some cases give good results.

CHAPTER V.

RELATIONS OF THE DIAMETERS OF BOREHOLES AND SPHERICAL CHAMBERS TO LINES OF RESISTANCE.

39. *Boreholes and Chambers parallel to Free Face.*—For cylindrical and spherical chambers in rock, with the aid of the above enunciated principles, we can deduce very simple relations of the diameters to the lines of resistance, when the direction of rupture is horizontal.

Let l and l_1 be the lengths, and d and d_1 the diameters of two cylindrical chambers or boreholes in rock, which are placed at right angles to the line of resistance or parallel to the free face; then, if A and A_1 are the areas of projection of the chambers parallel to their axes,

$$A = l\,d \quad \text{and} \quad A_1 = l_1\,d_1.$$

Therefore,
$$\frac{A_1}{A} = \frac{l_1\,d_1}{l\,d}.$$

But, as before explained, $\frac{A_1}{A} = \frac{P_1}{P}$, P and P_1 being the forces developed by the charges filling the chambers before rupture takes place; and since $\frac{P_1}{P} = \frac{S_1\,W_1}{S\,W}$, we have

$$\frac{A_1}{A} = \frac{S_1\,W_1}{S\,W}.$$

Substituting for $\frac{A_1}{A}$ the value given above, we get

$$\frac{l_1\, d_1}{l\, d} = \frac{S_1\, W_1}{S\, W}.$$

When, however, the lengths of the chambers are a given multiple of the diameters,

$$\frac{l_1}{l} = \frac{S_1}{S} \quad \text{and consequently} \quad \frac{d_1}{d} = \frac{W_1}{W}.$$

Therefore, in blasting in the same kind of rock when the cohesive resistance is not affected by joints and fissures, the diameters of the boreholes should be directly proportioned to the lines of resistance.

In the case of spherical chambers, whose projections are A and A_1, and diameters d and d_1, we have

$$\frac{A_1}{A} = \frac{\frac{\pi}{4} d_1^2}{\frac{\pi}{4} d^2} = \left(\frac{d_1}{d}\right)^2,$$

and

$$\left(\frac{d_1}{d}\right)^2 = \frac{S_1\, W_1}{S\, W}.$$

But $\frac{S_1}{S} = \frac{d_1}{d}$, and substituting $\frac{d_1}{d}$ for $\frac{S_1}{S}$ in the above equation, we get

$$\frac{d_1}{d} = \frac{W_1}{W}.$$

Consequently the same relations of the diameters to the lines of resistance subsist for spherical as for cylindrical chambers, viz. the diameters should be proportional to the lines of resistance.

By experiments in rock with a number of boreholes, varying in diameter from ¾ to 2½ inches, we have obtained results quite in accordance with the above formula, thus proving its correctness and establishing the principles on which it is based.

With gelatine dynamite in a very homogeneous and strong granite our experiments gave the following results :—

No. of Expt.	Diameter of Borehole.	Depth of Borehole.	Length of Charge.	Weight of Charge.	Line of Resistance.
	inches	ft. in.	inches	lbs.	ft. in.
1	¾	3 2	9	·22	2 4½
2	1	4 2	12	·50	3 2
3	1¼	5 3	15	1·00	4 0
4	1½	6 3	18	1·75	4 9
5	1¾	7 3	21	2·80	5 6
6	2	8 4	24	4·20	6 4
7	2¼	9 5	27	6·00	7 2

Manuel Eissler, in his valuable work on the modern high explosives, mentions that the ordinary mode of calculating charges is not exact, as it does not take into account the diameter of borehole and the whole face, and gives the following table of lines of resistance resulting for boreholes of $1\frac{1}{4}$, $1\frac{1}{2}$ and $1\frac{3}{4}$ inches diameter, supposing No. 3 dynamite to be employed.

No. of Experiment.	Diameter of Boreholes.		
	$1\frac{1}{4}$ in.	$1\frac{1}{2}$ in.	$1\frac{3}{4}$ in.
	Line of Resistance.		
1	$3\frac{1}{2}$ feet	4 feet	5 feet
2	$3\frac{3}{4}$ "	5 "	6 "
3	5 "	6 "	7 "

The depths of the holes given are the following: for No. 1, equal to line of resistance; for No. 2, half as long again as the line of resistance; and for No. 3, double the line of resistance.

As will be observed, the lines of resistance in the above table are proportional to the diameter of the boreholes.

40. *Boreholes Angled to a Single Exposed Free Face.*—If a borehole be placed at a less angle than 90 degrees with the line of resistance, as in Fig. 13, when a single exposed face is to be attacked, we shall have the following relations for the forces P and P_1, tending to produce rupture along the lines of resistance $r\,n$ and $q\,n$ perpendicular to the free face A B and borehole h respectively, if we put m for the length of charge in the borehole, d for the

diameter of borehole, a for the angle of the borehole with the line of resistance $r\,n$, W and W_1 for the lines of resistance along $r\,n$ and $q\,n$ respectively, K_1 for the modulus of shearing, and Q for the maxi-

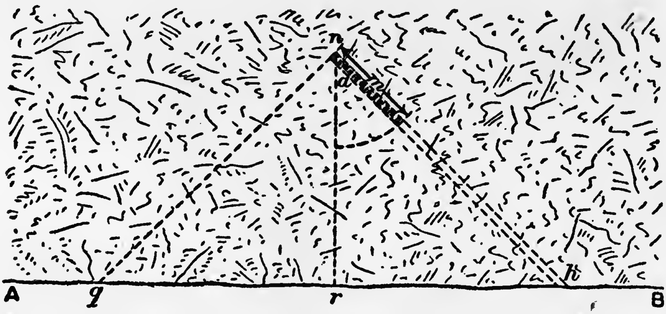

FIG. 13.

mum pressure or shock per unit of surface developed by the explosion of the charge, viz.

$$P = m\,d\,Q \sin a, \quad \text{and} \quad P_1 = m\,d\,Q.$$

For the resistances R and R_1 to the forces P and P_1 we may put

$$R = S\,W\,K_1 = 2\,(m \sin a + d)\,W\,K_1,$$

and

$$R_1 = S_1\,W_1\,K_1 = 2\,(m + d)\,W_1\,K_1.$$

The line of least resistance is determined by whether the ratio of the force to the resistance is greater along $r\,n$ than $q\,n$.

From the above equations,

$$\frac{P}{R} = \frac{m\,d \sin a}{2\,W \times (m \sin a + d)} \times \frac{Q}{K_1}$$

and

$$\frac{P_1}{R_1} = \frac{m\,d}{2\,W_1 \times (m + d)} \times \frac{Q}{K_1}.$$

But $W = W_1 \cos(90^\circ - a) = W_1 \sin a$, and

$$W_1 = \frac{W}{\sin a}.$$

Therefore, by substituting, we get

$$\frac{P_1}{R_1} = \frac{m\,d \sin a}{2\,W\,(m + d)} \times \frac{Q}{K_1}.$$

It is clearly evident that $\dfrac{m\,d \sin a}{2\,W\,(m \sin a + d)}$ is greater than $\dfrac{m\,d \sin a}{2\,W\,(m + d)}$ when $\sin a < 1$, and, consequently, $\dfrac{P}{R}$ is greater than $\dfrac{P_1}{R_1}$, from which we may conclude that there is a greater tendency to rupture along $r\,n$ than $q\,n$; and as it can be similarly demonstrated that there is less tendency to rupture along any other line between r and q, $r\,n$ is the line of least resistance to the blast.

$\dfrac{P}{R} = \dfrac{m\,d \sin a}{2\,W\,(m \sin a + d)} \times \dfrac{Q}{K_1}$ is a measure of the ratio of the force to the resistance when the borehole makes an angle a with the line of least resistance. For any smaller angle b, diameter of hole d_1, length of charge m_1, and line of least resistance W_1, we can put

$$\frac{P_2}{R_2} = \frac{m_1\,d_1 \sin b}{2\,W_1\,(m_1 \sin b + d_1)} \times \frac{Q}{K_1}.$$

Then, if we make $\frac{P}{R} = \frac{P_2}{R_2}$

$$\frac{m_1\, d_1 \sin b}{2\, W_1\, (m_1 \sin b + d_1)} = \frac{m\, d \sin a}{2\, W\, (m \sin a + d)}$$

and

$$\frac{W_1}{W} = \frac{(m\, m_1\, d_1 \sin a \sin b) + (m_1\, d\, d_1 \sin b)}{(m\, m_1\, d \sin a \sin b) + (m\, d\, d_1 \sin a)}$$

or

$$\frac{W_1}{W} = \frac{(m\, m_1\, d_1) + (m_1\, d\, d_1 \operatorname{cosec} a)}{(m\, m_1\, d) + (m\, d\, d_1 \operatorname{cosec} b)}.$$

This formula gives the relation of the lengths of charges, and diameters and angles of holes, for different lines of resistance W and W_1 in rock of the same cohesive strength.

If $m = n\, d$ and $m_1 = n\, d_1$

$$\frac{W_1}{W} = \frac{n\, d_1 + d_1 \operatorname{cosec} a}{n\, d + d \operatorname{cosec} b} = \frac{m_1 + d_1 \operatorname{cosec} a}{m + d \operatorname{cosec} b}.$$

If $W = W_1$ and the hole equal to the resistance W, whose diameter is d, is parallel to the free face, then $\operatorname{cosec} a = 1$, and we have

$$n\, d_1 + d_1 = n\, d + d \operatorname{cosec} b$$

$$(n + 1)\, d_1 = (n + \operatorname{cosec} b)\, d$$

and

$$d_1 = \left(\frac{n + \operatorname{cosec} b}{n + 1}\right) d.$$

If $n = 12$

$$d_1 = \left(\frac{12 + \operatorname{cosec} b}{13}\right) d.$$

When cosec $a = 1$ and $n = 12$

$$\frac{W_1}{W} = \frac{13\,d_1}{(12 + \text{cosec } b)\,d}.$$

And if $d = d_1$

$$\frac{W_1}{W} = \frac{13}{12 + \text{cosec } b}.$$

The formula $\frac{d_1}{d} = \frac{12 + \text{cosec } b}{13}$ gives the ratio of the diameters d and d_1 of holes, parallel and angled to a free face respectively, for the same line of resistance, whereas for parallel and angled holes of the same diameter $\frac{W_1}{W} = \frac{13}{12 + \text{cosec } b}$ gives the ratio of the lines of resistance.

From the above it is evident that a borehole will give the greatest efficiency when it is perpendicular to the line of resistance, and the least efficiency when the line of resistance coincides with the axis of borehole.

CHAPTER VI.

ON THE MAXIMUM DISTANCE APART THAT SIMILAR SHOTHOLES, WHEN IN LINE PARALLEL TO A FREE FACE, WILL DISLODGE THE WHOLE OF THE ROCK BETWEEN THEM WHEN FIRED SIMULTANEOUSLY, THE LINE OF RESISTANCE FOR EACH HOLE BEING THE SAME AS IF IT WERE TO BE FIRED INDEPENDENTLY, AND THE LINE OF RESISTANCE FOR TWO OR MORE SHOTHOLES SUPPORTING EACH OTHER.

41. *Maximum Distance which Shotholes should be Placed Apart in Strong and Homogeneous Rock.*—If equal charges be placed in two boreholes $h\ h_1$ (Fig. 14) drilled parallel to the free face A B, so that the lines of resistance are equal, and fired simultaneously, the effect under certain conditions is much greater than if each were fired separately. For strong and homogeneous rock, when the charges have a length $= 12\ d$, we get the following results.

(*a*) When the distance between the charges $h\ h_1$ is greater than twice the line of resistance W, two independent craters of rock, $m\ h\ n$ and $n\ h_1\ o$, will be dislodged.

(*b*) When the distance $h\, h_1$ between the charges is equal to, or less than twice the line of resistance W, the masses of rock $m\, h\, n$ and $n\, h_1\, o$ will be dislodged together with the intervening mass $n\, h\, h_1$.

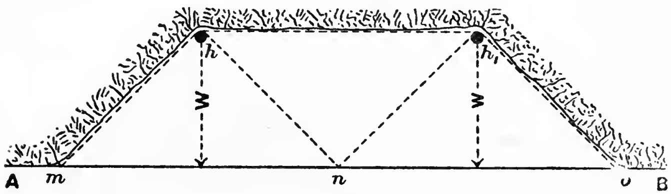

FIG. 14.

42. *Influence of the Cohesive Strength of Rock.*—The maximum distance that the holes can be placed apart varies according to the cohesive strength of the rock.

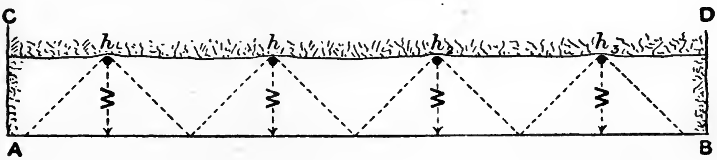

FIG. 15.

Suppose, for instance, the actual section of rock which the shotholes $h\, h_1\, h_2\, h_3$ (Fig. 15) would rupture if there were four lateral free faces, viz. A C B D (Fig. 15), and the free faces B D and E F shown in cross section (Fig. 16), when the line of resistance is of such length that the resistance to shearing is exactly equal to the resistance to fracture of the cross section h F by tension, assuming the rock to be perfectly inelastic, then, for the resistance to shearing for N shotholes we may put

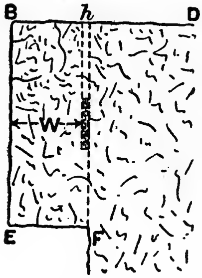

FIG. 16.

$$N(S \times W \times K_1) = N\,S\,W\,K_1,$$

K_1 being the modulus of shearing; and if the height of section is equal to the distance between the shotholes for the resistance of the cross section to fracture by tension, we shall have

$$N(eW \times eW \times K) = N\,e^2 W^2 K$$

(K being the modulus of rupture by tension and e W the distance between the shotholes) and therefore

$$N\,e^2 W^2 K = N\,S\,W\,K_1,$$

whence,
$$e = \sqrt{\frac{S}{W} \times \frac{K_1}{K}},$$

and
$$\frac{K_1}{K} = \frac{W}{S}e^2.$$

We have also (see Art. 37, p. 41)

$$A = S\,W\,C_a \quad \text{and} \quad A = S\,W_1\,C_{a1},$$

$$\therefore \quad \frac{W_1}{W} = \frac{C_a}{C_{a1}}.$$

Further, as the ratio of $\frac{K_1}{K} = C$ is a constant for different rocks,

$$e^2 = \frac{S}{W} \times C \quad \text{and} \quad e_1^2 = \frac{S}{W_1} \times C,$$

and consequently

$$\frac{W_1}{W} = \left(\frac{e}{e_1}\right)^2 \text{ and } \left(\frac{e}{e_1}\right)^2 = \frac{C_a}{C_{a1}}.$$

In blasting rock we have obtained the following results, viz. with charges in 1 inch diameter boreholes having a length of twelve times the diameter of borehole:

For very strong rock,

$$e = 2{\cdot}38 \quad\text{and}\quad \frac{S}{W} = \frac{26}{13} = 2.$$

For strong rock,

$$e = 2{\cdot}00 \quad\text{and}\quad \frac{S}{W} = \frac{26}{18} = 1{\cdot}44.$$

For moderately strong rock,

$$e = 1{\cdot}50 \quad\text{and}\quad \frac{S}{W} = \frac{26}{34} = 0{\cdot}765.$$

Therefore

$$\begin{aligned} \frac{K_1}{K} &= \frac{13}{26} \times (2{\cdot}38^2) = 2{\cdot}83 \\ &= \frac{18}{26} \times (2^2) \quad\;\; = 2{\cdot}77 \\ &= \frac{34}{26} \times (1{\cdot}5^2) \;\; = 2{\cdot}94 \end{aligned}$$

$$3\,\overline{)\,8{\cdot}54}$$

$$\text{Average value of } \frac{K_1}{K} = 2{\cdot}84.$$

Assuming then $\frac{K_1}{K}$ to have a constant value of $2{\cdot}84$, we can find the value of e from any values of $\frac{W}{S}$, or *vice versâ.* For instance, putting $e = 1$ and $S = 26$ inches for weak rock, we have

$\frac{W}{26} = 2 \cdot 84$ and $W = 73 \cdot 84$ inches for a 1 inch diameter borehole.

It is, however, important to note that, in consequence of the low cohesive strength of the rock, the chief factor in determining the charge in this case will be the force required to eject the rock after rupture.

From the above we have the following rule:—

For very strong rock, boreholes, having a length of charge $= 12\,d$, should be placed a distance $2\,W$ to $2 \cdot 38\,W$; for strong rock, a distance $1\frac{1}{2}\,W$ to $2\,W$; for moderately strong rock, W to $1\frac{1}{2}\,W$; and for weak rock, a distance W apart.

On the other hand, for the same rock it must be noted, that the value of e depends on the length of charge, as we have

$$e = \sqrt{2 \cdot 84 \frac{S}{W}} \quad \text{and} \quad e_1 = \sqrt{2 \cdot 84 \frac{S_1}{W}}.$$

Therefore

$$\frac{2 \cdot 84 \frac{S_1}{W}}{2 \cdot 84 \frac{S}{W}} = \left(\frac{e_1}{e}\right)^2 \quad \text{and} \quad \frac{S_1}{S} = \left(\frac{e_1}{e}\right)^2.$$

Consequently,

$$S_1 = \left(\frac{e_1}{e}\right)^2 S.$$

A 1-inch shothole gives $e = 1\frac{1}{2}$ and $S = 26$ for moderately strong rock when the length of charge $= 12\,d$.

Hence, to obtain $e = 2$ we must have

$$S = \left(\frac{2}{1\frac{1}{2}}\right)^2 \times 26 = 46{\cdot}22 \text{ inches.}$$

For S = 46·22 inches, the length of charge will be $\dfrac{46{\cdot}22 \text{ inches} - 2}{2} = 22{\cdot}11$ inches.

Simultaneous firing may, therefore, for hard rock

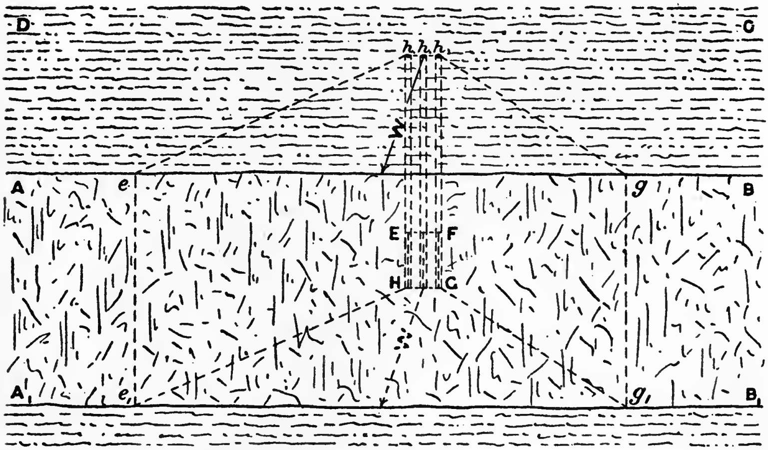

FIG. 17.

be productive of a greatly increased useful effect compared with the firing of the same charges consecutively, there being a saving of about 20 per cent. in the cost of blasting under certain conditions. It is, moreover, a valuable means of concentrating the forces of several charges to overcome a greater line of resistance than each is capable of when fired simultaneously.

43. *Line of Resistance for the combined Shearing Force of any Number of Similar Shotholes, equidistant from each other, in Line Parallel to a Free Face.*—In the case of a long line of free face (Figs. 17 and 18), a number of similar shotholes in line parallel thereto and equidistant from each other, when placed a certain distance k apart will overcome a line of resistance $W_n = q\,W$, W being the line of resistance corresponding to a single charge, if the charges be fired simultaneously. The value of q may be found in the following manner.

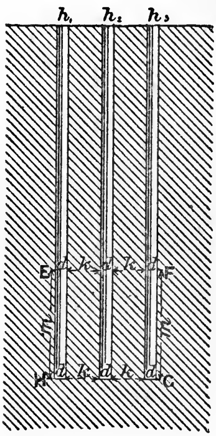

FIG. 18.

For one shothole,

$$W = \frac{A}{C_a S} = \frac{A}{C_a(2m + 2d)},$$

m being length of charge and d diameter of borehole, therefore we have the following values for S when all the shotholes are of the same diameter and placed near each other in line parallel to the free face,

For two shotholes $h\,h_1$ (Fig. 18),

$$S = EFGH = 2m + 4d + 2k.$$

For three shotholes $h\,h_1\,h_2$ (Fig. 18),

$$S = EFGH \times 2m + 6d + 4k.$$

And for N shotholes

$$S = 2m + 2Nd + (2N - 2)k.$$

Hence, for two shotholes,

$$W_1 = \frac{2A}{C_a(2m + 4d + 2k)};$$

For three shotholes,

$$W_2 = \frac{3A}{C_a(2m + 6d + 4k)};$$

For four shotholes,

$$W_3 = \frac{4A}{C_a(2m + 8d + 6k)};$$

For N shotholes,

$$W_n = \frac{NA}{C_a(2m + 2Nd + (2N - 2)k)}.$$

If $W_n = qW$ we have $\frac{A}{C_a W_n} = \frac{S}{q}$.

Consequently

$$\frac{N}{2m + 2Nd + (2N - 2)k} = \frac{q}{S}$$

$$NS = N(2qd + 2qk) = 2qm - 2qk.$$

$$N = \frac{2qm - 2qk}{S - (2qd + 2qk)},$$

$$N = \frac{2(m - k)}{\frac{S}{q} - 2(d + k)}$$

and

$$q = \frac{NS}{2((m + Nd) + k(N - 1))}.$$

When q is less than unity, the combined shearing force is not so great as the shearing force of each shothole acting independently, in which case the line of resistance will be limited by the latter.

44. *Economy of Firing several similar Charges close together in Line Parallel to a Free Face.*—Great economy may be obtained in blasting very hard rock which is without well defined joints, when there is a sufficient length of free face, by the use of a number N of similar shotholes placed close to each other in line parallel to the free face, and firing them simultaneously. Such economy is due to the greater line of resistance that may be blasted by their combined action than if each were fired singly, as the quantity of rock blasted increases as the cube of the line of resistance.

The value of $q = \dfrac{N\,S}{2\,((m + N\,d) + (N - 1)\,k)}$

(see Art. 43), for similar charges applied in boreholes of 1 inch diameter which are placed 3 inches apart, in line parallel to a free face, length of charge being 12 inches, is as follows:

(*a*) For $N = 1 \quad q = 1$
(*b*) ,, $N = 2 \quad q = 1\frac{9}{17}$
(*c*) ,, $N = 3 \quad q = 1\frac{6}{7}$
(*d*) ,, $N = 4 \quad q = 2\frac{2}{25}$

That is, the line of resistance increases as q with the number of charges.

On the contrary, the coefficient C_v in the formula

$$L = C_v W^3$$

decreases with the number of charges as the quantity of rock blasted increases in a greater ratio. And as we can put

$$L_1 = C_{v1} W_1^3$$

for the quantity of the charge in N shotholes, we have

$$N C_v W^3 = C_{v1} W_1^3$$

and consequently,

$$C_{v1} = \frac{N C_v}{q^3}.$$

By substituting the value of N and q, given above in this formula, we get

$$(a) \quad C_{v1} = C_v.$$

$$(b) \quad C_{v2} = \frac{2 C_v}{(1\frac{9}{17})^3} = \cdot 56 C_v.$$

$$(c) \quad C_{v3} = \frac{3 C_v}{(1\frac{6}{7})^3} = \cdot 46 C_v.$$

$$(d) \quad C_{v4} = \frac{4 C_v}{(2\frac{2}{25})^3} = \cdot 29 C_v.$$

C_v, as found for a single shothole 1 inch in diameter, namely, for a quantity of rock W^3, is, consequently, reduced to $\cdot 29$ C_v for a quantity of rock $(2\frac{2}{25} W)^3 = W_1^3$, blasted by the combined action of four shotholes of the same diameter and containing similar charges. The limiting value of C_v is that required for the ejection of the rock.

CHAPTER VII.

QUANTITY OF ROCK WHICH WILL BE LOOSENED UNDER THE USUAL CONDITIONS OF BLASTING OPERATIONS, WHEN THERE ARE NO WELL DEFINED JOINTS OR FISSURES.

45. *The usual Method of Exavating Rock by Blasting*, when there are no well defined joints, is in steps or benches with straight free faces at right angles to each other, as represented in Figs. 19, 20, 21, and, except when the rock is cut up into very large blocks by joints, in which case the line of resistance is so regulated as to enable the blasts to break right up to the joints, it invariably gives the best results.

46. *Form of Craters.*—Fig. 19 shows the crater $e f g g_1 f_1 e_1$, which will be formed by the blast of a single shothole in a step of rock when there are only two free faces $A B_1$ and $A C$; Fig. 20, the crater $e f C C_1 f_1 e_1$, which will be formed when there are three free faces $A B_1$, $A C$ and $B C_1$; Fig. 21, the mass of rock bounded by the sides $A B_1$, $A C$, $A D_1$, $B C_1$, $D C_1$, and $A_1 C_1$, which will be blasted when there are four free faces $A B_1$, $A C$, $A D_1$

and B C_1; and Fig. 22, the crater $e f g h h_1 g_1 f_1 e_1$ which will be formed when there are only two free faces as in Fig. 19, and several similar shotholes are fired in line parallel to the free face A B_1 simultaneously.

47. *Angle of Lines of Rupture.*—If the rock is a homogeneous mass, and there are only two free

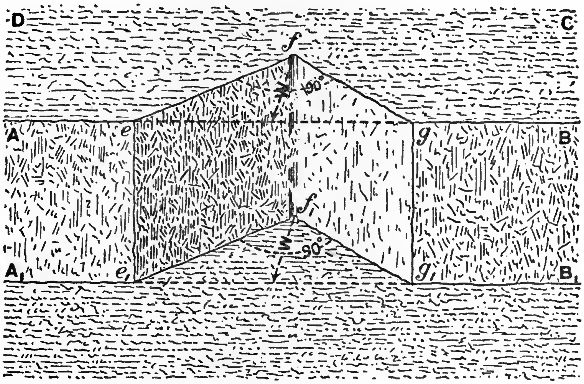

FIG. 19.

faces as in Fig. 19, the angle of the main lines of rupture $e f g$ or $e_1 f_1 g_1$ may be considered, for all practical purposes, to form a right angle or 90° with each other, or each to have an angle of 45° with the free face A_1 B_1, and in the case of there being other free faces as A D_1 and B C_1 (Fig. 21), the main lines of rupture for each may also make as great an angle as 90° if the face is of sufficient extent.

48. *Volume V of Rock Dislodged when there are Two Free Faces at Right Angles to each other.*—In accordance with the above, a shothole having two free faces will dislodge the volume of rock $e f g g_1 f_1 e_1$ (Fig. 19).

$$V = W^3 + \frac{m}{2} W^2.$$

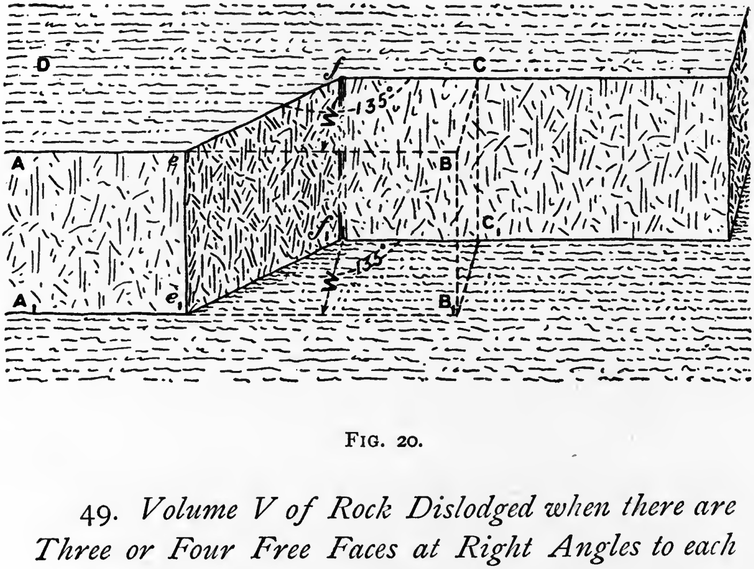

FIG. 20.

49. *Volume V of Rock Dislodged when there are Three or Four Free Faces at Right Angles to each other.*—If there are three free faces, the mass of rock dislodged will have a volume $e f C C_1 f_1 e_1 B_1 B$ (Fig. 20), and in case of four free faces the volume $A D C C_1 D_1 A_1 B_1$ (Fig. 22).

For three free faces $V = \frac{3}{2} W^3 + \frac{3}{4} m W^2.$

" four " $V = 2 W^3 + m W^2.$

50. *Volume V of Rock Dislodged by any Number of Similar Shotholes in a Step of Rock.*—In the case of several similar shotholes in line parallel to the face of rock A B_1 (Fig. 22), and the charges fired simultaneously, the mass of rock loosened will have a volume $e\,f\,g\,h\,h_1\,g_1\,f_1\,e_1$ (Fig. 22).

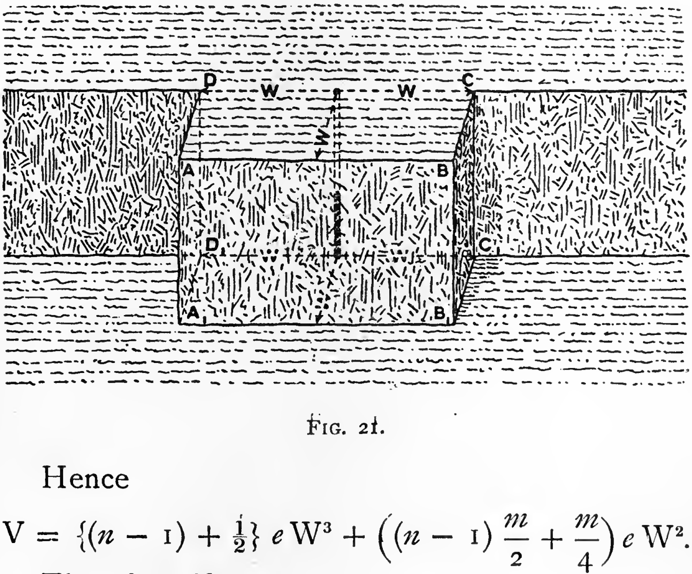

FIG. 21.

Hence

$$V = \{(n - 1) + \tfrac{1}{2}\}\, e\, W^3 + \left((n - 1)\frac{m}{2} + \frac{m}{4}\right) e\, W^2.$$

Therefore, if $e = 2$,

For two shotholes (fired simultaneously),

$$V = 3\,W^3 + \tfrac{3}{2}\, m\, W^2.$$

For three shotholes (fired simultaneously),

$$V = 5\,W^3 + \tfrac{5}{2}\, m\, W^2.$$

For four shotholes (fired simultaneously),

$$V = 7\,W^3 + \tfrac{7}{2}\, m\, W^2.$$

Consequently, in blasting with similar shotholes, when there are no joints or fissures to be considered,

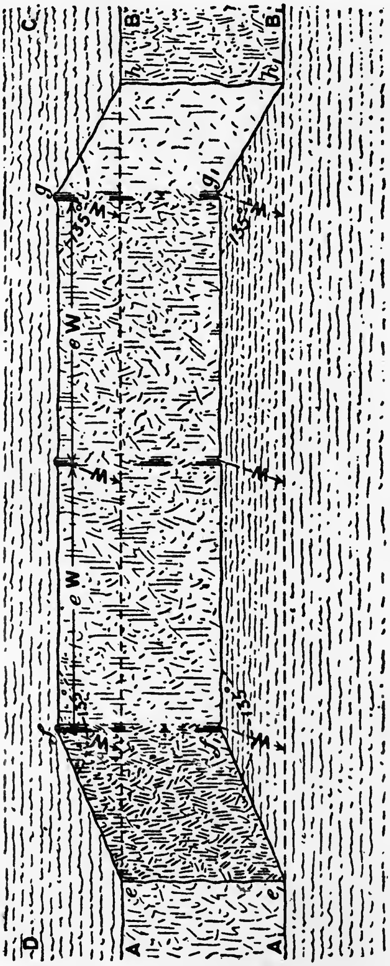

FIG. 22.

two holes fired simultaneously with two free faces will dislodge the same volume of rock as two such holes fired singly each with three free faces; three shotholes fired simultaneously with two free faces, the same volume of rock as two such holes fired singly each with three faces, and one other with four free faces; and four shotholes fired simultaneously with two free faces, the same volume of rock as two such holes fired singly each with three free faces, and two others fired singly each with four free faces.

51. *Volume of Rock blasted by a Concentrated Charge.*—According to our observations, the approximate volumes of rock which will be blasted in the case of a concentrated charge, and when the free faces are of sufficient extent to allow of full scope of action to the blast, are as follows:

For one free face . . .			$1\frac{1}{3}$ W^3.
,, two free faces at right angles to each other . .			$1\frac{2}{3}$ W^3.
,, three	ditto	ditto	$2\frac{1}{3}$ W^3.
,, four	ditto	ditto	3 W^3.
,, five	ditto	ditto	4 W^3.
,, six	ditto	ditto or a cubical block of stone	8 W^3.

The relative economy under the different conditions of free face and firing of the charges is evidently proportional to the volumes of rock blasted.

52. *Simultaneous and Consecutive Firing.*—It is evident from the above that the simultaneous firing of a number of shots will offer important advantages. This is especially the case, as before explained, when a number of shotholes are properly combined for the blasting of a long and straight wall of rock as indicated in Fig. 22, or for "unkeying" a single exposed surface of rock as by the four central shots Nos. 1 to 4 (Fig. 34). Cases, however, occur in which it is necessary to determine the order of the explosions to obtain the best effect, as for instance, for the enlarging shots numbered 5 to 24 (Fig. 34).

CHAPTER VIII.

THE LENGTH OF CHARGES IN BOREHOLES FOR RUPTURE BY SHEARING.

53. *Charges for Shearing.*—The best or most economical length for charges in boreholes may be deduced from the formula $W = \frac{A}{C_a S}$ in the following manner:

As $A = m\,d$, and $S = 2\,(m + d)$,

$$W = \frac{m\,d}{2\,C_a\,(m + d)}.$$

Therefore, when

$$m = d \qquad W = \frac{1}{2}\,\frac{d}{2\,C_a}$$

$$m = 2\,d \qquad W = \frac{2}{3}\,\frac{d}{2\,C_a}$$

$$m = 3\,d \qquad W = \frac{3}{4}\,\frac{d}{2\,C_a}$$

$$m = 4\,d \qquad W = \frac{4}{5}\,\frac{d}{2\,C_a}$$

$$m = 5\,d \qquad W = \frac{5}{6}\,\frac{d}{2\,C_a}$$

$$m = 6\,d \qquad W = \frac{6}{7}\,\frac{d}{2\,C_a}$$

$$m = 7\,d \qquad W = \frac{7}{8}\,\frac{d}{2\,C_a}$$

$$m = 8\,d \qquad W = \frac{8}{9}\,\frac{d}{2\,C_a}$$

$$m = 9\,d \qquad W = \frac{9}{10}\,\frac{d}{2\,C_a}$$

$$m = 10\,d \qquad W = \frac{10}{11}\,\frac{d}{2\,C_a}$$

$$m = 11\,d \qquad W = \frac{11}{12}\,\frac{d}{2\,C_a}$$

$$m = 12\,d \qquad W = \frac{12}{13}\,\frac{d}{2\,C_a}$$

On the contrary, for the coefficient $C_v = \frac{L}{W^3}$, L being the weight of the charge, and W the line of resistance, we have, if E m represents the weight of charge L,

$$C_v = \frac{E\,m}{W^3}.$$

Therefore, when

$$m = d \qquad C_v = \frac{E\,d}{\left(\frac{1}{2}\,\frac{d}{2\,C_a}\right)^3} = 8\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 2\,d \qquad C_v = \frac{E\,2\,d}{\left(\frac{2}{3}\,\frac{d}{2\,C_a}\right)^3} = 6\tfrac{3}{4}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 3\,d \qquad C_v = \frac{E\,3\,d}{\left(\frac{3}{4}\,\frac{d}{2\,C_a}\right)^3} = 7\tfrac{1}{9}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 4\,d \qquad C_v = \frac{E\,4\,d}{\left(\frac{4}{5}\,\frac{d}{2\,C_a}\right)^3} = 7\tfrac{13}{16}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 5\,d \qquad C_v = \frac{E\,5\,d}{\left(\frac{5}{6}\,\frac{d}{2\,C_a}\right)^3} = 8\tfrac{16}{25}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 6\,d \qquad C_v = \frac{E\,6\,d}{\left(\frac{6}{7}\,\frac{d}{2\,C_a}\right)^3} = 9\tfrac{19}{36}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 7\,d \qquad C_v = \frac{E\,7\,d}{\left(\frac{7}{8}\,\frac{d}{2\,C_a}\right)^3} = 10\tfrac{22}{49}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 8\,d \qquad C_v = \frac{E\,8\,d}{\left(\frac{8}{9}\,\frac{d}{2\,C_a}\right)^3} = 11\tfrac{25}{44}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 9\,d \qquad C_v = \frac{E\,9\,d}{\left(\frac{9}{10}\,\frac{d}{2\,C_a}\right)^3} = 12\tfrac{28}{81}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 10\,d \qquad C_v = \frac{E\,10\,d}{\left(\frac{10}{11}\,\frac{d}{2\,C_a}\right)^3} = 13\tfrac{31}{100}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 11\,d \qquad C_v = \frac{E\,11\,d}{\left(\frac{11}{12}\,\frac{d}{2\,C_a}\right)^3} = 14\tfrac{34}{121}\,\frac{(8\,E\,C_a^3)}{d^2}$$

$$m = 12\,d \qquad C_v = \frac{E\,12\,d}{\left(\frac{12}{13}\,\frac{d}{2\,C_a}\right)^3} = 15\tfrac{37}{144}\,\frac{(8\,E\,C_a^3)}{d^2}$$

From the values of W given above, it is evident that if the length of the charge m be infinitely increased beyond $12\,d$, W will only be increased $\frac{1}{12}$, or beyond $8\,d$ no more than $\frac{1}{8}$, and as C_v increases as shown with the length of charge, as a general rule, owing to the influence of the periphery of the charging chamber on the blast as explained below, the limits of the length of charge should vary between $8\,d$ and $12\,d$ according to the degree of economy required in the consumption of explosive.

54. *Influence of Form of Chamber on Shearing Force of Charge.* — According to the formula $W = \frac{A}{C_a\,S}$ it appears that an elongated charge, as in a borehole, is not a favourable form for obtaining the least resistance to a blast for a given line of resistance, for, as $\frac{A}{S}$ represents the influence of the form of chamber on the shearing force of the charge, the resistance will decrease as the sectional area, or projection of the chamber at right angles

to the direction of the blast, approaches a square, and is a minimum when such projection is a circle. This, however, only obtains in case there is only one free face, for if there are lateral free faces it is advantageous to have an elongated charge to insure the whole mass of rock being carried away to such free faces, as with a relatively high value of $\frac{A}{S}$ the blast would produce a conical cavity in the centre of the mass and not carry away the rock to the lateral free faces.

55. *The Length of Charge in Boreholes should be a Constant Multiple of the Diameter for Shearing.*—When the lengths of charges used in boreholes are made a constant multiple of their diameters, as, for instance, $n\,d$, we can put for the weight of charge L for a diameter of borehole d,

$$L = \cdot 7854\, d^2\, n\, d\, E = \cdot 7854\, E\, n\, d^3,$$

E representing the weight of a cubic inch of explosive, and for a diameter of borehole d_1

$$L_1 = \cdot 7854\, E\, n\, d_1^3.$$

Therefore,

$$\frac{L_1}{L} = \left(\frac{d_1}{d}\right)^3,$$

But as $\frac{d_1}{d} = \frac{W_1}{W}$,

$$\frac{L_1}{L} = \left(\frac{W_1}{W}\right)^3,$$

which evidently agrees with the formula

$$L = C_v W^3 \text{ as } C_v = \frac{L_1}{W_1^3}.$$

Hence, if we have found that a length of charge $n d$ in a borehole whose diameter is d, will give the proper charge for the line of resistance corresponding to this diameter of borehole and the weight of rock to be ejected, then a length of charge $n d_1$ will give the proper charge for the line of resistance corresponding to any diameter of borehole d_1 in the same rock.

CHAPTER IX.

THE BEST POSITION FOR A CHAMBER OR CHARGE WHEN THERE ARE TWO OR MORE FREE FACES AT RIGHT ANGLES TO EACH OTHER.

56. *Principle on which the Best Position for a Chamber may be determined.*—To obtain the best effect with a blast in rock there must be equilibrium of resistance on all sides of the line of resistance to the action of the charge; hence the position of the chamber should be determined on this principle.

57. *Rule for determining Distance of Chamber from Free Faces.*—As before explained (Art. 42), when two or more similar charges are situated a distance eW apart (e = 2 for strong rock, $1\frac{1}{2}$ for moderately strong rock, and 1 for weak rock) in homogeneous rock, parallel to a straight free face, and fired simultaneously, the whole of the intervening rock is dislodged; but when the distance between the holes exceeds eW each charge will blast a distinct crater. We may therefore conclude that the limiting distance of action for each charge is midway between the holes, or a distance $\frac{e\text{W}}{2}$.

Therefore, as the force of a blast in a borehole chamber is equally great on any side of the same except the ends, and such force will overcome the same resistance of rock on any side having a free face, the distance of any lateral free face from the side of borehole should be equal to the line of resistance; and, on the contrary, for a free face at right angles to the axis of borehole the distance of same from the centre of the charge should be $\frac{eW}{2}$, as the pressure of the blast on the end of the borehole is comparatively small, and could not produce rupture acting independently of the lateral pressure in the hole.

From the above considerations we have deduced the following rule for determining the proper position for a borehole chamber, viz. :—

The distance from the centre of a charge to any lateral free face, measured perpendicularly to the axis of the borehole, should be the same as the line of resistance, and to any end free face, measured in line with the axis of borehole, a length $\frac{eW}{2}$.

When $e = 2$ the distance from the centre of charge to any free face should be the same as the line of resistance, which may be adopted in practice as sufficiently accurate under most conditions.

Therefore, for two free faces at right angles to each other the proper position for a borehole charge is that illustrated in Fig. 23, in plan, and Fig. 24, in section, in which A B and C E represent the free faces, h the borehole, D the depth of borehole, m

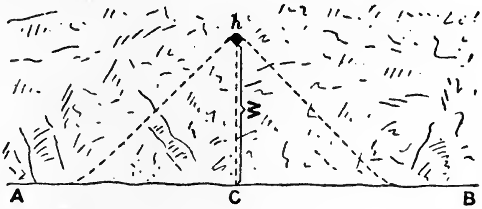

FIG. 23.

the length of charge, and T the tamping, or length of borehole above the charge.

Accordingly,

$$D = m + T.$$

And as $T = W - \frac{m}{2}$

$$D = \frac{m}{2} + W.$$

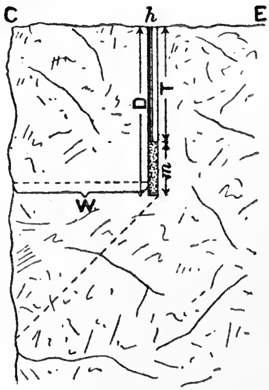

FIG. 24.

For three free faces the position of the charge should be that indicated in Fig. 25 in plan, and Fig. 26, in section.

58. *Main Lines of Rupture.*—For rock of homogeneous composition and uniform texture the main lines of rupture, $h\,a$, $h\,e$ and h B (Fig. 25), would

reach the surface as indicated by the dotted lines, that is, they make an angle of 180° between the

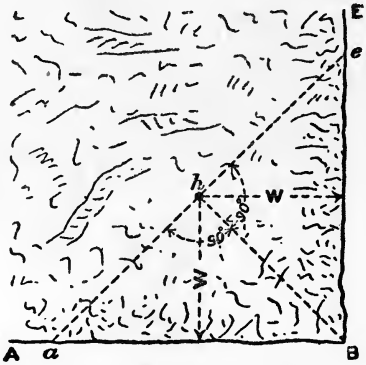

FIG. 25.

two lateral free faces, or an angle of 90° for each free face.

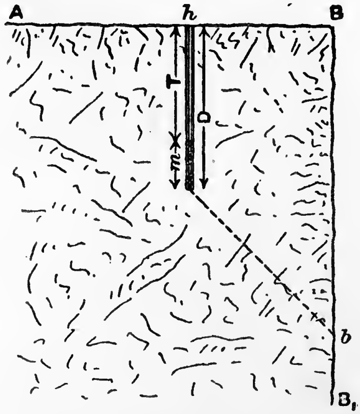

FIG. 26.

Owing to the want of homogeneity in rock, and to the existence of joints and fissures, the outer line of rupture will not, in practice, run so regularly as indicated by the dotted lines. In case of rupture by shearing the line of rupture is a slightly convex curve, as shown in Fig. 1.

59. *Irregular Faces of Rock.*—A circumstance which will influence the position of the chamber, sometimes in a very important degree, and which must be taken into account in estimating the line of resistance, is the irregularity of the faces of the rock, which, instead of forming unbroken planes parallel to the borehole, are broken up more or less by projecting bosses and deep depressions. Experience and good judgment, combined with a knowledge of the principles of blasting, must guide the blaster in this case.

CHAPTER X.

BOREHOLE CHARGES.

60. *Formulæ for Weight of Borehole Charges.*—Calling d the diameter of borehole, $n\,d$ the length of charge, g the specific gravity of the explosive, U the cubical contents of charge, and L the weight of charge in lbs., we have

$$U = \cdot 7854\, d^2 \times n\, d$$
$$U = \cdot 7854\, n\, d^3.$$

The weight of one cubic inch of any explosive in lbs. is $\cdot 036\, g$, and consequently, when d is expressed in inches,

$$L = \cdot 7854\, n\, d^3 \times \cdot 036\, g$$
$$L = \cdot 0283\, n\, g\, d^3.$$

When $n = 12$, and $g = 1 \cdot 6$ as for dynamite,

$$L = \cdot 0283 \times 1 \cdot 6 \times 12\, d^3$$
$$L = \cdot 5434\, d^3.$$

The weight of charge is also given by the formula

$$L = C_v\, W^3.$$

But as the charge must be applied in a chamber giving sufficient pressure area to the blast, at right angles to the line of resistance, to overcome the cohesive strength of the rock, it is often more useful to have it expressed in terms of C_a and W, by substituting the value of C_a in terms of C_v in the above formula. The value of C_a in terms of C_v may be found in the following manner:

$$\text{Since } L = C_v W^3 = {}^{\boldsymbol{\cdot}}3396\, g d^3.$$

$$\left(\frac{d}{W}\right)^3 = \frac{C_v}{{}^{\boldsymbol{\cdot}}3396\, g}.$$

$$\frac{d}{W} = \sqrt[3]{\frac{C_v}{{}^{\boldsymbol{\cdot}}3396\, g}}.$$

But for a borehole chamber whose projection is $A = C_a S W$, we have $A = n d^2$, and $S = (n + 1)\, 2\, d$.

Therefore $$C_a = \frac{n d}{2(n + 1) W},$$

and

$$\frac{d}{W} = \left(\frac{2n + 2}{n}\right) C_a.$$

From the above values of $\frac{d}{W}$ we have

$$\left(\frac{2n + 2}{n}\right) C_a = \sqrt[3]{\frac{C_v}{{}^{\boldsymbol{\cdot}}3396\, g}}$$

and

$$C_a = \left(\frac{n}{2n + 2}\right) \sqrt[3]{\frac{C_v}{{}^{\boldsymbol{\cdot}}3396\, g}}.$$

When $n = 12$

$$C_a = \frac{12}{26}\sqrt{\frac{C_v}{\cdot 3396\,g}} = \frac{6}{13}\sqrt[3]{\frac{C_v}{\cdot 3396\,g}}$$

and

$$C_v = 3{\cdot}454\,g\,C_a^3.$$

When W is expressed in feet

$$C_v = 5969\,g\,C_a^3.$$

Substituting these values of C_v in the formula $L = C_v W^3$, we have, when W is expressed in inches,

$$L = 3{\cdot}454\,g\,C_a^3 \,.\, W^3;$$

when W is expressed in feet

$$L = 5969\,g\,C_a^3 \,.\, W^3.$$

Suppose, for example, for a borehole in very strong rock, that $C_a = {\cdot}02$, then C_v must be

$$3{\cdot}454\,g \times {\cdot}02^3 = {\cdot}00002763\,g$$

to enable the blast to produce rupture.

If W in the formula $L = C_v W^3$ be taken in feet, and in the formula $A = C_a\,S\,W$ in inches,

$$C_v = {\cdot}00002763\,g \times 1728 = {\cdot}048\,g.$$

CHAPTER XI.

THE INFLUENCE OF FISSURES, JOINT AND BEDDING PLANES IN DETERMINING THE CHARGE.

61. *Favourable Conditions for Quarrying Operations.*—A consideration of great importance is the existence of fissures, joint planes and bedding planes, also lines of statification. It often happens that a bed of rock is cut up by such planes into detached blocks of greater or less dimensions, which must be considered as more or less unsupported faces to determine the proper position for a charge, and the length of the line of resistance. In some quarries joints traverse rocks in straight and well determined lines, and are slightly open, thus affording to the quarryman the greatest aid in the extraction of blocks of stone. When a sufficient number of joints cross each other the whole mass of rock is split into symmetrical blocks, and offers the best possible conditions for quarrying operations.

62. *Rupture without Shearing. The Resistance to Rupture of any Section of Rock limited by Joints or Free Faces.*—In the case of joints and free faces, as in blasting a mass of rock *e f l k f g l m*, Fig. 27,

which is bounded by the free faces A B_1 and A D, the vertical joints *e f g h* and *k l m n*, and the bedding joint $e_1 g m k_1$, if the joints have little or no cohesion along their surfaces, and they are parallel or diverge towards the front face A B_1, the cohesive resistance

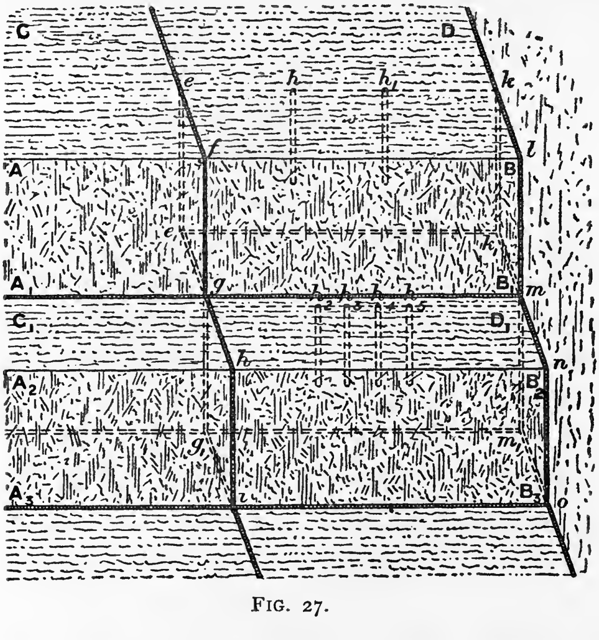

FIG. 27.

of the rock to be overcome by the shotholes $h\,h_1$ will be proportional to the smallest section of the mass through the shotholes, which should be bored between the joints and parallel to the front face. Suppose F and F_1 to be any two such sections of similar rock, varying according to the distance between the joints, and that the same could be ruptured by shotholes having charges of the same kind of explosive, and whose respective charging chambers have projections A and A_1 parallel to the front face A B; then for any given line of resistance W it is evident that we should have the following relations of the quantities F and F_1, A and A_1, and S and S_1, viz.

$$\frac{F_1}{F} = \frac{A_1}{A} = \frac{S_1}{S}.$$

On the contrary, for rocks of different cohesive strength, and shotholes of the same diameter, the lines of resistance should be proportional to the cohesive strength of the rocks, as we should have

$$F = e^2 W^2, \quad \text{and} \quad F_1 = e_1^2 W_1^2$$

and

$$\frac{F_1}{F} = \left(\frac{e_1 W_1}{e W}\right)^2.$$

But we can put F = the section that would require the same force to rupture it as the section F_1 in a different rock, under which conditions $A = C_a$, S W equals $A = C_{a1}$ S W_1, and therefore

$$\frac{C_a}{C_{a1}} = \frac{W_1}{W}.$$

Consequently,

$$\frac{F_1}{F} = \left(\frac{e_1}{e}\right)^2 \left(\frac{C_a}{C_{a1}}\right)^2,$$

and as $\left(\frac{e_1}{e}\right)^2 = \frac{C_{a1}}{C_a}$ (see Art. 42, page 54)

$$\frac{F_1}{F} = \frac{C_a}{C_{a1}}$$

and

$$F_1 = \frac{C_a}{C_{a1}} F.$$

For very strong rock (see Table I.), when $C_a = \cdot 032$, we find $e = 2 \cdot 27$, and for a 1-inch diameter shothole charged with dynamite, A = 12 square inches, and W = 14·37 inches.

Therefore,

$$F = e^2 W^2 = (2 \cdot 27)^2 \times (14 \cdot 37)^2 = 1064 \text{ sq. inches}$$
$$= 7 \cdot 38 \text{ sq. feet.}$$

Hence, for a 1 inch diameter shothole in weak rock whose coefficient is $C_a = \cdot 008$,

$$F_1 = \frac{\cdot 032}{\cdot 008} \times 7 \cdot 38 = 29 \cdot 52 \text{ sq. feet.}$$

The sectional area of rock 29·52 sq. feet will offer the same resistance to rupture as the line of resistance for the shothole, which agrees with the formula $W = \dfrac{A}{C_{a1} S} = 4$ feet $9\frac{1}{2}$ inches, will offer to shearing.

On the other hand, it is evident that F varies as the square of the line of resistance, and as the line of resistance is proportional to the diameter d of borehole for any diameter of shothole, we can put

$$F = \frac{\cdot 032}{C_a} \times 7 \cdot 38 \, d^2.$$

This formula is very useful in practice for determining the proper number of similar holes to rupture any given section of rock.

For example, it is required to blast a section of rock 26 feet long and 10 feet high, 8 feet back from the main face, the coefficient of the rock being ·008 for dynamite, when the section is bounded by free faces, or joints offering no resistance.

The line of resistance being 8 feet, we can adopt holes of any diameter which will not shear a greater thickness of rock than 8 feet.

According to the formula $W = \frac{A}{C_a S}$, $1\frac{1}{2}$ inch diameter shotholes are equal to a line of resistance of 7 feet 2 inches, and we may, therefore, adopt holes of this or any smaller diameter according to the ballistic and shattering effect required.

For a $1\frac{1}{2}$-inch diameter shothole, we have

$$F = \frac{\cdot 032}{\cdot 008} \times 7\cdot 38 \times (1\tfrac{1}{2})^2 = 66\cdot 42 \text{ sq. feet,}$$

that is to say, each $1\frac{1}{2}$-inch shothole is equal to the rupture of a section of rock whose area is 66·42 sq. feet.

But the whole section to be ruptured is

$$26 \times 10 = 260 \text{ sq. feet.}$$

Consequently, the number of holes required is

$$\frac{260}{66\cdot 42} = 4 \text{ nearly.}$$

The holes should be placed, as shown in Fig. 27, in the lower bench to give the best effect, viz. perpendicular to the top face A B, and so that the distance of the end holes from the sides is equal to the line of resistance of the charges, viz. 7 feet 2 inches, and the centre holes equidistant from each other and the end holes.

The charge of dynamite required for each hole is 1·833 lb., being given by the formula $L = \cdot 5434\, d^3$, Art. 60, page 80, and for the four holes $1 \cdot 833 \times 4 = 7 \cdot 332$ lbs.

The volume of rock blasted will be $26 \times 10 \times 8 = 2080$ cubic feet.

The chambers should be situated in the central part of section, and therefore the depth of each hole will be

$$\frac{10 + m}{2} = 5 \text{ feet } 9 \text{ inches.}$$

The section, however, may be blasted with fewer holes, if the length of charge and depth of hole be increased, since we have

$$\frac{F_1}{F} = \frac{A_1}{A} = \frac{S_1}{S}.$$

Supposing then that we make $A_1 = 2\,A$, by doubling the length of charge it is clear that half the number of holes will suffice, and that they should have a depth of

$$\frac{10 + m}{2} = \frac{10 + 3}{2} = 6 \text{ feet } 6 \text{ inches.}$$

Economy in the boring of holes in this case is, therefore, attained by increasing the length of charge.

The two $1\frac{1}{2}$-inch holes, 6 feet 6 inches deep, should be bored, as indicated in Fig. 27, in the upper bench.

Comparing the above with a case of stronger rock, as, for instance, when $C_a = \cdot 014$, then

$$F = \frac{\cdot 032}{\cdot 014} \times 7 \cdot 38 \times (1\tfrac{1}{2})^2 = 37 \cdot 96 \text{ sq. feet.}$$

And the number of $1\frac{1}{2}$ inch holes required if $m =$ 1 foot 6 inches is

$$\frac{260}{37 \cdot 96} = 7 \text{ holes.}$$

But experience shows that the length of charge may be one-half of the depth of hole under the given conditions. Therefore, if the holes be bored to a depth of 7 feet, the length of charge in each may be $\frac{7}{2}$ feet = 3 feet 6 inches, and there will be required for the work to be done

$$\frac{7 \times 1 \text{ foot } 6 \text{ inches}}{3 \text{ feet } 6 \text{ inches}} = 3 \text{ holes,}$$

instead of 7 holes as for the shorter length of charge 1 foot 6 inches. On the other hand, the total weight of charge will be the same for the seven as for the three holes viz.:

$$7 \times 1 \cdot 833 \text{ lb.} = 3 \times 4 \cdot 277 \text{ lbs.} = 12 \cdot 831 \text{ lbs.}$$

The ballistic force of the blast for the same length of charge will be directly proportional to the square of the diameter of the boreholes. To reduce the same so as to just crack the rock from its bed, the diameter of the holes must be diminished.

For example, if 1-inch holes be used,

$$F = \frac{\cdot 032}{\cdot 014} \times 7\cdot 38 = 16\cdot 87,$$

and the number of holes of this diameter for a length of charge $= 12\,d = 1$ foot is

$$\frac{260}{16\cdot 87} = 16 \text{ holes nearly,}$$

which number may be reduced to $\frac{16}{4} = 4$ holes by making the length of charge 1 foot × 4 = 4 feet.

A charge of dynamite 1 foot long in a 1-inch hole weighs ·543 lb.; hence, the total weight of the charge will be

$$16 \times \cdot 543 \text{ lb.} = 4 \times 2\cdot 172 \text{ lbs.} = 8\cdot 69 \text{ lbs.}$$

There will, in consequence, be 8·69 lbs. to project the mass in this case, instead of 12·831 lbs. as in the other. In case there are no bedding joints, $e\,g\,m\,k_1$, and $g_1\,i\,o\,m_1$ (Fig. 27), the 1-inch and 1½-inch shotholes must be placed 2 feet 9 inches, and 4 feet 1½ inches (the lines of resistance for shearing) back from the free face A B.

63. *Length and Position of Charge for Shearing in Beds of Rock.*—In the case of beds of rock as

illustrated in Fig. 28, the lines of rupture produced by the explosion of the charge *m*, in the borehole, will evidently be limited by the bedding plane or joint C D, and if the joint C D is an open one, we may assume that there is practically no resistance to the blast along C D. In this case, when the

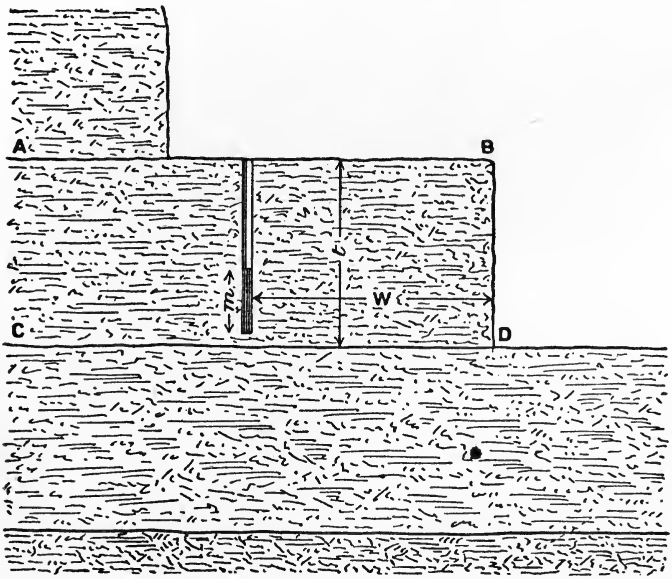

FIG. 28.

thickness *t* of the bed is less than *e* W (W being the line of resistance if there were no joint C D), the length of charge should be proportioned to the relative resistance, which may be ascertained in the following manner.

In homogeneous rock, when there are no joints or fissures, we have for the resistance to a blast $R = S\,W\,K_1 = e^2\,W^2\,K$, hence a section of rock whose section is $e^2\,W^2$ may be a measure of the resistance of any shothole. On the same principle, the relative measure of the resistance to rupture when we are dealing with a bed of rock limited in thickness to less than $e\,W$, as shown in Fig. 28, if we denote the thickness of the bed by t, is $e\,W \times t$, which is evidently a smaller quantity than $e^2\,W^2$. Therefore, the length of charge should be reduced so that the force of the blast is proportional to the resistance, as the force of a blast is proportional to the length of charge. Putting then m and m_1 as the lengths of charge required to overcome the resistances $e^2\,W^2$ and $e\,t\,W$ we have

$$m : m_1 :: e^2\,W^2 : e\,t\,W,$$

whence

$$m_1 = \frac{t\,m}{e\,W}.$$

For strong rock we have $e = 2$, and consequently,

$$m_1 = \frac{t\,m}{2\,W}.$$

That is, when

$t = W$	$m_1 = \frac{1}{2}m$
$t = 1\frac{1}{4}W$	$m_1 = \frac{5}{8}m$
$t = 1\frac{1}{2}W$	$m_1 = \frac{3}{4}m$
$t = 1\frac{3}{4}W$	$m_1 = \frac{7}{8}m$
$t = 2\,W$	$m_1 = m.$

If the joint C D is somewhat open it offers little or no resistance, and the proper position of the charge will be midway between the joint planes A B and C D. On the contrary, if the joint C D is tight the charge should extend to the same, and it should be adjusted between these limits according to the tightness of the joint C D, on the prin-

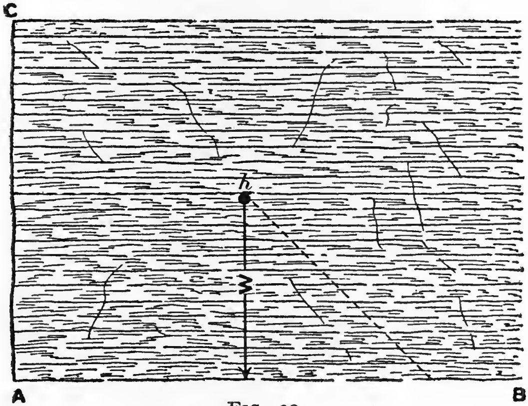

FIG. 29.

ciple that there should be equilibrium of resistance on all sides of the line of resistance.

When there are lateral free faces or joints we may place the charge a maximum distance W from the same when this is possible.

The charge should always be located in whole rock to prevent free escape of the gases and consequent reduction of the power of the blast. This rule, of course, only holds good when the strata are thicker than the length of the charge.

In laminated or slaty rock the resistance to rupture is invariably less along the planes of stratification than across the strata. Therefore, in order to locate a shothole advantageously in, or parallel to, the planes of stratification when there are two lateral free faces A B and A C (Fig. 29), the former parallel to the planes of stratification and the latter at right angles thereto, the shothole *h* should be so situated that the distance from the free face A C is 1 to $1\frac{1}{4}$ its distance from the free face A B, which is the line of least resistance. The proper distance of the shothole *h* from the free face A C must be found by trial shots.

CHAPTER XII.

BLASTING IN CUTTINGS, STOPES OR QUARRIES.

64. *Placing of Shotholes in Cuttings or Stopes.*—In a cutting or stope the rock should be blasted in steps with straight faces or walls (as illustrated in Figs. 30, 31 and 32, plan, longitudinal section and end view), so that each step is completely removed by the series of shotholes $a\,a_1\,a_2\,a_3$ and $b\,b_1\,b_2\,b_3$ placed in line parallel to the free faces A B C D and C D E F. All the shotholes should be of the same length and diameter, and contain equal charges. We can make the line of resistance that which each single shothole will overcome when fired separately, which we will call W, or a length $W_1 = \frac{N\,S}{2\,\{(m + N\,d) + k\,(N - 1)\}} \times W$, by using a number of holes in close juxtaposition. In the former case the central holes should be placed a distance $e\,W$ apart, as before explained, and the end holes a distance $\frac{e\,W}{2}$ from the nearest central holes. The end shots, $a\,a_3$ and $b\,b_3$, are necessary to maintain the profile of the cutting or stope.

65. *Irregular Surface Line of Rock.*—In case the rock has an irregular surface line, or a ledge of rock has to be removed, as in Fig. 33, regular

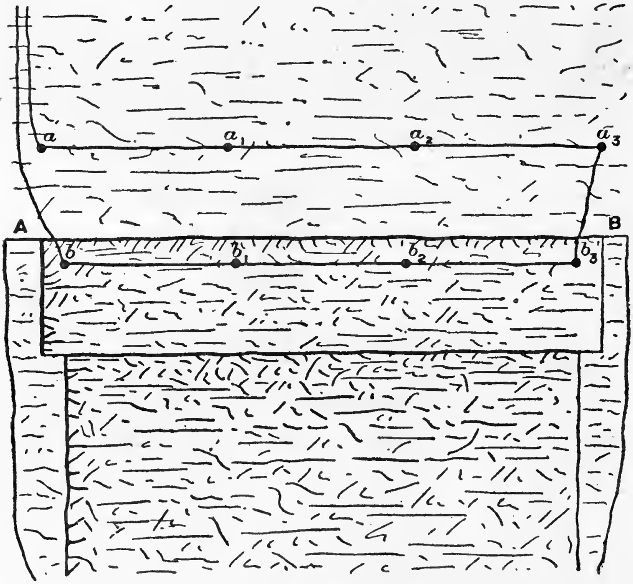

FIG. 30.

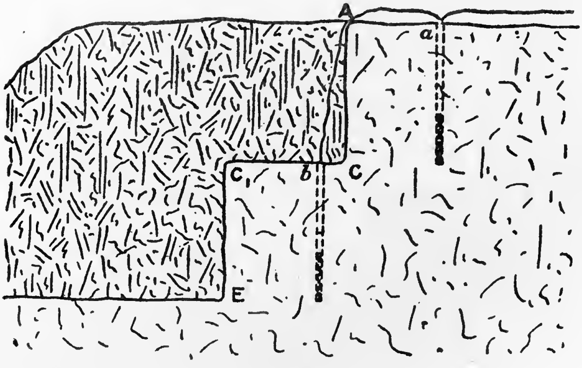

FIG. 31.

steps may be established by removing the section of rock *d e b* by the shotholes $h\,h_1\,h_2\,h_3$, and the section of rock *b c a* by the shotholes h_4 and h_5. To obtain a level floor it is often advantageous to bore horizontal holes, as *h*, shorter vertical holes being used, and the bottom blasted by horizontal holes.

66. *Joints.*—When there are regular and well defined joints, offering comparatively little resistance

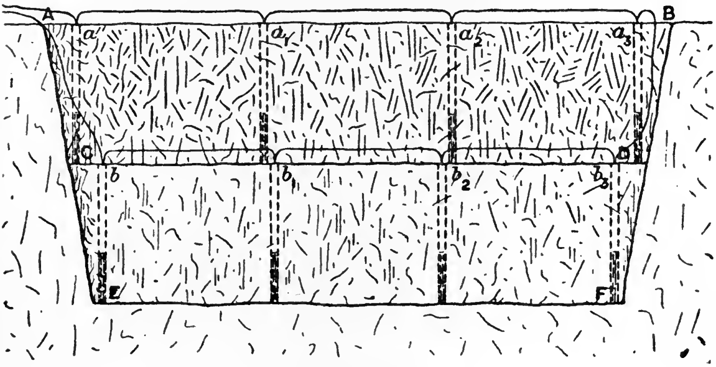

FIG. 32.

along their surfaces, shotholes should be placed to take advantage of this favourable condition for the excavation of the rock.

In granite quarries the joints may be very irregular, as in the Aberdeen quarries, or run in a very regular manner, as in the Cornish quarries. The position of the joints is of first importance in selecting a site for a quarry. For instance, it is found in granite formations that the direction of the beds generally corresponds with the outline of the hill; consequently the bedding planes are horizontal when the surface is horizontal, and are inclined when the surface is sloping. When the angle of dip of the beds is great this condition is very disadvantageous for a quarry, as not only is the quarrying of the stone more expensive but there is considerable risk of accidents from the blocks having a tendency to slip down the steep planes.

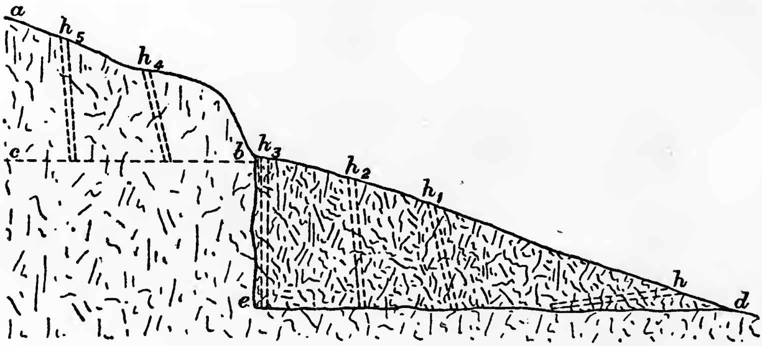

FIG. 33.

The greatest economy is obtainable when the bedding joint planes are either horizontal or dip very slightly towards the direction in which the rock is to be blasted. When it is necessary to open a quarry in a steep side of a hill the opening should not be made straight into it, as the masses of rock which stand separated from each other by natural divisions (joints) might fall on the workmen engaged beneath them.

CHAPTER XIII.

PLACING OF SHOTHOLES WHEN THERE IS ONLY A SINGLE EXPOSED SURFACE FOR ATTACK, AND NUMBER OF SHOTHOLES REQUIRED FOR A HEADING OR SHAFT.

67. *Removal of an Entering Portion of Rock.*—In the foregoing considerations the holes have been assumed to be drilled in the most favourable position for blasting, viz. parallel to a free face, which is only practicable when there are two or more free faces available. When, however, we have to attack a single exposed surface, or free face, as frequently occurs in driving headings and shaft sinking, one or more holes have to be drilled at an angle to the free face to remove an entering portion of rock and leave the surrounding rock unsupported, which is called "angling the holes to unkey the rock," or "taking out the key."

By the unkeying of the rock a new free face is provided, approaching more or less to a right angle with the other, so that the succeeding or enlarging shotholes, when drilled in line with the heading or shaft, will have a free face approximately parallel to the axes of the hole to break against. An illustration of the centre cut method of attacking a single exposed surface when there are no joints or fissures available is given in Figs. 34, 35 and 36, in which 1 to 4 are the breaking-in shots and 5 to 20 the enlarging ones. Fig. 34 shows face of attack and

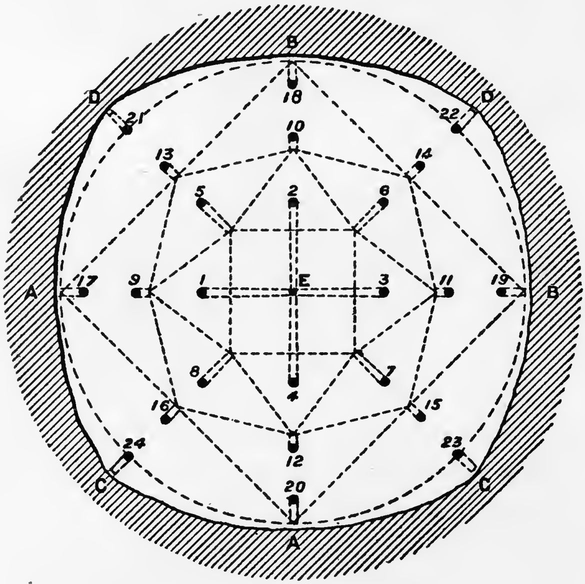

FIG. 34.

plan of holes, and Figs. 35 and 36 the holes in section.

The line of resistance W_1 for the breaking-in shots is shown in Fig. 35. The effect is greatest if the shotholes 1, 2, 3 and 4 meet, or approach so nearly each other that the intervening rock is fissured or pulverised, as in that case a pressure surface for the gases from the explosion of the charges will be produced parallel to the face of the heading between the limits of the charges.

68. *Arrangement of Holes in Headings or Shafts.* Supposing all the shotholes besides the breaking-in ones to have the same length and diameter, the best arrangement of holes will evidently be that

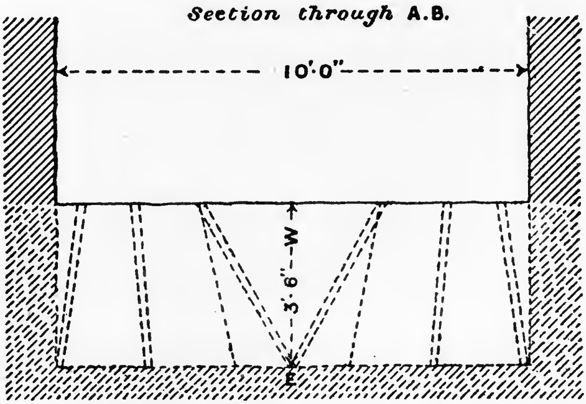

FIG. 35.

which gives equal resistance to each, whereas the number of holes will depend on the size of heading or shaft, tenacity of the rock, and strength of explosive used.

In the Figs. 34, 35 and 36, the holes are shown arranged on this principle, viz. first, 4 breaking-in shots, numbered 1 to 4, converging to the centre E of the heading; secondly, the series of holes numbered 5 to 8, whose distance at the bottom from E is W, the line of resistance adopted for all the holes excepting the breaking-in ones; thirdly, the series numbered 9 to 12, which are situated a distance W + (W sin 45°) from E; fourthly, the series numbered 13 to 16, having a radius 2 W from E; fifthly, the series numbered 17 to 20, whose radius from E is 2 W + W sin 45°; and sixthly, the series num-

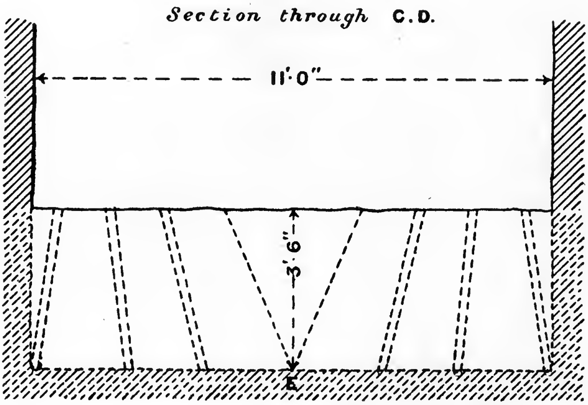

FIG. 36.

bered 21 to 24, whose radius from E is 3 W. The measurements from E are to the bottom of the respective holes. W varies approximately from 1 foot 6 inches to 2 feet 6 inches, according to the hardness and tenacity of the rock, for holes 3 feet 6 inches long and $1\frac{1}{2}$ inch diameter, and must be found by trial shots, or calculated from the coefficient C_a of the rock if this is known. The position of the bottom of the holes is easily found by dividing the circle by the four diameters A B and C D, and intersecting these lines by circles having radii W, 1·7 W, 2 W, 2·7 W.

The dotted lines show the lines of fracture at the bottom of the holes.

69. *Number of Breaking-in Shots required.*—In order to find the number of breaking-in shots required, when fired simultaneously, to unkey the rock, let us assume the line of resistance to be p W, and substituting this value for W_1 in the formula

$$\frac{W_1}{W} = \frac{m\,m_1\,d_1 + m_1\,d\,d_1 \text{ cosec } a}{m\,m_1\,d + m\,d\,d_1 \text{ cosec } b}$$

we get

$$p = \frac{m\,m_1\,d_1 + m\,d\,d_1 \text{ cosec } a}{m\,m_1\,d + m\,d\,d_1 \text{ cosec } b}.$$

It is most convenient to compare the power of "angled" holes with that of similar holes parallel to a free face. Hence, we should put $a = 90^\circ$ and cosec $a = 1$; $m = 12\,d$ and $m_1 = 12\,d_1$. By substituting these values in the above formula we obtain

$$p = \frac{13\,d_1}{(12 + \text{cosec } b)\,d} = \frac{13}{12 + \text{cosec } b} \cdot \frac{d_1}{d}.$$

As, however, all the shotholes can be so placed as to act as one hole of p times the diameter of hole equal to a line of resistance W if similarly angled to the free face, which will be the case if the holes meet at a point, as in Fig. 35, we can therefore put $\frac{d_1}{d} = G$ for the number of holes required to blast a line of resistance p W, wherefore

$$p = \frac{13}{12 + \text{cosec } b} \cdot G$$

and

$$G = \frac{12 + \text{cosec } b}{13} \cdot p.$$

If, for example, all the holes have the same diameter, and the line of resistance for the breaking-in shots is double that of the enlarging ones, or $p = 2$, and the angle of the breaking-in holes with the line of resistance $= 8°$, or cosec $b = 7{\cdot}19$, then

$$G = \frac{(12 + 7{\cdot}19) \times 2}{13} = 3 \text{ holes.}$$

70. *Side Cut.*—If there are natural side walls available in the driving of a heading or level, as occurs in the case of veins and lodes, or the strata are proceeding in the same line as the heading so as to present the edges in front, the rock should be unkeyed in the side, which is called the "side cut," an example of which is given in Fig. 37 in plan and Fig. 38 in section.

To enable the key charges to overcome the greatest possible line of resistance along the wall, or joint *a b*, when it is so tight that the gases from the explosions cannot escape through it, experience shows that the keyholes should have just sufficient length to strike the joint, and not extend beyond it. On the contrary, where there is a somewhat open joint or fissure along the wall the keyholes should not quite reach to the same, as the charges must be

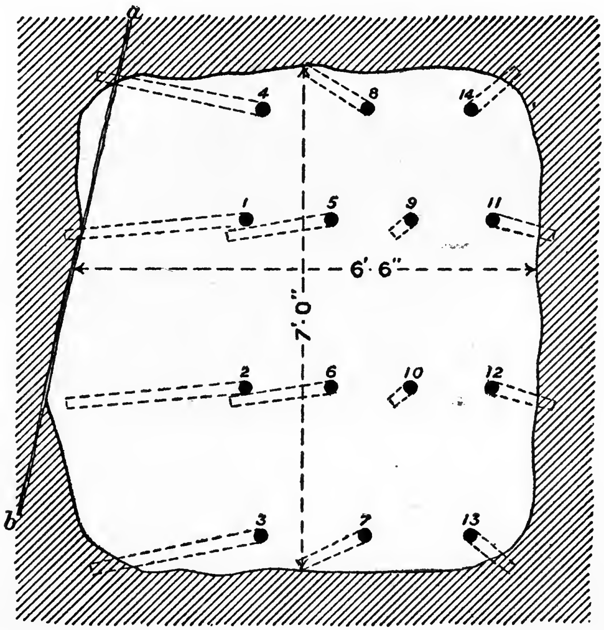

FIG. 37.

located in whole rock to give the best effect. In the case of a very open joint it should be considered in all respects as a free face, and the keyholes placed nearly parallel to it. The wall or joint *a b* may reduce the resistance to the breaking-in shots nearly one-half when it offers very little resistance along its surface to the rupture of the rock.

71. *Bottom Cut.*—Another method of unkeying the rock when there are no joints available is that illustrated in Fig. 39, which consists in placing several holes (*a*) along the side or bottom to assist the "angled" holes (*b*), to unkey the face A B and maintain the profile of the bottom of the heading

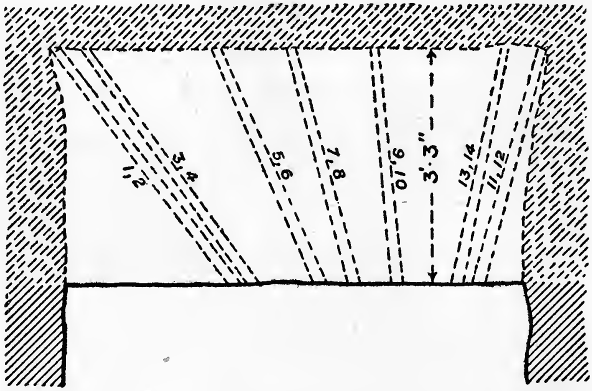

FIG. 38.

by firing charges in the holes (*a*) simultaneously with charges in the holes (*b*).

72. *Useful Formulæ for Determining the Number of Shotholes* required for headings or shafts of a given size may be found in the following manner:—

Let V denote the volume of rock blasted to advance a heading the length of the cut, and C the coefficient of the strength of the rock, then for the blasting power required to loosen the volume of rock V, we may put

$$P = C V.$$

Therefore, if the coefficient of the rock be C_1 instead of C, we shall have

$$P_1 = C_1 V.$$

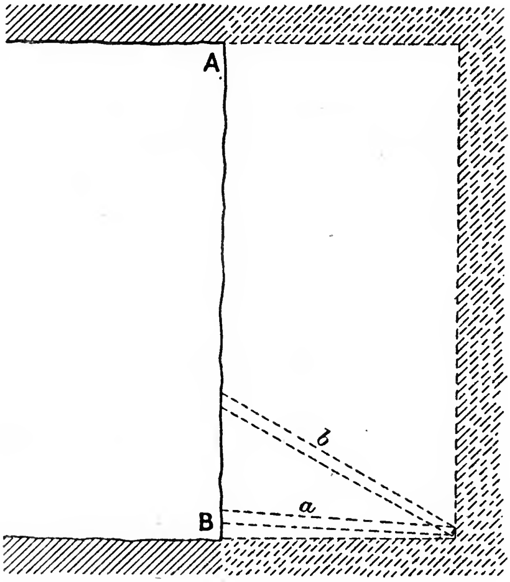

FIG. 39.

Consequently,

$$\frac{P_1}{P} = \frac{C_1}{C}.$$

But it is evident that

$$\frac{C_1}{C} = \frac{C_{a1}}{C_a},$$

and therefore that

$$\frac{P_1}{P} = \frac{C_{a1}}{C_a}.$$

Then, if N similar shotholes, having the same diameter and loaded with the same explosive, will develop a power P, and N_1 such shotholes a power P_1, we shall have

$$P = N\,M\,A, \quad \text{and} \quad P_1 = N_1\,M\,A,$$

M representing the pressure or shock on each unit of surface of the chambers from the explosion of the charges.

Therefore,

$$\frac{P_1}{P} = \frac{N_1}{N},$$

and

$$\frac{N_1}{N} = \frac{C_{a1}}{C_a}.$$

If for, say, a 3½ feet advance in a 7 × 7 foot heading twenty 1¼ inch diameter shotholes are required as illustrated in Fig. 35, then

$$C_{a1} = \frac{A}{S\,W_1} = \frac{18\frac{3}{4}}{32\frac{1}{2} \times 19} = \cdot 03.$$

Introducing these values for C_{a1}, and N_1 in the above formula, we have

$$N = C_a \frac{20}{\cdot 03} = 667\ C_a.$$

Assuming that N and N_1 are proportional to the volumes V and V_1 of rock for any given "advance" in a heading or "sink" in a shaft, then, since the volume of rock in the above example is $7 \times 7 \times 3\frac{1}{4} = 159\frac{1}{4}$ cubic feet, we may put

$$N = 667\, C_a \frac{V}{159\frac{1}{4}},$$

or

$$N = 4{\cdot}24\, C_a V.$$

This formula gives the number of similar shot-holes required in a heading or shaft, the coefficient of the rock for the explosive to be used being C_a, and the volume of rock being V for the "advance" or "sink."

CHAPTER XIV.

HOW TO FIND THE COEFFICIENTS C_a AND C_v BY TRIAL SHOTS.

73. *Trial Shots.*—The coefficient C_a in the formula $A = C_a S W$, and the coefficient C_v in the formula $L = C_v W^3$, must be found by trial shots in the rock which is to be blasted.

This is done by selecting a step of rock as free from fissures and joints as possible, with two fairly straight and smooth faces, A B and B C, at right angles to each other (Figs. 40 and 41), drilling several vertical holes a distance, say, 3 W apart, each of the same diameter (say 1 inch) from the face A B, parallel to the face B C, with varying lines of resistance, so that the depth of the holes $= W + \frac{m}{2}$, and the radius of the free face $> W$; then inserting a charge (with fuse) of the explosive compound whose strength is to be tested, say, for a length $= 8\,d$, filling the holes with moist clay tamping to the top, and firing the charges.

74. *Coefficient C_a.*—Suppose, for instance, that

three 1-inch holes, h_1 h_2 h_3 (Fig. 40), are bored in gneiss with lines of resistance 2 feet, 2 feet 6 inches and 3 feet, for which the corresponding depth of holes would be $D = 4d + W$, viz.

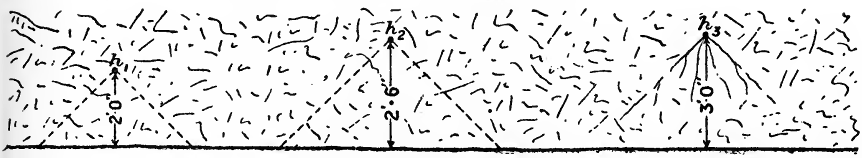

FIG. 40.

2 feet 4 inches, 2 feet 10 inches and 3 feet 4 inches respectively; $8d = 8$ inches of charge being used in each hole, with the result, for dynamite, that the 2 feet 4 inch and 2 feet 10 inch holes completely dislodged the rock, and that the 3 feet 4 inch hole only produced cracks, then, as for the holes which dislodged the rock, the resistance is greatest for the 2 feet 10 inch hole, and it has produced the desired effect, we may conclude that 2 feet 6 inches is the proper line of resistance for dynamite under the given conditions.

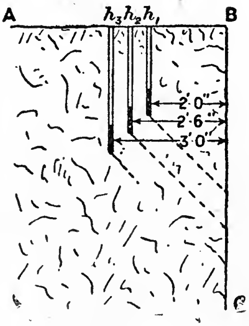

FIG. 41.

From the equation $A = C_a S W$, we have

$$C_a = \frac{A}{S W}.$$

Then, as A, according to the above trial shots, = 8 inches × 1 inch = 8 sq. inches, and S = 2 (8 inches + 1 inch) = 18 inches

$$C_a = \frac{8}{18 \times 30} = \cdot 015 \text{ nearly.}$$

To control this coefficient, and prove its correctness, calculate for larger and smaller holes according to the formula $A = \cdot 015\, S\, W$ for greater and smaller resistance than 2 feet 6 inches, and see if analogous results be obtained. If the strength of the rock should vary, then C_a has to be modified accordingly.

75. *Coefficient C_v.*—From the formula $L = C_v W^3$, we have

$$C_v = \frac{L}{W^3}.$$

According to the result of the above mentioned trial shots, the weight of charge L in ounces, taking the specific gravity of dynamite as 1·6, and the weight of a cubic inch of dynamite = ·036 × 1·6 × 16 = ·9216 oz., was

$$\begin{aligned} L &= \cdot 7854 \times \cdot 9216 \times 8 \times (1 \text{ inch})^2 \\ &= 5 \cdot 79 \text{ oz.} \end{aligned}$$

Then, as W = 2 feet 6 inches,

$$C_v = \frac{5 \cdot 79}{(2\tfrac{1}{2} \text{ feet})^3} = \frac{5 \cdot 79}{15 \cdot 625} = \cdot 37.$$

Supposing, however, that in very tight rock we required a $1\frac{1}{2}$ inch diameter borehole instead of a 1 inch to overcome the line of resistance 2 feet 6 inches, then we should have

$$C_v = \frac{5 \cdot 79 \times (1\frac{1}{2})^3}{(2\frac{1}{2} \text{ feet})^3} = \frac{19 \cdot 54}{15 \cdot 625} = 1 \cdot 25.$$

This high value of C_v is evidently solely due to the great resistance of cohesion of the rock, and the form of chamber, for it cannot be admitted that the charge required in this case is greater for the ejection of the rock after rupture has taken place, than in the former.

When the coefficient C_v, as in this case, is too high for the volume of rock blasted by a single shothole, if there be sufficient length and depth of free face, it may be reduced by arranging several similar shotholes closely in a line parallel to the free face, and equidistant apart, and firing them simultaneously. According to the formula

$$\frac{W_1}{W} = q = \frac{N\,S}{2\{(m + N\,d) + (N - 1)\,k\}}$$

the line of resistance will increase as the number of shotholes (within certain limits), and for a number N of $1\frac{1}{2}$ inch diameter shotholes, having a length of charge $12\,d$, which are placed a distance k, say 3 inches, apart, and which are singly equal to a line of resistance 2 feet 6 inches, for a given rock we shall have

(*a*) For $N = 1 \quad q = 1$ and $W = 2\frac{1}{2}$ feet.
(*b*) „ $N = 2 \quad q = 1\frac{3}{10}$ „ $W = 4$ „
(*c*) „ $N = 3 \quad q = 2$ „ $W = 5$ „

The relative volumes of rock blasted will be as under:

For (*a*) $W^3 = (2\frac{1}{2})^3 = 15\frac{5}{8}$ cubic feet.
„ (*b*) $W_1^3 = (4)^3 = 64$ „
„ (*c*) $W_1^3 = (5)^3 = 125$ „

and consequently, as 1 : 4 : 8.

Therefore, as $C_v = 1 \cdot 25$ for one shothole,

$C_v = \frac{2}{4} \times 1 \cdot 25 = 0 \cdot 625$ for two such shotholes;

$C = \frac{3}{8} \times 1 \cdot 25 = 0 \cdot 47$ for three such shotholes;

under the given conditions. On the contrary, the depths of the shotholes should be $W + \frac{m}{2}$, namely,

For (*a*) 2 feet 6 in. + 9 in. = 3 feet 3 in.
„ (*b*) 4 „ + 9 „ = 4 „ 9 „
„ (*c*) 5 „ + 9 „ = 5 „ 9 „

Consequently, the relative lengths of borehole for the rock blasted will be as follows:

For $(a) = 3$ feet 3 inches.

$$,, \quad (b) = \frac{2 \times 4\frac{3}{4}}{4} = 2 \text{ feet } 4\tfrac{1}{2} \text{ inches.}$$

$$,, \quad (c) = \frac{3 \times 5\frac{3}{4}}{8} = 2 \text{ feet } 2 \text{ inches.}$$

The above example shows that in hard rock a great saving may be effected in the boring of the rock, as well as in the quantity of explosive used, when the shotholes support each other.

CHAPTER XV.

THE TAMPING, OR STEMMING, OF SHOTHOLES.

76. *Results of Sir J. F. Burgoyne's Experiments on the Resistance of Various Kinds of Tamping.*—According to General Sir J. F. Burgoyne's extensive experiments on the resistance of various kinds of tamping to the action of powder charges in boreholes, clay dried to a certain extent is, all things considered, the best material for tamping; broken brick, tempered with a little moisture, the next best material; and rotten stone, without hard gritty particles of stone, as good as either; but the latter is generally objectionable on the ground that it is likely to lead to an occasional substitution, or mixture of hard gravel, which is subject to strike fire.

Sir J. F. Burgoyne obtained the following results by his experiments with powder charges:—

(*a*) "In holes of 1 inch diameter, charges of 2 ounces of powder will not blow out above 7 inches of clay tamping.

(*b*) "In holes of 2 inches diameter, charges of 2 ounces of powder will blow out about 18 inches of clay tamping, and not more.

(*c*) "In holes of 3 inches diameter, charges of 2 ounces of powder will not blow out above 19 or 20 inches of clay tamping.

(*d*) "Increase of charges does not produce a greater effect upon good tamping. For instance, 4 ounces of powder had scarcely, if at all, more effect than 2 ounces, so far as can be judged under the different circumstances.

(*e*) "When the rock is opened by the explosion the effect on a tamping of clay, or other tough material, is greatly reduced, the action upon the rock in opening appearing to be much more rapid than on the tamping; even where the rock is separated across the line of the hole itself the tamping is usually found adhering to the sides."

77. *Length of Tamping Required for Powder Charges.*—The latter is a very favourable circumstance in blasting, and for powder charges we may conclude that 7, 18 and 20 inches of tightly packed clay tamping is the minimum required for boreholes whose diameters are 1, 2 and 3 inches respectively.

78. *Tamping for High Explosives.*—For such explosives as the nitroglycerine compounds, which develop their full power instantaneously, there is

not time for the tamping to yield before the full shock of the gases is delivered upon the sides of the chamber, and if the shock is sufficient to burst the rock rupture will be effected before any considerable proportion of the force of the explosion is lost by the escape of gases out of the shothole. Hence, for such explosives a very light tamping, as, for instance, the hole filled with water, will act very efficiently. In general it is sufficient to use a tamping of water, but when this cannot be applied, as when the holes are horizontal or inclined above the horizon, a few inches of clay or paper pushed tightly home into the holes is all the tamping required.

CHAPTER XVI.

ON THE DIFFERENT METHODS OF ARRANGING BOREHOLES IN DRIVING AND SINKING.

79. *Systems of Placing Holes for Driving and Sinking.*—It is essential when rock drills are employed that the arrangement of the boreholes should be such as will allow of every facility for boring them with such machines, and also minimise the number of holes and weight of explosive necessary for a given advance in the heading, level or shaft. With this object in view the following two distinct systems of holes for driving and sinking are in use, viz. :—

(*a*) The centre cut, which consists of centre holes surrounded by others more or less concentric therewith, and angled so as to allow the explosive to remove, first, a centre core or key; second, the rock encircling the core.

(*b*) The square cut, in which the shotholes are mostly parallel to the sides of the heading, level or shaft, which is given a more or less rectangular form, the holes being angled so as to admit of the removal of, first, an entering wedge; second, of the rock on each side of the wedge. The core or wedge may be either removed at the centre, side or bottom.

80. *Diameter of Holes.*—In driving headings or sinking shafts experience shows that holes having a diameter varying from $\frac{3}{4}$ to $1\frac{1}{2}$ inch at bottom are most economical in hard rock if charged with the strongest high explosives, and, on the contrary, holes of larger diameter, say from $1\frac{1}{2}$ to $2\frac{1}{2}$ inches in diameter, and charged with a strong low and cheap explosive in weak rock.

All the holes in a heading or shaft should have the same diameter, and the best arrangement is to give an equal resistance of rock to each and placing each hole so as to get the full benefit of the free faces formed by the firing of the preceding holes.

81. *Best Length for an "Advance" in a Heading, Level or Shaft.*—In hard and tight rock the best length for an advance with each set of holes is one-half of the width or diameter of the heading, level or shaft, but it may be three-fourths of such width or diameter in soft and loose rock with advantage. The arrangement of holes must enable the whole of the rock to be blasted away within the limits of the heading or shaft.

82. *Key-Holes.*—It is obvious that the "key" holes should meet at the bottom and be fired simul-

taneously to give the best effect, as the resistance of the core is a minimum under these conditions.

83. *Centre Cut in a Heading.*—Figs. 42, 43, 44 and 45 give an example of the best method of placing holes of one diameter for the centre cut; Fig. 42 indicating their position on the face of the

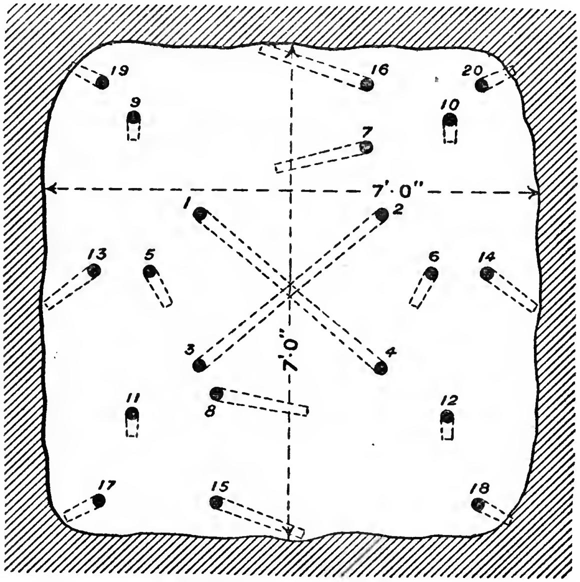

FIG. 42.

heading, Fig. 45 in elevation, and Figs. 43 and 44 in plan. The holes are shown as they would be drilled by rock-drills, and are so placed in accordance with the principle that the line of resistance must be the same for each hole except the breaking-in ones, that is, when they are all of the same diameter, and consequently equal to the same line of resistance. Sufficient holes must be bored to enable the whole section of rock *a b c d*, giving a lineal advance of 3 feet 3 inches, to be removed, as if this object were not attained it would be necessary to fix up the machines again simply to bore new holes for the removal of the remaining rock before another complete set of holes could be bored for a further

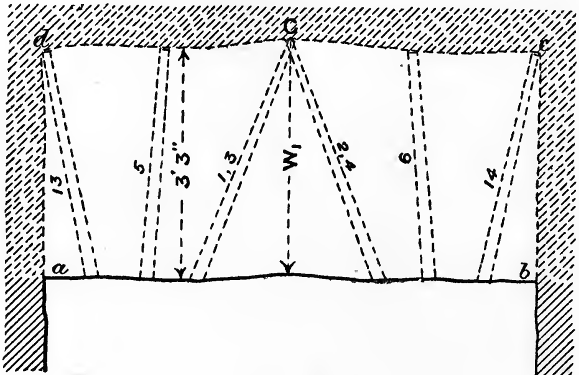

FIG. 43.

lineal advance of the heading. This system of holes enables two rock-drills to be employed most advantageously, and even four drills may be used at one time in a small heading when the greatest speed in driving is to be attained.

In the case of strong and hard rock, as, for instance, when the line of resistance is 1 foot 9 inches for $1\frac{1}{8}$ inch diameter holes, and they are charged with gelatine dynamite, 20 holes are required to remove the whole section of rock *a b c d*. The proper length of charge in each hole will be $12 \times 1\frac{1}{8} = 13\frac{1}{2}$ inches, which corresponds to, say, $\frac{3}{4}$ lb. of gelatine dynamite (*see* Table II.). Four breaking-in shots are generally adopted in practice, whereas three holes should suffice according to the formula $G = \left(\frac{12 \operatorname{cosec} b}{13}\right) p$, *b* being the angle made by the

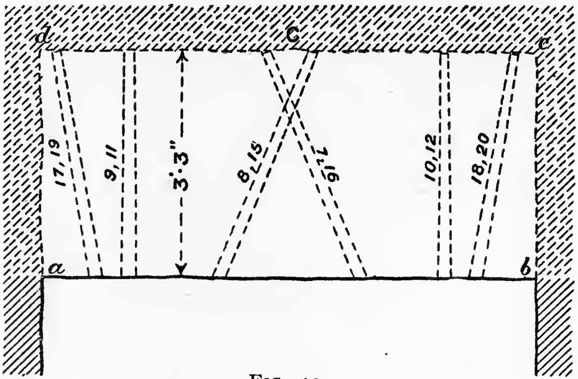

FIG. 44.

breaking-in shots with the line of resistance and *p* the ratio of the lines of resistance for the breaking-in and enlarging shots. These holes must converge to a point in the heading to ensure the removal of the core, as the resistance of the rock to rupture is thereby reduced to a minimum. This is found very difficult in practice, and it is therefore desirable to use an extra hole to ensure the removal of the core if these holes be not so placed. The order of firing the holes is as follows:—First, the breaking-in shots Nos. 1, 2, 3, 4, simultaneously; then the enlarging shots either consecutively or simultaneously (the result will be precisely the same) in the following order:—First volley, Nos. 5, 6, 7, 8; second volley, Nos. 9, 10, 11, 12; third volley, Nos. 13, 14, 15, 16;

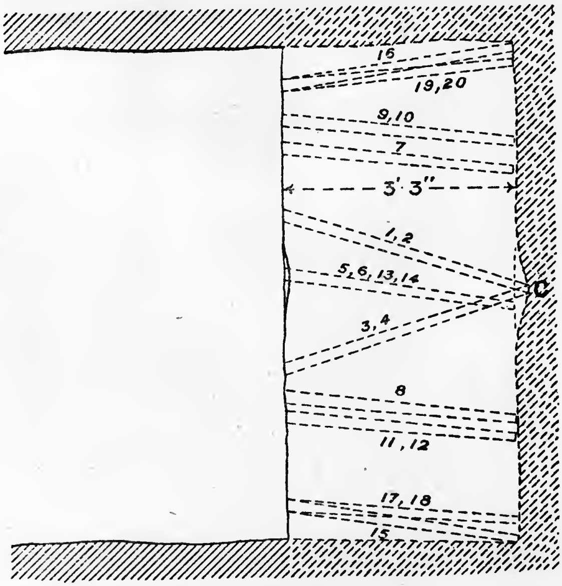

FIG. 45.

and fourth volley, Nos. 17, 18, 19, 20; this order giving the full advantage of the free face formed by the preceding holes, and therefore, the most economical result obtainable.

It is assumed in this case that the rock is fairly homogeneous, but on the other hand small irregular joints would not reduce the resistance appreciably, and should not be considered in the arrangement of the holes.

In the case of weaker or bedded rock the line of resistance would be greater, and fewer holes required.

The proper charge of gelatine dynamite for a $1\frac{1}{8}$ inch diameter borehole is $\frac{3}{4}$ lb., and, therefore, the weight of explosive required for a lineal advance of 3 feet 3 inches is

$$20 \times \tfrac{3}{4} \text{ lb.} = 15 \text{ lbs.}$$

84. *Square Cut in a Heading.*—Figs. 46, 47, 48 and 49 give an example of the best method of placing the shotholes for the square cut; Fig. 46 illustrating the position of the holes on the face of the heading; Fig. 49 in elevation; and Figs. 47 and 48 in plan. This system of placing the holes differs essentially from the centre cut, in that the length of free face is at first developed instead of being successively increased by each series of shots as in the centre cut. Two or four rock drills can be used most advantageously in this case as with the centre cut, whereas the arrangement of the holes is simpler, and facilitates the boring of the same.

Under the same conditions as in the example given of the centre cut system, viz. in strong and hard rock, with $1\frac{1}{8}$ inch diameter boreholes, and the line of resistance being 1 foot 9 inches, 22 holes are required for a lineal advance of 3 feet 3 inches, but four of the holes are shorter, and only three dry holes have to be bored as compared with five dry holes with the centre cut system. As the holes are bored slightly conical, the shorter holes will have a

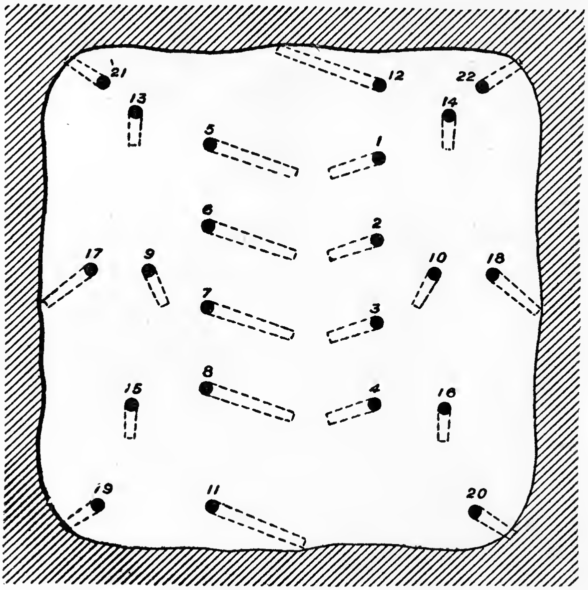

FIG. 46.

diameter of $1\frac{1}{4}$ inches at bottom, as against $1\frac{1}{8}$ inch for the longer holes.

The entering wedge *e f g* (Fig. 47) is best removed in two stages, namely, first the part *e h g* by breaking-in shots Nos. 1, 2, 3, 4, and then the part *e f h*, by breaking-in shots Nos. 5, 6, 7, 8, as the line of resistance would be more than these eight holes could overcome, if Nos. 1, 2, 3, 4 were continued so as to meet Nos. 5, 6, 7, 8, and fired simultaneously.

It has been before demonstrated that if q represents the line of resistance for N shotholes placed a distance k apart in line parallel to a free face, and

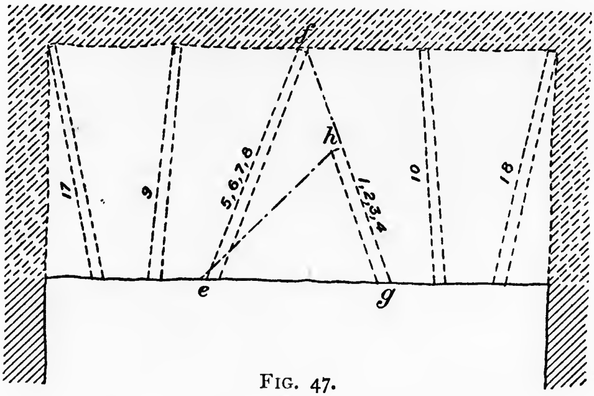

FIG. 47.

fired simultaneously, the line of resistance being unity for one shothole, that we shall have

$$q = \frac{\text{N S}}{2\{(m + \text{N}d + k(\text{N} - 1)\}}.$$

But in the example we have N = 4, S = $29\frac{1}{4}$ inches, $m = 13\frac{1}{2}$ inches, $k = 14$ inches, and $d = 1\frac{1}{8}$ inch for the four longer holes, and $1\frac{1}{4}$ inches for the shorter holes; therefore, for the four holes Nos. 5, 6, 7, 8, we have

$$q = \frac{4 \times 29\frac{1}{4}}{2\{13\frac{1}{2} + 4 \times 1\frac{1}{8} + 14(4 - 1)\}} = \frac{117}{120} = 0 \cdot 9.$$

Hence the line of resistance these four holes will break by their combined shearing action is 1 foot 9 inches × 0·9 = 1 foot 7 inches, which is less than for each fired singly, and their line of resistance,

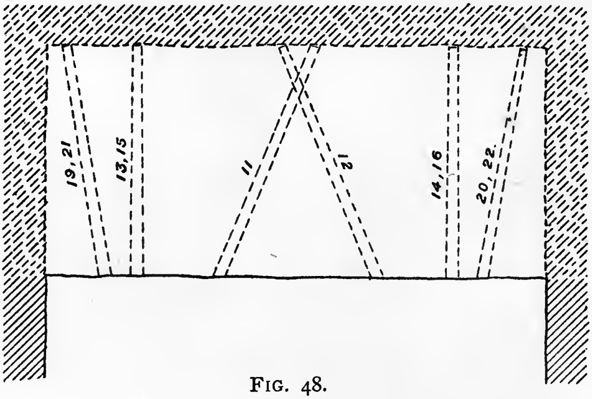

FIG. 48.

therefore, cannot exceed that corresponding to each, viz. 1 foot 9 inches.

On the other hand, if Nos. 1, 2, 3 and 4 were continued to meet Nos. 5, 6, 7, 8 we should have four pairs of holes supporting each other when fired simultaneously, the relative power of each pair being

$$\frac{2 \times 29\frac{1}{4}}{2(13\frac{1}{2} + 2 \times 1\frac{1}{8} \text{ inch})} = \frac{58\frac{1}{2}}{31\frac{1}{2}} = 1 \cdot 85.$$

Consequently, for the four pairs of holes fired simultaneously, we shall have

$$q = 1\cdot85 \times 0\cdot9 = 1\cdot67,$$

that is, their combined shearing power is not equal

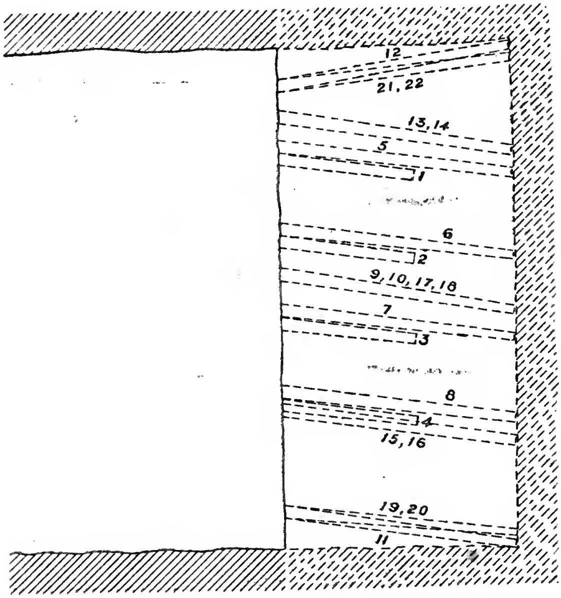

FIG. 49.

to a greater line of resistance than

$$1 \text{ foot } 9 \text{ inches} \times 1\cdot67 = 2 \text{ feet } 11 \text{ inches},$$

which is reduced by the small angle of the holes, namely, 20° with the line of resistance, to 2 feet 11 inches × ·88 = 2 feet 7 inches, whereas the actual line of resistance is 3 feet 3 inches. If the line of resistance be 1 foot 9 inches for $1\frac{1}{8}$-inch holes, it will be 2 feet for the $1\frac{1}{4}$-inch diameter holes, Nos. 1, 2, 3, 4, but as these holes are placed at a less angle than 90° with the line of resistance, they are only equal to a line of resistance of 2 feet × ·88 = 1 foot 9 inches.

It will therefore be seen that there is a saving in the expense of boring by the shorter No. 1, 2, 3, 4 holes, and moreover the charge is better placed for doing the work, so that a better result is obtainable.

The order of firing the shotholes is as follows :—

1st volley, Nos. 1, 2, 3, 4, simultaneously.
2nd volley, Nos 5, 6, 7, 8, simultaneously.
3rd volley, 9, 10, 11, 12, either simultaneously or consecutively.
4th volley, Nos. 13, 14, 15, 16, either simultaneously or consecutively.
5th volley, 17, 18, 19, 20, either simultaneously or consecutively.

The effect will be precisely the same whether the enlarging shotholes are fired simultaneously or consecutively.

85. *Side Cut in Headings.*—The side cut offers the very important advantage, when only one rock drill is employed in a heading, that all the holes may be drilled most advantageously with one fixing of the tunnel column. It is specially applicable when

the heading is proceeding in the same line with vertical strata, or in a vein or lode, as this will enable the breaking-in shots to be located in the most favourable position for "unkeying" the rock along a joint or wall on the side. Figs. 50 and 51

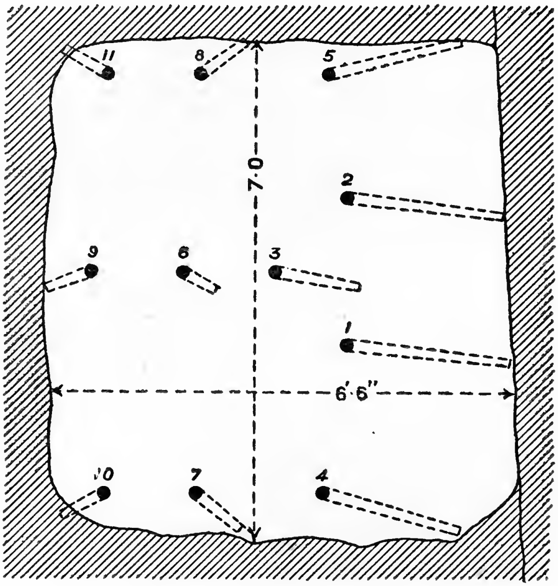

FIG. 50.

are an example of how the holes are placed in moderately strong rock, there being supposed to be a joint or wall at the side; where there is no joint or wall to facilitate the "unkeying" of the rock, two additional breaking-in shotholes, 1 and 2 as shown in Fig. 54, will be necessary.

The line of resistance being 2 feet 7 inches for a 1 inch diameter hole charged with gelatine dynamite, eleven 1-inch shotholes are necessary to remove the section of rock *a*, *b*, *c*, *d*, giving an advance of 3 feet 3 inches in a heading 7 feet × 6 feet 6 inches. The holes should be fired in the following order:—

1st volley, Nos. 1, 2, simultaneously.
2nd volley, Nos. 3, 4, 5, consecutively.

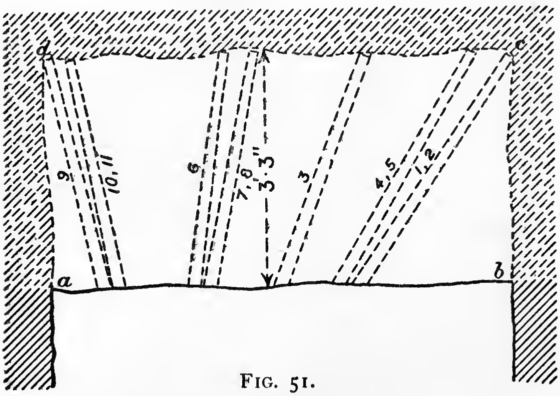

FIG. 51.

3rd volley, Nos. 6, 7, 8, consecutively.
4th volley, Nos. 9, 10, 11, consecutively.

Figs. 52 and 53 are another example of the side cut when the line of resistance is 2 feet 3 inches, or for strong rock in case there is a joint or wall at the side. The number of 1 inch diameter holes required under these conditions for a lineal advance of 3 feet

3 inches is 14, which should be fired in the following order :—

1st volley, Nos. 1, 2, simultaneously or consecutively.

2nd volley, Nos. 3, 4, 5, 6, consecutively.

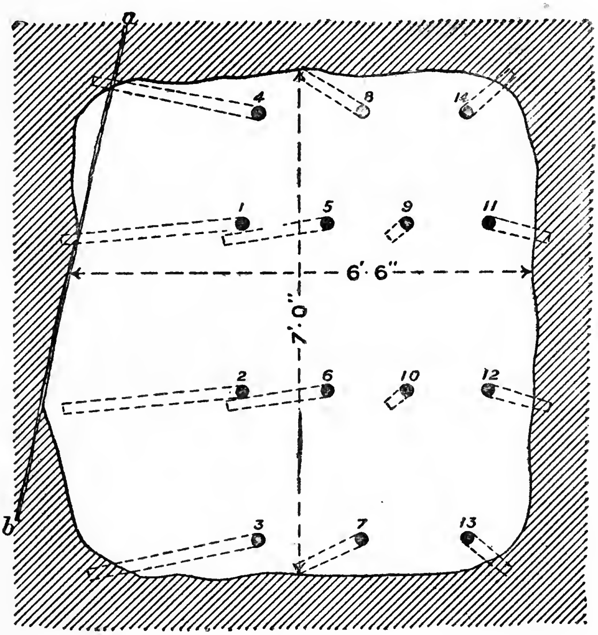

FIG. 52.

3rd volley, Nos. 7, 8, 9, 10, consecutively.

4th volley, Nos. 11, 12, 13, 14, consecutively.

Figs. 54, 55 and 56 are a further example of the side cut for a heading 7 feet × 6 feet 6 inches, when the rock is very hard and tight, 23 $1\frac{1}{8}$ inch diameter holes being employed. This diameter hole is equal to a line of resistance of 1 foot 3 inches in such rock when charged with gelatine

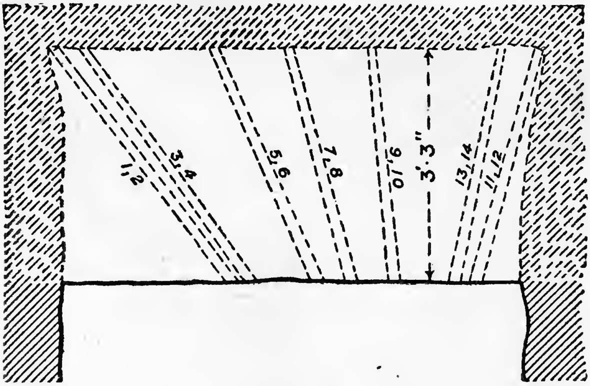

FIG. 53.

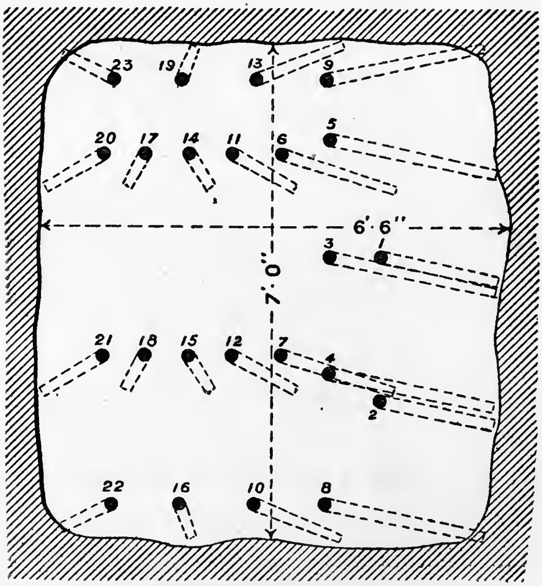

FIG. 54.

dynamite. Nos. 1, 2, 3, 4 are the breaking-in shots, Nos. 1 and 2 being fired first simultaneously, and then Nos. 3 and 4 simultaneously. The enlarging

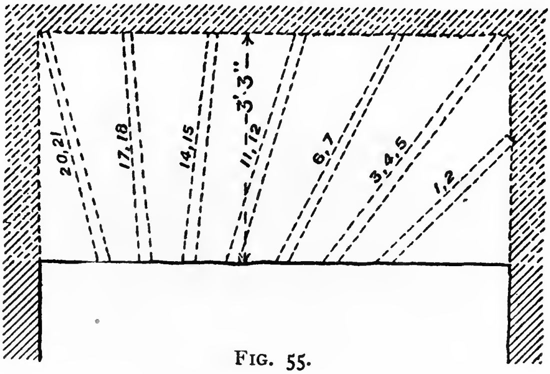

FIG. 55.

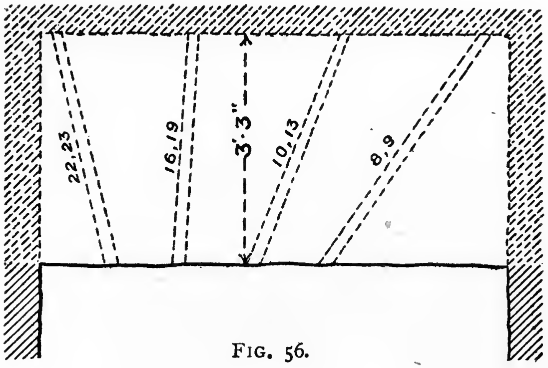

FIG. 56.

shots are then fired consecutively as follows: first, Nos. 5, 6, 7, 8; second, Nos. 9, 10, 11, 12; third, Nos. 13, 14, 15, 16; fourth, Nos. 17, 18, 19; fifth, Nos. 20, 21, 22 and 23. The 23 holes will advance the heading 3 feet 3 inches, the charge for each hole being about $\frac{3}{4}$ lb., and for the advance of 3 feet 3 inches, $23 \times \frac{3}{4} = 17\frac{1}{4}$ lbs. of gelatine dynamite.

86. *Square Cut in a Shaft or Rise.*—For sinking a rectangular shaft or driving up a rise the square cut is best adapted, as it facilitates the boring of the holes. In a shaft 14 feet × 8 feet, if the coefficient

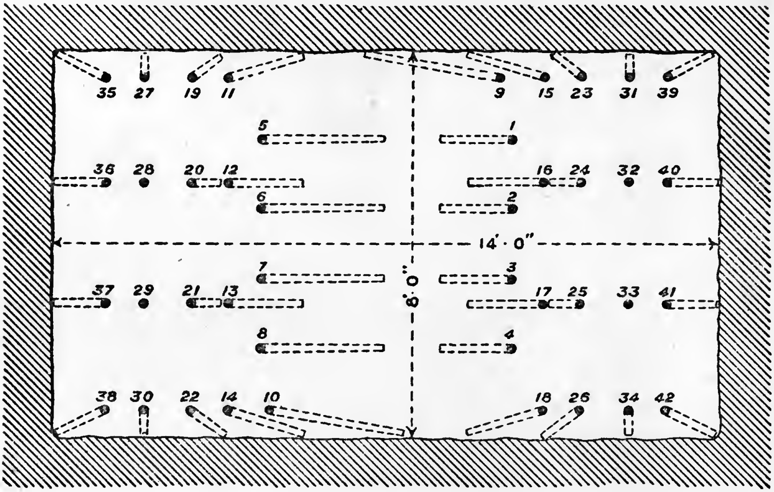

Fig. 57.

of the rock is ·024 for the explosive to be used, and $1\frac{1}{8}$ inch diameter holes be bored, to sink the shaft 3 feet 6 inches, 42 holes will be required as indicated in Figs. 57, 58, 59, 60, 61, the holes being so placed as to give a line of resistance of 1 foot 9 inches to each, in accordance with the formula $W = \frac{A}{C_a S}$. The best result will be obtained when the charges are fired in the following order :—

1st volley, Nos. 1, 2, 3, 4, simultaneously.
2nd volley, Nos. 5, 6, 7, 8, simultaneously.

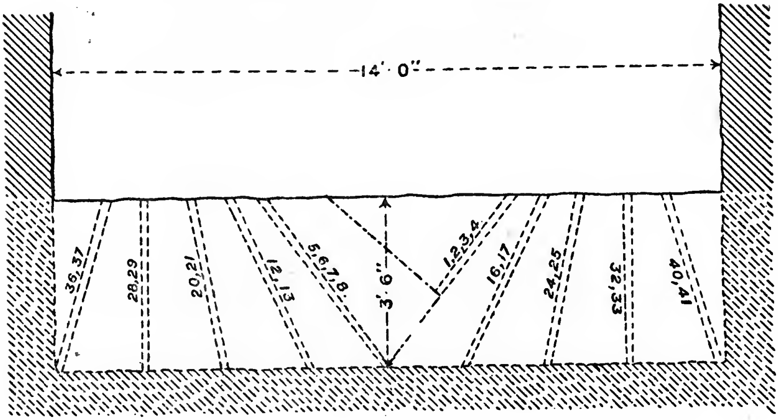

FIG. 58.

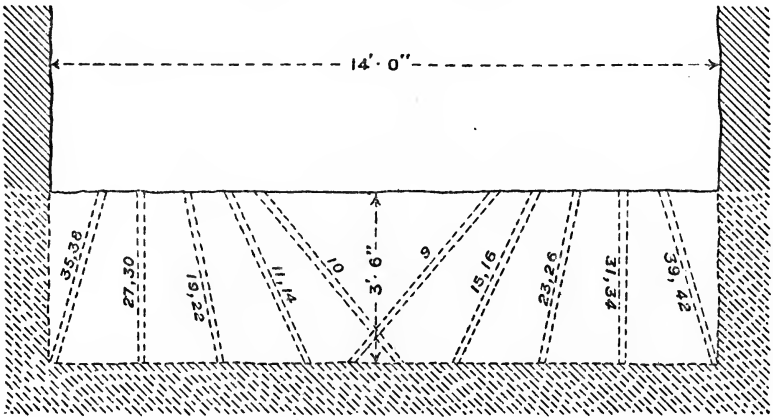

FIG. 59.

3rd volley, Nos. 9, 10, consecutively.
4th volley, Nos. 11, 12, 13, 14, 15, 16, 17, 18, simultaneously.

5th volley, Nos. 19, 20, 21, 22, 23, 24, 25, 26, simultaneously.

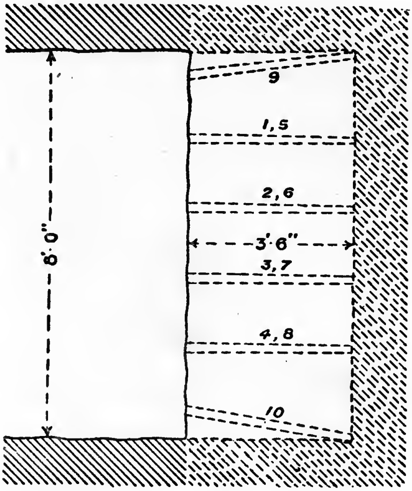

FIG. 60.

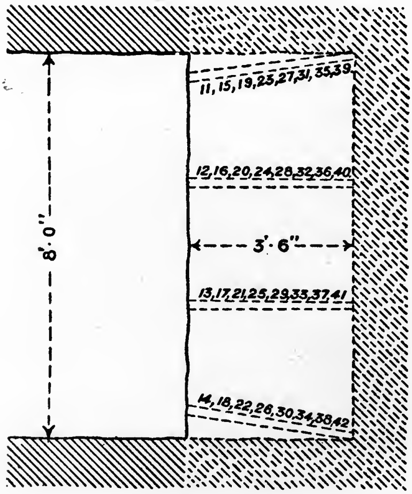

FIG. 61.

6th volley, Nos. 27, 28, 29, 30, 31, 32, 33, 34, simultaneously.

7th volley, Nos. 35, 36, 37, 38, 39, 40, 41, 42, simultaneously.

If the coefficient of the rock has been found for gelatine dynamite, each hole should have a charge of $\frac{3}{4}$ lb. of this explosive, and consequently the total

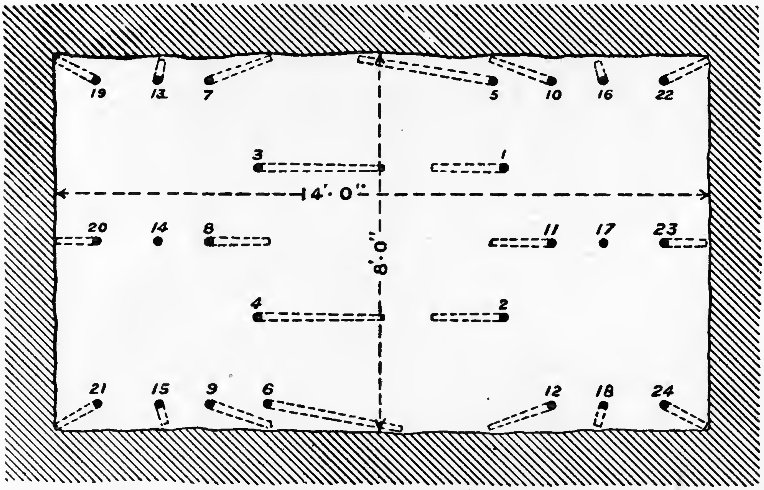

FIG. 62.

weight of gelatine dynamite required to sink the shaft 3 feet 6 inches would be

$$\tfrac{3}{4} \text{ lb.} \times 42 = 31\tfrac{1}{2} \text{ lbs.}$$

If, however, the coefficient of the rock is ·018 for gelatine dynamite, or the line of resistance is 2 feet 4 inches for a $1\frac{1}{8}$ inch diameter hole, then only 24 such holes will be required as illustrated in Figs. 62, 63 and 64. The charge for each hole will be the same as in the preceding example, viz. $\frac{3}{4}$ lb., and the charges fired as follows:—

1st volley, Nos. 1, 2, simultaneously.
2nd volley, Nos. 3, 4, simultaneously.
3rd volley, Nos. 5, 6, consecutively.

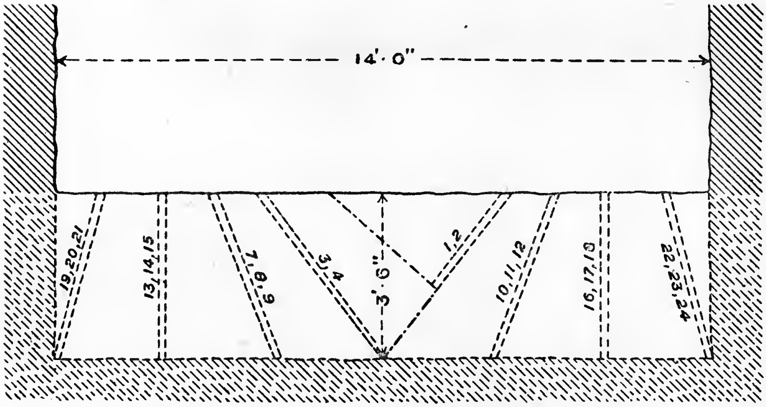

Fig. 63.

4th volley, Nos. 7, 8, 9, 10, 11, 12, simultaneously.

5th volley, Nos. 13, 14, 15, 16, 17, 18, simultaneously.

6th volley, Nos. 19, 20, 21, 22, 23, 24, simultaneously.

The total weight of gelatine dynamite required for sinking 3 feet 6 inches in this case is $\frac{3}{4}$ lb. × 24 = 18 lbs.

87. *Centre Cut in a Circular Shaft.*—On the other hand, the centre cut is most suitable for sinking a circular shaft. For instance, if it is required to sink a circular shaft 10 feet in diameter, and the coefficient of the rock is ·021 for gelatine dynamite, or corresponding to a line of resistance of 1 foot 10 inches for $1\frac{1}{8}$ inch diameter holes, 24 such holes

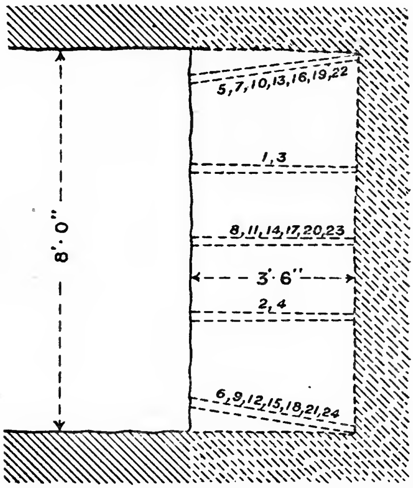

FIG. 64.

will be necessary to sink the shaft 3 feet 6 inches, if placed as indicated in Figs. 65, 66, 67. The charges should be fired in the following order :—

1st volley, Nos. 1, 2, 3, 4, simultaneously.
2nd volley, Nos. 5, 6, 7, 8, simultaneously or consecutively.

3rd volley, Nos. 9, 10, 11, 12, simultaneously or consecutively.

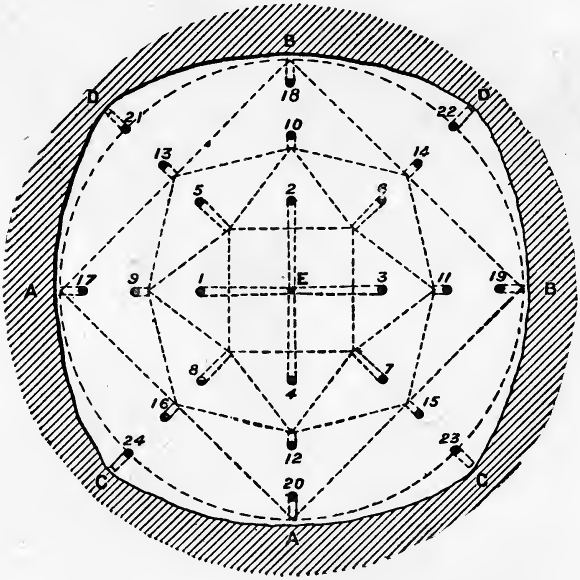

FIG. 65.

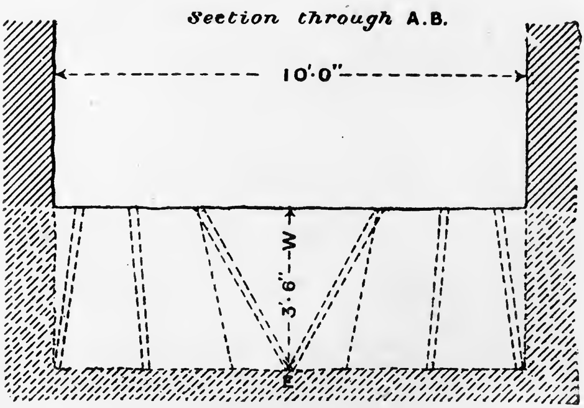

FIG. 66.

4th volley, Nos. 13, 14, 15, 16, simultaneously or consecutively.

5th volley, Nos. 17, 18, 19, 20, simultaneously or consecutively.

6th volley, Nos. 21, 22, 23, 24, simultaneously or consecutively.

A very good result is also obtained when all the holes comprising the last two volleys are fired simultaneously.

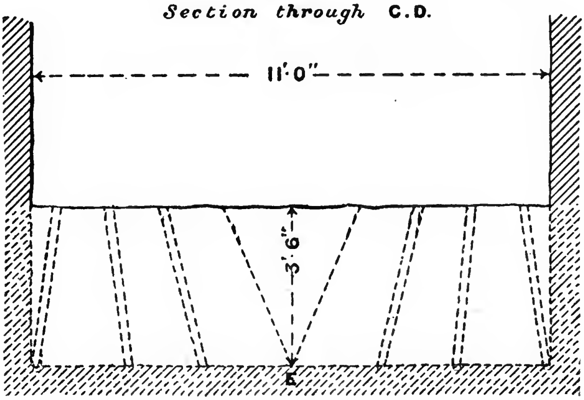

FIG. 67.

The total weight of explosive required for sinking the shaft a depth of 3 feet 6 inches is

$$24 \times \tfrac{3}{4} \text{ lb.} = 18 \text{ lbs.}$$

88. *Diagrams of Holes for Headings and Shafts.*—It is obvious that the diagrams are applicable to any length of line of resistance, if such length be taken as the scale of the diagrams.

CHAPTER XVII.

SAFETY FUSE.

89. *The "Miner's Safety Fuse"* was invented by William Bickford of Tuckingmill, Cornwall, and patented by him in 1831, to provide means of conveying fire to the charge in blasting, which would obviate the various dangers then inseparable from that operation, and thus avoid the distressing accidents constantly occurring in the mining district in which he resided, chiefly from premature explosion of charges.

Since its invention it has been greatly improved, and is now universally employed as the means of firing the charge in every kind of mining, quarrying, and subaqueous blasting, both with powder and the various modern explosives.

Good safety fuse will burn with certainty at the rate of about 2 feet per minute, and be capable of resisting considerable pressure without injury. For wet ground, it should be of such a character as to admit of being used without protection.

90. *Principal Kinds of Fuse.*—Safety fuse may be described as a cord of hemp or gutta-percha,

carrying a column of fine gunpowder in its centre (see Fig. 68), and covered with one or more coats of tape and varnish. The following are the principal kinds made by Messrs. Bickford:—

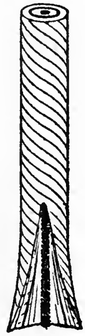

FIG. 68.

No. 1 (*Small Safety Fuse*) is the cheapest and smallest kind of fuse made, and is recommended only for immediate use in dry ground.

No. 2 (*Safety Fuse*) is adapted for all ordinary blasting in dry ground.

No. 3 (*White Safety Fuse*) is adapted for use in dry ground and in close places, as it emits little smoke in its combustion. It burns without flame and is, therefore, especially suited for use in collieries.

No. 4 (*Red Safety Fuse*) is somewhat similar to No. 3, but it is rendered considerably more damp-proof.

No. 5 (*Double Wove Fuse*). This is a black varnished fuse, intermediate in size between Nos. 2 and 6, and in damp-resisting power between Nos. 4 and 6. It is a very serviceable fuse for general purposes.

No. 6 (*Thread Sump Fuse*). This fuse is adapted for use in damp ground, and is so prepared as to resist the action of any moderate degree of humidity. It will bear rougher treatment, both before it is used and during the process of tamping, than the preceding kinds, and is strongly recommended where careful handling by the operatives cannot be relied on.

No. 7 (*Small Tape Fuse*). This consists of the addition of a coating of tape and extra varnish to No. 1 fuse, to enable it to stand a much greater degree of wet and pressure, but it is only recommended for immediate use and home service.

No. 8 (*Tape Sump Fuse*). This fuse is adapted for use in wet ground. It is so protected as to operate efficiently when the tamping is saturated with water.

No. 9 (*Double Tape Sump Fuse*). This fuse is intended for blasting in very wet ground.

Nos. 10 and 11 (*Treble-Coun'ered and Thread-Countered Fuse*). These fuses are covered with a larger quantity of protecting material than any of the preceding, and are, therefore, better able to withstand the roughest treatment and greatest pressure. Their increased diameter has prevented their use in some cases, but they have often been found extremely useful when the nature of the ground is not only wet but rough, and the tamping substance gritty and liable to cut. No. 10 is rather the tougher of the two, and No. 11 the more waterproof. They are both designed for home service only.

No. 12 (*Small Gutta-percha Fuse*). This is the smallest kind of gutta-percha fuse. It is extensively used as a substitute for No. 13, but it is not recommended to be stored for any considerable time before use, nor in any case for deep subaqueous blasting.

No. 13 (*Gutta-percha Fuse*). This fuse is adapted for subaqueous blasting, where it is not liable to much motion from waves or currents, nor subjected to great pressure. It has answered its intended purpose after it has been under water for 24 hours, with a pressure of 40 lbs. to the inch, this being equivalent to the weight of water at a depth of more than 90 feet.

N.B. 100,800 coils of this fuse were used in excavating the Manchester Ship Canal.

No. 14 (*Tape Gutta-percha Fuse*). This fuse consists of the application to No. 13 fuse of an exterior protecting coating of tape and composition varnish, which not only somewhat increases its water-resisting properties, but delays the oxidation of the gutta-percha; it consequently retains its efficiency for a much longer time, and is therefore well adapted for service in distant countries. This fuse is extensively used by the Indian and Foreign Governments, and is highly recommended for use in tropical countries, owing to the effect of these climates in oxidising unprotected gutta-percha. It is made in two sizes.

No. 15 (*Double Gutta-percha Fuse*). This fuse is far stronger and more waterproof than Nos. 12, 13 and 14, being protected and strengthened by an additional coating of gutta-percha and other material. It will act in a greater depth of water, and bear a greater strain than the preceding. After having been twenty-four hours under water, it burns freely with a pressure of 140 lbs. to the inch, this being equivalent to the weight of water at a depth of more than 300 feet. It is made in two sizes.

No. 16 (*Impermeable Subaqueous Fuse*). This fuse is specially made to order for the deepest submarine blasting.

Nos. 17, 18, 19 and 20 (*Metallic Fuses*). These fuses have many objectionable features, and cannot be recommended.

No. 21 (*Gutta-percha Countered Metallic Fuse*). This fuse, like the four preceding, has many objectionable features, and will not withstand rough treatment. It has, however, been found valuable where resistance to great pressure is required, but without severe tension, and may be said to provide the maximum resistance to pressure, relatively to a minimum of diameter.

No. 22 (*Treble Wove Fuse*). This fuse has been extensively used, and is found well suited to all the ordinary requirements of blasting in mines, quarries and railway works. It combines a waterproofing almost equal to No. 9, with great toughness and firmness, and a small diameter.

No. 23 (*White Tape Fuse*). This fuse is similar to No. 8, having many of the advantages of No. 3, together with a degree of waterproofing almost equal to No. 8. The varnish of this fuse is not susceptible to effect from a high temperature; and it is therefore strongly recommended for use in warm climates.

No. 24 (*White Double Tape Fuse*). This fuse is similar in construction to No. 9, but, having a white exterior varnish, is more suited for use in hot countries.

No. 25 (*Colliery Fuse*). This fuse was designed as a means of obviating the dangers previously attending the use of safety fuse in fiery or gaseous collieries, as pointed out in the Report of the Royal Commission on Accidents in Mines (1886), and to meet the requirements of the new Colliery Acts. It is guaranteed to burn without emitting flame or sparks laterally; and, used in conjunction with Bickford's patent safety lighter and patent nippers, is one of the best as well as one of the simplest means for conveying fire to the water-cartridge or other charge in the most fiery or gaseous pits with safety.

For certain slate quarries in England and Wales, where very deep boreholes, charged with unusually large quantities of powder, are employed, it is an object of great importance to convey the fire to the bottom of the charge, instead of exploding it at or near the top, as would be the case with ordinary fuses, for which purpose the colliery fuse No. 25 is most suitable.

No. 26 (*White Treble Wove Fuse*). This fuse consists of a substitution of white varnish for black on a fuse similar to No. 22. It possesses the same general suitability to all ordinary blasting requirements, together with special adaptation for exportation to tropical climates, changes of temperature having no deteriorating effect on the covering.

No. 27 (*White Countered Gutta-Percha Fuse*). This fuse consists of the application of a yarn covering with an exterior white varnish to a small gutta-percha fuse. Its advantages are that it is rendered waterproof by its gutta-percha covering, whilst its protecting yarns make it tough and durable, and the finishing varnish does not become either adhesive or brittle under extremes of temperature.

N.B.—138,369 coils of this fuse were used in driving the Severn Tunnel, and 75,150 coils in the excavation of the Manchester Ship Canal.

Those most commonly employed with dynamite, blasting gelatine, gelignite, gelatine-dynamite, tonite, and the other high explosives, are Nos. 3, 4, 5, 8, 9, 13, 14, 23, 24, 25, 26 and 27.

91. *Applications of the Different Fuses.*—The fuses of the above list are applicable as follows:—

For dry ground, Nos. 1, 2 and 3.

For damp ground, Nos. 4, 5 and 6.

For wet ground, Nos. 7, 8, 23 and 26.

For very wet ground, Nos. 9, 10, 11, 12, 17, 22, 24 and 27.

For under water, Nos. 13, 14, 15, 18, 19 and 20.

For deep water, Nos. 15, 16 and 21.

For slate quarries, Nos. 1, 2, 3, 7 and 25.

For fiery collieries, No. 25.

For detonators, Nos. 3, 4, 5, 7, 8, 13, 14, 23, 25, 26 and 27.

92. *Method of Using Fuses.* — With regard to the method of using safety fuse for igniting powder, one end of the required length of same is placed in the charge, and the hole then tamped with any soft substance (for example clay or rotten-stone), which will not cut the fuse. When the blasting agent is one of the high explosives a suitable detonator must be placed on the end of the fuse which enters the charge. After the charge has been properly tamped, the other end of the fuse should be directly ignited, and it will then slowly and surely burn to the charge if care has been taken to select that quality which the operation requires.

In all subaqueous operations, great care must be taken that, at the union of the fuse and cartridge, there is a perfectly watertight joint; and the fuse must be strengthened as much as possible both at and near the place of junction, to guard against the breaking of the joint, which would allow the water to soak into the charge.

93. *Selection of Fuses for Different Climates.*—

Fuses Nos. 2, 5, 6, 8, 9 and 27 are supplied in a form specially suitable for exportation into either warm or cold climates. That varnish which would be suitable for a cold country becomes soft and sticky if exposed to much heat; while that suitable for a hot country becomes hard and brittle if exposed to great cold. This inconvenience is remedied by the special preparation of the varnish to suit any given temperature. Nos. 3, 4, 25 and 26 fuses need no such special preparation, being naturally suited for any climate. Nos. 23 and 24 fuses are perfectly safe in hot, but are not recommended for very cold climates. Of the gutta-percha fuses, No. 14 is the kind recommended for export.

94. *Storing of Fuses.*—The fuses should be kept in a dry room, so that the powder may not be affected by damp, and they will retain their efficiency until the varnish has lost its essential oil. Care should be taken that they are not touched by any greasy or oily matter, as this rapidly penetrates through the varnish to the gunpowder, and prevents the proper burning of the fuse. Gutta-percha fuses, if kept as here described, will retain their efficiency as long as the gutta-percha does not become brittle through oxidation.

If, through exposures to cold, the tar-varnish should become brittle or crack, so that there is a danger of the column of gunpowder in the centre of the fuse becoming damped in use, it may be remedied by very slightly greasing the varnish of the fuse immediately before it is used; and if, through exposure to heat, it should become soft and sticky to the touch, this may be obviated by rubbing into it a little whiting or any other similar powder.

95. *Fuse Lighter for Collieries.*—Bickford's patent fuse lighter is an invention to ignite fuse without exposing spark or flame, and consists of a tin tube containing a tiny glass bead or tube filled with sulphuric acid, and embedded in a chlorate mixture. The fuse having been inserted into the open end, and the mouth of the lighter closed firmly, but not too tightly around it, is lighted by squeezing the centre of the tube with a suitable nippers supplied therewith, in consequence of the chemical action which takes place inside the tube on the release of the acid by the crushing of the glass bead, but which produces no objectionable external effect.

These lighters are extensively used in the principal collieries, and the manufacturers assert that their colliery fuse, lighters and nippers together, provide the simplest and most economic means yet devised for conveying fire to the charge in gaseous or fiery pits with absolute security. The price of the lighter is such as to place it within the reach of every user of explosives.

96. *Patent Volley-Firer and Instantaneous Fuse.*—The advantages of simultaneous firing, which were until recently obtainable only by means of the electric machines and appliances described in

Chapter XVIII., are now offered by the use of Bickford's volley-firer and instantaneous fuse.

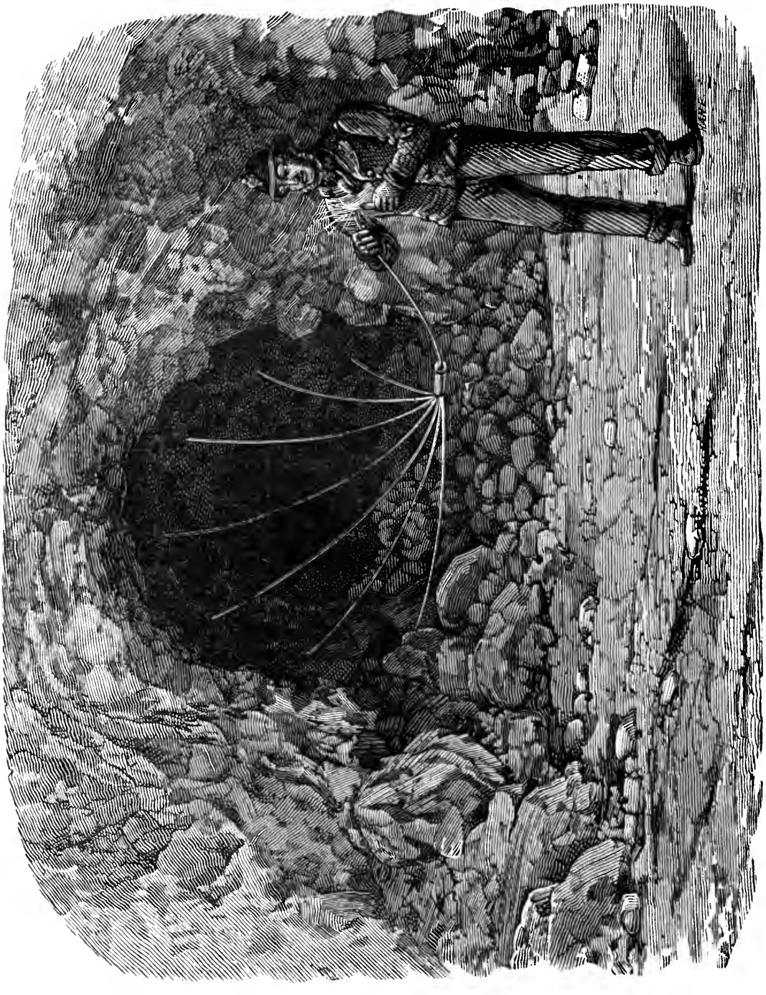

Fig. 69.

The invention is exceedingly simple (see Fig. 69), a number of instantaneous fuses being united in one recipient, termed the volley-firer, and simultaneously fired by means of a single ordinary safety fuse, which enters the volley-firer at the opposite end, as shown in the accompanying illustration.

The volley-firers are supplied in forms and sizes suitable for all the various conditions of mining, quarrying and tunnelling, and containing any number of instantaneous fuses from two or three to sixteen (which may be of equal or different lengths), those containing even a larger number having frequently been used with greater success; a single report being invariably given by the explosion of the different charges.

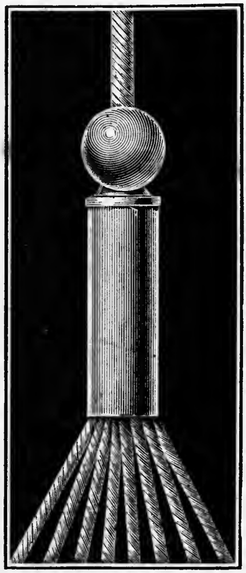

FIG. 70.

The volley-firers are made in two ordinary forms. The one illustrated in Fig. 70 is the pattern used in general blasting in mines; and the other, or T pattern, Fig. 71, is more especially adapted for quarry and other surface work where it is sometimes desirable to bring down a large face of rock.

To enable operatives to adapt the instantaneous fuses to any variable length suiting particular operations, the inventors supply the volley-firers with fuses looped as shown in Fig. 72, so that if the whole length of the fuse so looped is, say, 10 feet,

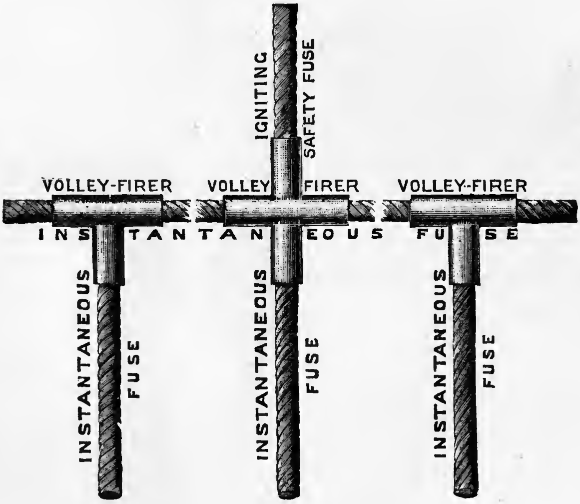

FIG. 71.

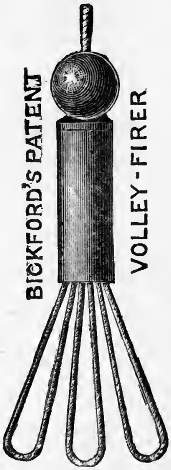

FIG. 72.

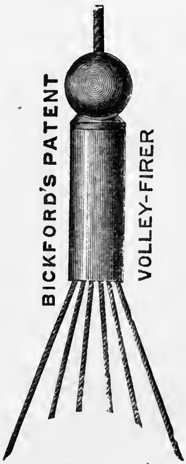

FIG. 73.

the blaster can cut it into lengths of 3 feet and 7 feet, 4 feet and 6 feet, or any proportions of 10, without detaching them from the volley-firers or affecting the simultaneousness of the explosion. Fig. 73 shows the volley-firer after the loops have been cut.

The operator should be careful to prevent any oil or greasy substance touching the instantaneous fuses. He should also be particular in fixing the instantaneous fuses into the cap or charge, so that the junction between safety fuse, disc, and instantaneous fuses is not disturbed or loosened; and he should always remember that the instantaneous fuse, which is coloured red, burns at a speed of about 120 feet per second.

The Bickford instantaneous fuse and volley-firer have been proved in actual work to be a very useful auxiliary for firing several shots simultaneously, as it is equally effective, more convenient and cheaper in first cost than the apparatus required for electric firing. On the other hand, electric fuses are cheaper than instantaneous fuses, and electric firing will therefore be found more economical for blasting on a large scale.

CHAPTER XVIII.

ELECTRIC SHOT-FIRING.

97. *Advantages of Electric Firing.*—Electric shot-firing is in some cases conducive to economy in rock blasting, and is always advantageous from the safety point of view. Its advantages are now fully established, and in connection with coal mining especially it has made rapid headway in late years.

98. *High and Low Tension Electricities for Electric Firing.*—There are, as is well known, two kinds of electricity, called high tension and low tension, and high tension batteries to fire high tension fuses or detonators, and low tension batteries to fire low tension fuses or detonators, the latter generally being termed quantity batteries.

99. *High Tension Batteries.*—High tension batteries are smaller, lighter to carry, and cheaper to make than low tension, and are therefore in more general favour in this country, especially for single shot firing. The high tension of the electricity (about 180 volts) gives it a capacity to send a spark of fire across the small space between the joints of

two insulated wires of a fuse. Some sensitive chemical composition is placed between these points, the spark fires it, and this communicates the flash to the explosive in the detonator which surrounds the fuse.

The great disadvantage with regard to these batteries, especially with those having permanent magnets, is that they are not constant, that is, that they soon lose their power and require to be sent to an electrician to be remagnetised ; and, further, the chemical composition in the fuse is liable to deterioration by time or damp. The composition also varies in its sensibility to ignition, thereby causing misfires.

100. *Low Tension Batteries.*—The low tension or quantity batteries are not constructed to give a spark, but to heat a piece of platinum wire placed across the points of the wires of the fuse by means of a current of low tension electricity (about 2 volts), in the same manner as we see the wires of an electric lamp heated, and thereby causes the ignition of the priming composition in the fuse.

The advantage of the low tension battery is that it will last for years without recharging or repairing. but it has the disadvantage of being larger, heavier, and more expensive (in first cost) than the high tension battery ; but this is of small account in an operation going on continuously, such as engineering, contractors' work, or sinking or driving headings, as the battery is not then required to be carried about as it is in coal getting.

A quantity battery requires a larger cable than a high tension one to fire many shots simultaneously at a long distance.

101. *High and Low Tension Detonators and Exploders.*—Low tension detonators have several advantages over high tension ones, because the bridge being made of platinum wire, instead of a powdered chemical composition, is more stable, more accurate, and less liable to deterioration; and furthermore (and this is the most important point) they can be tested in a few minutes by means of a galvanometer, and so proved to be good or bad. This, on the other hand, cannot be done with a high tension detonator, which, like a lucifer match, cannot be said to be good or bad till it is fired. It is evident that there is a great advantage in being able, before commencing work, to test the detonators, and throw aside any of doubtful quality. A man going to fire a hundred shots with high tension detonators cannot say how many will fire or fail; whereas a man using low tension detonators can say, with almost absolute certainty, that all the hundred will fire, if he has tested them beforehand. This testing, moreover, does not delay him, as a hundred could be tested in a few minutes, and a galvanometer for this purpose is not an expensive instrument, and the use of it can be learnt in a few minutes.

Fig. 74 shows a sectional view of Abel's low tension fuses with detonator removed, and Fig. 75 a sectional view of Abel's high tension fuses, as manufactured by Messrs. Siemens Brothers & Co., Ltd. Before firing, a detonator of suitable power is carefully inserted in the orifice in the wooden head.

For high tension electric blasting, a Siemens twistbar exploder, T class, may be highly recommended; and for low tension electric blasting, a Siemens twistbar exploder Q class.

Fig. 76 gives an inside view of Siemens' magneto-electric mine exploder, which is built up of a number of powerful permanent magnets, having between their poles a Siemens armature which is rotated by means of a

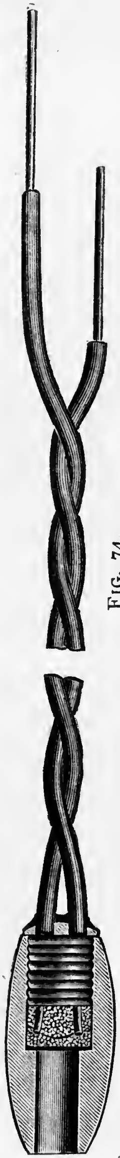

FIG. 74.

FIG. 75.

handle and toothed gearing. The working parts are enclosed in a strong dust-proof case of polished

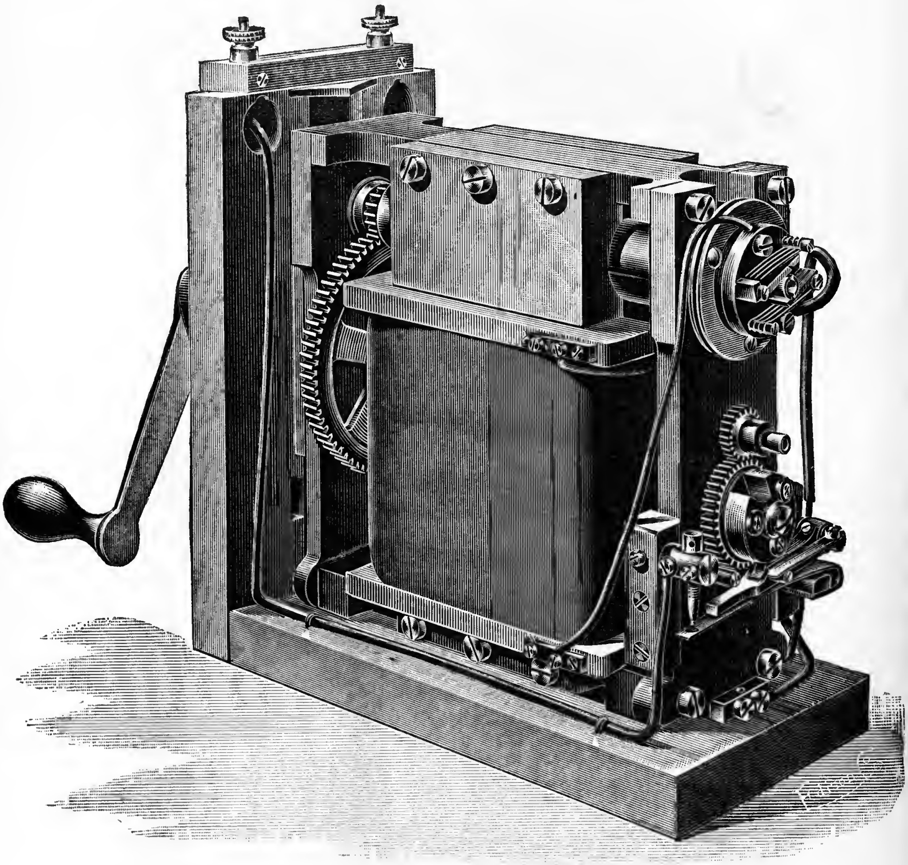

FIG. 76.

teak, which is fitted with a substantial handle for carrying, also with terminals and firing key.

The mechanism of the exploder is so arranged, that current cannot possibly pass into the line while the handle is being turned, until the press button or key is pushed in.

It is desirable, however (see directions for the use of exploders, Art. 108), to leave one leading wire at the exploder disconnected until all the connections at the shotholes are completed, and until all persons have retired to a safe place. This necessary precaution applies to all types of mine exploders.

These exploders weigh from 11 to 28 lbs., and are designed to fire from 10 to 35 high tension fuses in series.

Fig. 77 is an illustration (with cover removed) of Siemens' dynamo-electric mine exploder, in which instrument electro-magnets are substituted for the permanent magnets of the magneto machines, and have between their poles a Siemens armature rotated as described above, and carrying a commutator, by means of which the currents of alternate direction generated in the armature are caused to flow only in one direction through the coils of the electro-magnets. These latter are in circuit with the wire of the revolving armature, and during rotation the residual magnetism of the soft iron electro-magnet cores at first excites weak currents, which passing into the coils, increase the magnetisation of the core, thus inducing still stronger currents in the armature wire. This accumulation by mutual action goes on until the limit of magnetic saturation of the iron cores is reached.

The mechanism of this instrument is so arranged

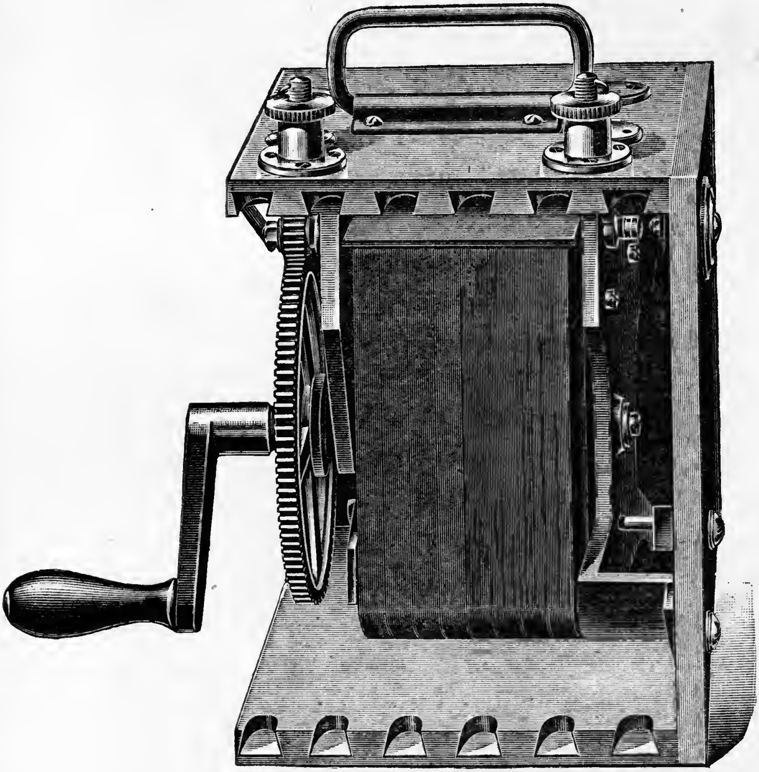

Fig. 77.

as to break the short circuit existing in the instrument after every two revolutions of the driving handle, thereby causing the current to pass out into the line, through the distant fuses and back by the return wire or earth to the instrument. The fuses being practically either an interruption of the circuit as in the tension fuse, or a great increase in its resistance by the interposition of a badly conducting substance as in the quantity fuse, the consequent action is that either a spark passes between the interrupted portions of the conductor, or the piece of bad conductor becomes highly heated, causing ignition of the explosive substance contained in the fuse.

These exploders weigh from 28 to 73 lbs., and are designed to fire 28 to 50 high tension fuses in series.

Fig. 78 illustrates Siemens' latest form of dynamo-electric mine exploder in case, with lid open and fuses in position, which they have given the name of Twist exploder.

In external appearance the machine presents a strong wooden case 14½ inches high, 8⅛ inches long and 5¾ inches wide, with leather shoulder strap for carrying purposes. The weight complete is about 26 lbs.

Inside this case, Fig. 78, is firmly fixed a series dynamo-electric machine, and on the axis of its armature, which is vertical, is fitted a pinion, which gears into a wheel mounted by a ratchet and pawl coupling on a screw spindle or "twist." The latter is fitted to revolve in two bearings, and has rapid screw-threads (similar to an Archimedean drill stock)

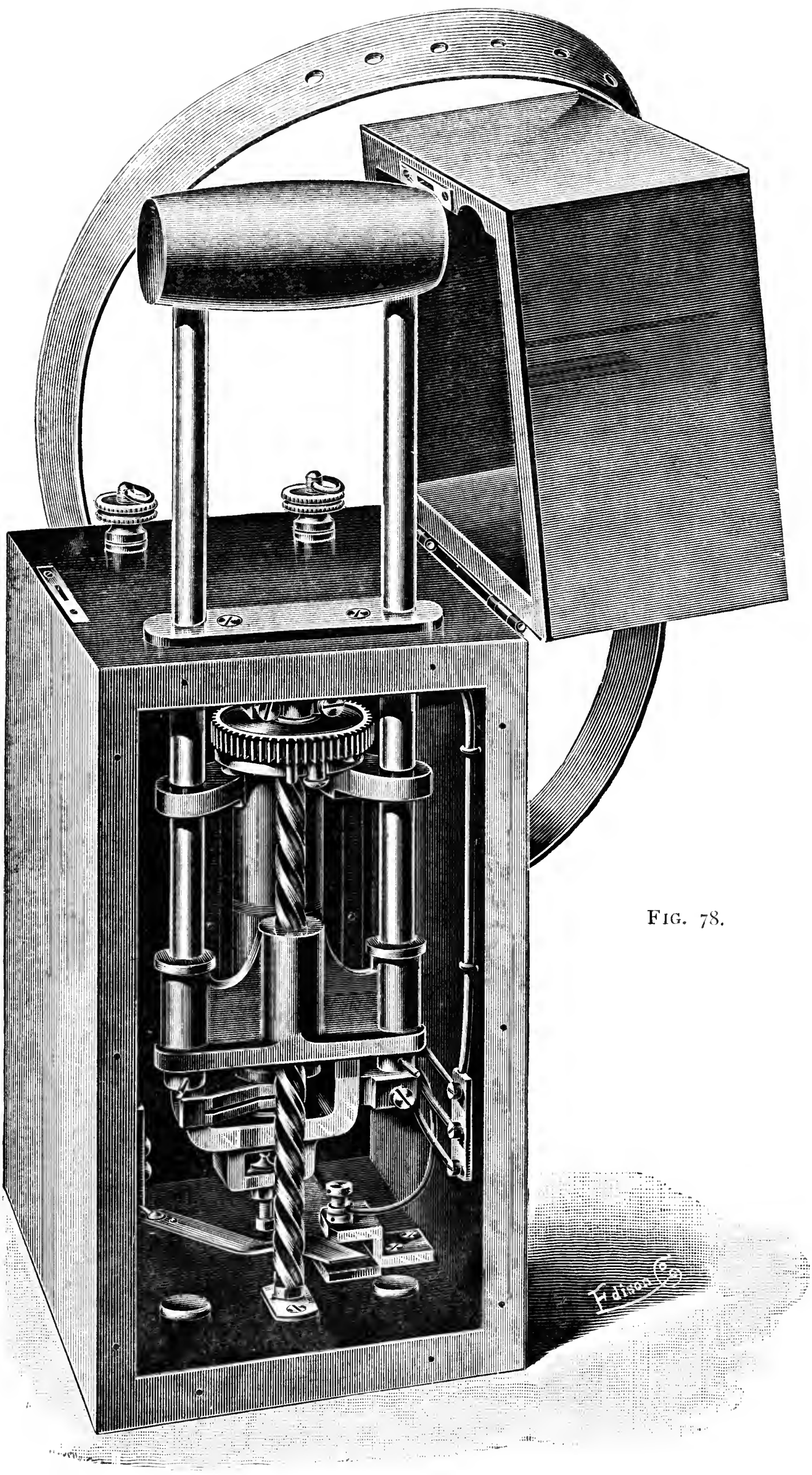

FIG. 78.

engaged by a nut, which is on a cross-head connected by two rods to a handle. These rods, which are carried through guides, project through the upper part of the machine.

The apparatus is worked as follows :—

The lid of the case, which can be secured by means of lock and key, is folded back on its hinges, exposing the handle and the terminals. In firing, the handle must first be pulled up slowly as far as it will go ; this causes the "twist" to revolve, but owing to the interposition of the ratchet and pawl, the armature of the dynamo remains stationary. The operator then pushes the handle quickly down, causing the "twist" gear and armature to revolve rapidly. The current generated passes through the coils of the magnets, through the lower contact spring and bridge at the bottom of the machine, and through the resistance coil at the side, thereby exciting the field, and causing generation of increased current until the cross-head strikes the upper spring, which is pressed down into contact with the spring beneath, at the same time breaking the contact at the bridge, thus causing the current which is generated, and which is now at its greatest strength, to pass by the terminals into the cable, and to the fuses.

The operation is best performed by placing the machine on the ground, and forcing the handle sharply down with both hands. Care must be taken that the terminals of the machine are not touched by any person during this operation.

102. *Selection of Electric Fuses and Battery or Exploder.*—Abel's detonator fuses, as manufactured by Siemens Bros., are probably the best, but are too expensive for very small blasts, in which case those manufactured on the Nobel system are to be recommended.

Siemens T exploders have been proved to fire simultaneously the following numbers of Nobel's standard and reliance fuses in parallel, namely :—

Exploder T_1, 16 fuses ; exploder T_2, 20 fuses ; exploder T_3, 25 fuses.

Up to 500 yards, the length of cable used makes no appreciable difference, and no appreciable alteration in the number of the above fuses which can be fired simultaneously in parallel.

Siemens Q exploders have been proved to fire simultaneously the following numbers of Nobel's platinum fuses, coupled in series, and fired through 100 yards cable with copper conductor, ·040 of an inch, namely :—

Exploder Q_1, 5 fuses ; exploder Q_2, 7 fuses ; exploder Q_3, 10 fuses.

With very sensitive fuses (Abel's), and a strong battery, the arrangement in series may be adopted, but when these conditions are wanting, better results will be obtained with the parallel arrangement. In the series arrangement one faulty fuse may prevent all or some of the others being fired, whereas in the parallel arrangement, each fuse is independent of the other.

The batteries used may be either magneto-electric or dynamo-electric machines. The former have been designed to meet the demand for an economical and reliable machine for industrial blasting operations, and may be most highly recommended; their price is about one-half of that of the dynamo-exploders.

103. *Choice of suitable Leading Wires and Cables.* The choice of the most suitable leading wires and cables depends on the nature of the blasting operations that have to be undertaken and the distance of the exploder from the shotholes. As a general rule, tinned copper wires insulated with gutta-percha should be employed for subaqueous work, and india-rubber covered wires for surface blasting on land and subterranean work. In cases where there is no likelihood of damage being caused to the wires leading from the exploder, unarmoured cables such as Siemens No. 142, No. 5276, No. 143, No. 5278, No. 7H, No. 7Q, No. 533 and No. 5274 can be used, these requiring two separate lengths of wire from the exploder to the shotholes; or Siemens' concentric cables No. 5277, No. 5279, No. 5369 and No. 5275, or twin cables can be adopted. These latter types have the main and return wires together in one cable, so only one length is needed for connecting the exploder with the fuse. When the operations are of

such a nature as to necessitate the use of armoured cables, Siemens' types of cables No. 6385, No. 6386, No. 6383 and No. 6384 should be used, the armouring of which can be utilised for the return half of the circuit.

104. *Precautions to be taken to ensure Insulation of Joints and Wires.*—In electrical blasting it is of the first importance that each joint in the fuse wires, and the joints between the wire of each end fuse and the leading wires or cables from the machine should be so well protected as to avoid any chance of earth contact or short circuit. The bared conducting wires, after having been scraped clean, are twisted together, and then, in the case of gutta-percha-covered wire, slightly smeared with Chatterton compound and covered helically with a narrow strip of thin gutta-percha sheet, which is pressed by means of moistened fingers around the twisted wires until they are quite insulated. As regards india-rubber-covered wires, the insulating material should be removed for about one inch from the end, and a short piece of india-rubber tube slipped over one of the cores; the conducting wires are then cleaned, twisted together, the india-rubber tube slipped over them, and tied at each end tightly over the insulation of each core. In situations where it might be troublesome to cover the temporary joints in gutta-percha wires in the way described above, the method with the india-rubber tube can be employed.

105. *Testing Low Tension Fuses with Galvanometer.*—In testing low-tension (platinum) fuses with the galvanometer, care must be exercised to put the fuse several inches into an iron pot, pipe or other suitable receptacle before connection is made with the galvanometer. When this is done the fuse wires are connected to the terminals. If the needle moves, the fuse is good, and if it remains stationary it is bad. Only one fuse should be tested at a time. High tension fuses cannot be tested in this way.

106. *Fitting of Electric Detonator Fuse to Charge.* The electric fuse is fitted to the last cartridge of a charge in the following manner. A hole is made in the cartridge by means of a small pointed wooden stick, and the detonator, attached to the fuse, pressed lightly into the hole until it is completely buried in the cartridge, the operator then tying the cartridge paper lightly over the wires with a piece of twine to prevent the withdrawal of the fuse from the explosive. The cartridge is then gently pressed into a borehole by the wooden tamping rod, and the tamping carried out in the usual manner.

107. *Connecting of Wires to Fuses for Firing in Series or Parallel.*—The method of connecting the wires for firing in series or in circuit is illustrated in Fig. 79, and Fig. 80 shows the method of attaching four high- or low-tension fuses in direct contact or in parallel. The same methods are applicable to any

larger number of fuses that may be exploded by the high- or low-tension battery employed.

For single shot firing the one wire of the fuse is first connected to one end of one of the two conducting wires of the cable and the other wire of the fuse

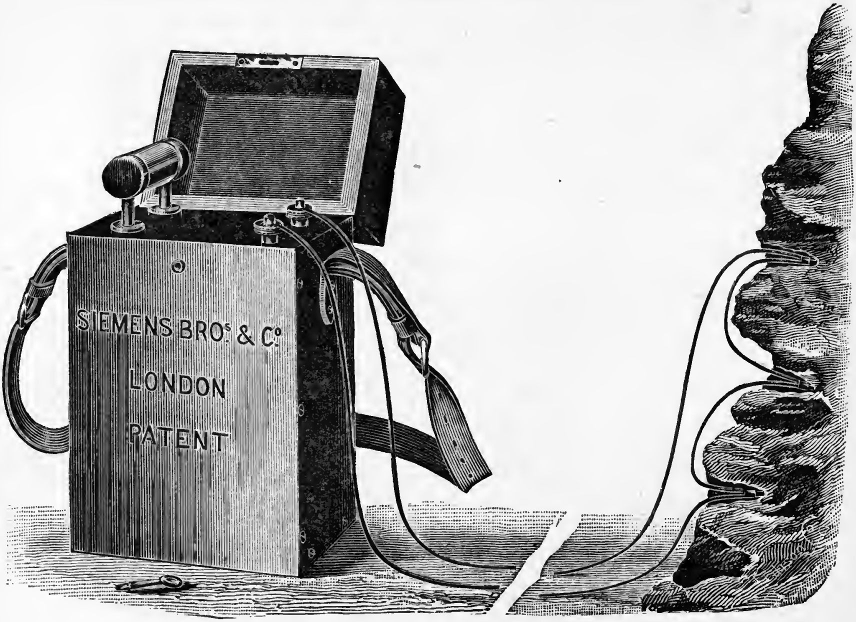

FIG. 79.

to one end of the other conducting wire, and then the other ends of the conducting wires to the two terminals of the exploder.

For firing several shots in series or in circuit, Fig. 79, the fuse wires are connected so that one of the wires of the first shot, taken in the order of the series in circuit, is connected to one of the wires of the second shot, and the other wire of the second shot to one of the wires of the third shot, and so on in succession until the whole series is so connected. The second wire of the first shot is then joined to one end of one of the conducting wires of the cable, and the second wire of the last shot to one end of the other conducting wire. Finally the other ends of the conducting wires are connected to the exploder to complete for firing.

For firing in direct contact or in parallel, Fig. 80,

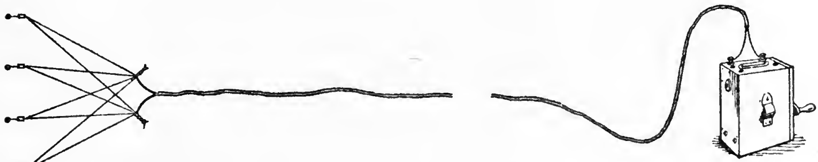

FIG. 80.

one wire of each fuse is connected direct to one end of one of the conducting wires, and the other wire of each fuse to one end of the other conducting wire, whereupon the other ends of the conducting wires are connected to the exploder as before.

108. *Directions for Use of Exploders.*—The ends of the leading wires for attachment to the exploder must be scraped until a clean metallic surface is obtained, then connected to the two terminals of the exploder and tightly clamped by means of the milled heads. It is advisable to leave one leading wire at

the exploder disconnected until all the connections at the shotholes are completed. The leading wires or cables must be brought to a place where the operator can stand in perfect safety from the blast.

To fire with Siemens' magneto-exploders, when the final connection has been established, it is firmly held down whilst the handle is turned several times, and as soon as a good speed has been obtained the press button at the side of the case is suddenly pushed in while the handle of the machine is still being turned at the highest speed, so that the maximum current of the machine is sent into the line. The handle must not be turned slowly, as this would produce a weak and uncertain current, and the press button must not be touched till the full power of the machine is developed, that is, when the handle is being turned at the highest attainable speed.

In using Siemens' dynamo-exploders, before completing at the machine the final connection of the current, the handle of the exploder must be slowly turned until a click is heard. The handle is then left at rest, the two leading wires are clamped to the terminals of the exploder, then with two or three rapid turns of the handle the current is developed, and sent automatically into the circuit of the leading wires and fuses. If the dynamo quantity exploders are fitted with a press button or firing key, as in the case of the magneto-exploders, in place of the automatic firing arrangement the handle of the machine is rapidly turned, and when a good speed is got up, the key is pressed down to send the current into the circuit.

The working of Siemens' twist exploder is explained in Art. 101.

As regards the use of the rackbar exploder, the connections of the cable to the two terminals of the exploder should not be made until everything is ready for firing, and all persons have retired to safety. In firing, the handle must be pulled up as far as it will come, and then forced down as quickly as possible. These operations are best performed when both hands are used together. The hand must not touch the terminals of the exploder during shot-firing.

Immediately after completing the firing the operator should first disconnect the cable from the exploder, then disengage it from any chance debris which may have fallen upon it, and coil it up. This instruction should be followed even when by defective manipulation a misfire has occurred.

109. *Points to be attended to in Electric Blasting.* It may be well to point out that electrical blasting has not been as successful as it should have been, because it has been carried out by men with no knowledge of electricity, and without any opportunity of learning the elementary principles. A missed shot was immediately attributed to bad electric fuses, whereas it was often caused by an unsuit-

able or insufficiently charged battery, an inferior leaky cable, wrong handling, or dirty, verdigrised implements.

To ensure success the following points must be attended to :—

1. That the battery, wire and detonators are suitable to each other.

2. That the battery is of sufficient power.

3. That the electric fuses, especially high tension ones, are stored in a dry place, and that everything is kept as dry and clean as possible.

4. That all the joints are bright at the point of contact of the wires, and well made, and that the joints do not touch each other.

5. That the wires do not kink or twist so as to cut the insulation during the process of tamping.

N.B.—If the insulation is cut the fuse is useless for wet ground or a wet hole, and should be set aside.

6. That the operators' hands do not touch the terminals of the battery when firing.

7. That the battery is not connected to the cable until every man is in safety.

In cases where only short lengths of conducting wires are employed, and these are frequently shifted or subjected to such rough usage as would endanger the insulation of the cable, the use of electricity of small electromotive force is advisable, for this will pass without loss through a cable with faults in the insulation, through which currents of greater force would readily escape. In such cases, therefore, the quantity exploder should be selected. On the other hand, where the conducting wires remain for the greater part of their length undisturbed, high tension electricity is preferred, because its greater power of overcoming resistance enables it to fire a larger number of fuses through longer distances. On this account the tension exploder is better suited for the firing of a large number of fuses simultaneously through long circuits.

CHAPTER XIX.

ON EXPLOSIVES AND THEIR SELECTION FOR ROCK BLASTING.

110. *A Good Explosive* must be reasonably safe to handle, transport and store under ordinary conditions, and allow of such amount of rough usage as it may fairly be expected to meet with, and should not give off actively poisonous gases or vapours, nor deleterious ones, either before or after explosion.

111. *List of Explosives for Rock Blasting.*—The following explosives, which are divided into five classes, fulfil the various requirements of strength, safety, handiness and economy for rock blasting, and, excepting rack-a-rock, were authorised for manufacture in, or importation into the United Kingdom on January 1st, 1897.

Class 1. *Gunpowder.*

Ordinary gunpowder in grains or pellets.
Compressed gunpowder.

Class 2. *Nitrate Mixture.*

Chilworth special powder.
Dahmenite A.
Electronite No. 2.

Fortiss explosive No. 1.
Safety blasting powder.
Westfalite.

Class 3. *Nitro-Compounds.*

Amberite No. 1.
,, No. 2.
Ammonite.
Ardeer powder.
Ballistite.
Bellite.
Blasting amberite.
Blasting gelatine No. 1.
,, ,, No. 2.
Carbo-dynamite.
Carbonite.
Collodion cotton.
Di-flamyr.
Dynamite No. 1.
,, No. 2.
Electronite No. 1.
Faversham powder.
Guncotton.
Gelatine-dynamite No. 1.
,, ,, No. 2.
Kynite.
Lithofracteur.
Matagnite gelatine.
Nitrated guncotton.
Oarite.

Potentite.
Primers for gelatines.
Roburite No. 1.
„ No. 2.
„ No. 3.
Stonite.
Rosslyn blastite.
Schultze blasting powder.
Tonite, or cotton-powder, No. 1.
„ „ No. 2.
„ „ No. 3.

Class 4. *Chlorate Mixture.*

In the United Kingdom there are no authorised explosives of this class.

Rack-a-rock is an explosive of this class, of which 240,399 lbs. were used in conjunction with 42,331 lbs. of dynamite for the removal of the Hell-Gate Rocks in America in 1885.

Class 5.

Pure fulminate of mercury, either alone or mixed with chlorate of potash, for detonators.

112. *High and Low Explosives.*—For the purpose of rock blasting, explosives have been divided, as before explained, into two classes, viz. "high" and "low." Class 1 is invariably low, but the others are more or less high, according to their composition.

From their action being comparatively slow, the low explosives are most suitable for quarrying rock and coal in large blocks, and the high explosives when it is desired to shatter the rock and excavate it in the most economical and expeditious manner. The former are therefore generally adopted for quarrying rock for building purposes, and the latter for tunnelling, the driving of headings and levels, and shaft sinking, when rock has to be excavated.

113. *Influence of the Strength and Density of an Explosive on the Cost of Boring Holes.*—For blasting hard rock, the economical value of an explosive depends chiefly on its strength and density, as the cost of the boreholes or chambers which have to be bored will depend on these qualities. For instance, if the density of ordinary gunpowder is 1·00, and of dynamite 1·6, and the latter is two and a half times as strong as the former weight for weight, the relative size of chamber required for the dynamite as compared with the powder will be $\frac{1}{1 \cdot 6 \times 2 \cdot 5} = \frac{1}{4}$th. Consequently, fewer holes will have to be bored for dynamite than for ordinary gunpowder. The cost of boring the holes is generally the most important consideration in blasting hard rock, as it is the chief item of the total cost, varying approximately in inverse ratio to the comparative strength by volume of the explosive used.

114. *A very valuable quality for an explosive*

is that it should be plastic. A borehole cannot be charged with any rigid explosive without bearing air interstices, which cause a loss of blasting power.

115. *Methods of Reducing the Shattering Effect of the High Explosives.*—The shattering effects of high explosives on rock may, when desirable, be greatly reduced by either of the following methods.

a. By adopting such form of chamber as will contain a small charge, and yet give a comparatively large area of pressure therein to the gases from the explosion.

b. By distributing the charge in several shot-holes of small diameter spaced equally apart in a row, and fired simultaneously, instead of one shot-hole of larger diameter to do all the work, so that the shock of the explosion is distributed in the rock.

c. By filling the shotholes in (*c*) with water, and firing a small quantity of the high explosive on the top of each column of water simultaneously, so that thereby only just sufficient hydraulic pressure is set up in the holes to crack the rock from its bed.

The high explosives may, therefore, be advantageously adopted for blasting rock for building material, as well as for other purposes.

116. *Advantages of Gunpowder.*—When the cohesion of a mass of rock is small, as when it is cut up into blocks by joints or natural divisions, and the rock is to be used for building material, and boreholes of small diameter will contain sufficient

gunpowder or low explosive to crack and eject the rock from its bed, there is manifestly a great advantage in using this explosive, as it is cheaper and has less tendency to shatter the rock.

117. *Relation between the Maximum Lines of Resistance which may be Blasted in Homogeneous Rock with Shotholes of one diameter, charged with different Explosives, and the Maximum Pressures developed by such Explosives.*—If the maximum line of resistance be W, which a charge of one explosive, when applied in a borehole of a given diameter, will overcome in homogeneous rock, then it is evident that we can put for the resistance,

$$R = S\,W\,K_1;$$

and, on the contrary, for a charge of another explosive in the same rock, if the line of resistance be W_1

$$R_1 = S\,W_1\,K_1.$$

The ratio of these resistances is

$$\frac{R_1}{R} = \frac{S\,W_1\,K_1}{S\,W\,K_1} = \frac{W_1}{W}.$$

For the shearing force of the one charge we have

$$P = M\,A,$$

and for the other

$$P_1 = M_1\,A.$$

M and M_1 being the maximum pressures per unit of surface that can be developed by the different explosives in the chambers in which they are applied.

Hence, the ratio of these forces is

$$\frac{P_1}{P} = \frac{M_1 A}{M A} = \frac{M_1}{M}.$$

To obtain the maximum effect from a blast, the force must be just equal to the resistance, or we must have $P = R$ and $P_1 = R_1$.

Therefore

$$\frac{P_1}{P} = \frac{R_1}{R};$$

and as $\frac{P_1}{P} = \frac{M_1}{M}$ and $\frac{R_1}{R} = \frac{W_1}{W}$, we have

$$\frac{M_1}{M} = \frac{W_1}{W}.$$

Hence, the maximum pressures developed by explosions in homogeneous rock of the same cohesive strength are proportional to the maximum lines of resistance that can be sheared by similar volumes of charge exploded in boreholes of one diameter.

The above relation may be used with great advantage for the testing of the comparative strength of explosives for blasting.

118. *Explosives most generally employed for Rock and Coal Blasting.*—The following explosives, which may be divided into high explosives, low explosives, and safety explosives, are those most generally employed for rock and coal blasting and are, therefore, selected for description.

High Explosives.

Dynamite.
Gelignite.
Gelatine-dynamite.
Blasting gelatine.
Tonite or cotton-powder.
Blasting amberite.
Electronite.

Low Explosives.

Ordinary gunpowder.
Compressed gunpowder.

Safety Explosives.

Ammonite.
Ardeer powder.
Bellite No. 1 and No. 3.
Carbonite.
Dahmenite A.
Electronite No. 2.
Roburite No. 3.
Westfalite.

119. *Dynamite.*—Dynamite is an admixture of nitro-glycerine with a porous infusorial earth called Kieselguhr, which consists mainly of silica.

Dynamite No. 1 consists of about 75 parts by weight of nitro-glycerine and 25 parts by weight of Kieselguhr, and some other substances which are sufficiently absorbent to prevent exudation of the nitroglycerine.

Dynamite No. 2 is milder and slower in its action than dynamite No. 1, and was introduced to compete with gunpowder where the great power and local shattering effect of No. 1 dynamite was undesirable, for instance in slate and granite quarries, but it is now little used in this country. It consists, according to Major Cundell's 'Dictionary of Explosives,' of not more than 18 parts by weight of nitro-glycerine mixed with 82 parts by weight of a preparation composed of nitrate of potassium 71 parts, charcoal not less than 10 parts, ozokerit 1 part.

Dynamite is a plastic mass varying in colour from buff to reddish brown. The direct contact of water disintegrates it with a separation of the liquid nitroglycerine, and when used in wet places it should be protected from contact with water. When ignited in small quantities it simply burns away fiercely, but fatal accidents have arisen by warming it upon a shovel, in an oven, and in various other ways. If it be warmed up to a temperature approaching the exploding point (about 350° F.), it becomes exceedingly sensitive to shock or blow, and once that point is reached, it does not simply ignite, but explodes with great violence. It is exploded by detonators, and it is of the highest importance that complete detonation should be effected, not only to obtain the full effect of the explosive, but to avoid the formation of noxious gases. In cold weather it congeals, and must be thawed before use.

It is supplied in cartridges of any size, and has a specific gravity of 1·6.

120. *Detonators.*—Detonators are metallic capsules, usually of copper, resembling very long percussive caps, charged with a small quantity of pure fulminate of mercury, or a mixture of the same with chlorate of potash, and occasionally other substances. They are usually made in eight sizes, and according to their strength are numbered 1 to 8, No. 1 being the weakest and No. 8 the strongest. No. 3 detonators should be used for ordinary dynamite, and Nos. 6 and 7 for blasting gelatine, gelatine-dynamite, gelignite, tonite, blasting amberite and electronite.

121. *Gelignite.*—Gelignite is an explosive compound consisting of nitro-glycerine and nitro-cellulose with a certain proportion of nitrate of potash and wood-meal, or, in other words, it is a form of gelatine-dynamite, and is the latest form of the gelatinous explosives invented by Mr. Alfred Nobel. Being a plastic compound, it will on being tamped fit accurately and easily into any borehole, thus entirely preventing the formation of an air chamber, which in the case of gunpowder and other rigid compounds may reduce the explosive force by fully 10 per cent. It is practically unaffected by immersion in water, and when detonated by Nobel's gelatine detonators,

which are required to develop its full energy, no noxious fumes are produced by its explosion; and being more insensible to shock than dynamite, it is therefore a safer and even more reliable explosive; moreover, its action is slower and it has a much less local shattering effect. Like all nitro-glycerine explosives it is liable to become congealed in cold weather, and somewhat insensitive even at a temperature as high as 45° F., and is then unsuitable for charging boreholes, but it can be safely and readily thawed for use by putting the cartridges into a watertight vessel, and then placing such vessel in warm water. The specific gravity of gelignite is 1·55.

Gelignite is supplied in cartridges of any size, and may be highly recommended for pitsinking, quarrying and tunnelling, metalliferous mining, limestone blasting, blasting in damp workings and for submarine blasting and harbour or dock works.

The advantages of gelignite over dynamite are:—

1. It is relatively cheaper.

2. Its explosive energy is greater by fully 12 per cent.

3. It is practically unaffected by damp or submersion in water.

4. It is entirely free from noxious fumes when properly exploded by means of gelatine detonators.

5. It is more easily manipulated because of its greater plasticity.

6. It is less shattering, and in quarrying, therefore, it brings down the rock in much larger blocks.

7. It is more economical.

122. *Gelatine-Dynamite.* — Gelatine-dynamite is a compound of nitro-cellulose with a certain proportion of nitrate of potash, and is much more powerful than No. 1 dynamite. Miners who have tested it in comparison with dynamite affirm that it yields results fully 25 per cent. better than those of dynamite. In appearance it resembles a thick jelly of a brownish colour, and it is a plastic compound less tough than No. 1 blasting gelatine, but otherwise similar to that very strong explosive. This plasticity of the gelatinous compounds renders them very convenient for charging boreholes, as their cartridges will much more readily adapt themselves to the size of the holes. In common with all nitro-glycerine compounds it congeals in cold weather, when it is unsuitable for charging boreholes, but it readily assumes its original consistency when heated in the hot water warming pan.

Gelatine-dynamite being more insensible to shocks than dynamite, requires in order that the full strength of the compound may be developed, to be exploded by a stronger detonator than is used for dynamite. The strong sextuple detonators manufactured by Nobel's Explosive Company, Limited, should invariably be used for this explosive. The employment of weak detonators, resulting in the consequent imperfect explosion of gelatine-dynamite, has led to numerous complaints caused by the explosive giving off disagreeable fumes, from which, with perfect detonation, it is entirely free. Should sextuple detonators not be at hand, dynamite primers and ordinary detonators may be substituted, but the necessity for following so inconvenient a method should be avoided.

Gelatine-dynamite possesses in a high degree the character of a safety blasting agent, and on this account, and in view of its water-resisting properties, it has been specially recommended by the Royal Commission on Accidents in Mines in their report, presented to both Houses of Parliament in 1891, as the most suitable explosive for safety blasting in coal mines. It is unaffected by direct submersion in water, and this valuable property renders it specially suitable for blasting in damp workings, and for submarine blasting.

The following are the chief advantages possessed by gelatine-dynamite.

1. It is very much more powerful than dynamite.

2. It is relatively cheaper than dynamite.

3. It is unaffected by submersion in water.

4. It is, when properly exploded with gelatine detonators, entirely free from noxious fumes.

5. It is a more convenient, more easily handled, and more economical than dynamite.

The specific gravity of gelatine-dynamite is 1·55, and it is supplied in cartridges of any size.

123. *Blasting Gelatine.*—This powerful explosive is a compound of nitro-glycerine, of which it contains 93 per cent., together with a special quality of nitro-cotton. Its disruptive force is enormous, being not less than 50 per cent. greater than that of No. 1 dynamite. In appearance it somewhat resembles a thick jelly of a brownish colour; but some of the qualities have various ingredients incorporated with the jelly, with the object of modifying the force of the explosion. In its normal state it is a tough plastic mass having a specific gravity of 1·55. Its plasticity makes it very convenient for charging bore-holes, as by squeezing the cartridges with a wooden rod, they can be made to fill the boreholes completely. In cold weather it hardens, and loses its jelly-like character. In this state it may be used for open blasting, but it is not recommended to use it frozen for charging boreholes, as its hardness renders it incapable of accommodating itself to the inequalities of the hole, and attempts to force it might result in accident. Frozen blasting gelatine may be softened for use, with perfect safety, in the hot water warming pans employed for thawing dynamite. The effect produced by the explosion of frozen blasting gelatine is more violent than that produced by the explosion of the unfrozen material, because the soft elastic blasting gelatine yields to

the shock of the explosion, while the frozen material does not.

Blasting gelatine is more insensible to shocks than dynamite, and hence it is necessary to employ for its explosion a stronger detonator than is used with the latter compound. With the view of meeting this requirement, gelatine detonators of a special strength are supplied by the manufacturers, and these should invariably be employed in order that the full power of the blasting gelatine may be developed. If from any cause the gelatine detonators cannot be obtained, a dynamite primer instead of a blasting gelatine one may be used, and the charge can then be readily exploded by means of an ordinary detonator.

Blasting gelatine is adapted for blasting in very hard rock, but it is at the same time better suited than dynamite for blasting in mild or soft rock, where a shattering effect is not required, because, although the disruptive force of the former is much greater than that of the latter, the transmission of the explosion throughout its mass is less rapid. The enormous power of blasting gelatine being developed more slowly, the shattering effect of its explosions is therefore less severe than in the case of dynamite.

Blasting gelatine is not damaged by immersion in water, and it is therefore specially suitable for submarine mining. When dynamite is immersed in water, the nitro-glycerine begins to exude, and on that account, in using dynamite for subaqueous blasting, special means must be employed to prevent the escape of the nitro-glycerine. Water, on the other hand, has no action on blasting gelatine, and it may be exploded effectively, after lying for months under water. It is supplied in cartridges of any size.

The following distinguished authorities on explosive compounds have expressed unqualified approval of the No. 1 blasting gelatine manufactured by Nobel's Explosives Co., Limited:—

Sir Frederick A. Abel, C.B., F.R.S., Chief Chemist of the British War Department, in an address on Explosive Agents, delivered in St. Andrew's Hall, Glasgow, on 1st March, 1883, stated: "It is in every respect the most perfect explosive known."

Brigadier-General Henry L. Abbot, Corps of Engineers, United States Army, after conducting a series of experiments, extending over several years, with explosives of every description, manufactured both in America and Europe, concludes his official report to the American Government thus: "These experiments show that this explosive is the most powerful ever tested here, and that it is most admirably suited to submarine mining."

When set fire to by a fuse, or by other means, or when insufficiently detonated, blasting gelatine burns rapidly without explosion, and gives off disagreeable fumes. This shows the absolute necessity for using the powerful gelatine detonators to ensure complete explosion.

The special advantages No. 1 blasting gelatine possesses over dynamite may be summarised as follows :—

1. It is 50 per cent. more powerful than dynamite, and is the strongest known explosive.

2. It is relatively cheaper than dynamite.

3. It is perfectly effective in water without using the special precautions requisite in the case of dynamite.

4. It is, when properly exploded, entirely free from noxious fumes.

5. It is less shattering in effect than No. 1 dynamite.

124. *Tonite.*—Tonite, like dynamite, is one of the "high" explosives, but differs from the majority of this class, inasmuch as it does not contain any glycerine in its composition. It may be described as a nitrated guncotton, the nitrate usually employed being that of barium. For use it is compressed into cartridges of various sizes and weights, which are covered with brown paper steeped in paraffin wax to render them impervious to water.

Tonite can only be fired by a specially prepared detonator of a very strong character, ordinary detonators being much too weak to effect this result in the majority of instances. It does not freeze, and is consequently without this objectionable feature of any explosive. It is strongly recommended for its safety in transit, storage and manipulation ; may be used in any climate, and gives off very little smoke when fired. One of the greatest advantages of this powder is that the holes need not be so large nor so deep as those required by ordinary gunpowder ; thereby saving a great deal of labour and expediting the work. Another advantage is that it is quite free from poisonous matter. It can be used in places where gunpowder would utterly fail, such as in soft beds, between two layers of rocks, or inserted in fissures without any boring whatever. The strength is about four times that of blasting powder, or equal to No. 1 dynamite. The charges may be taken to have a density of about 1·5. The cartridges should invariably be stored in a dry place until they are required to be used, as the quality of this explosive is injured by moisture. Where the cartridges have to remain in water for more than five minutes, additional protection is necessary in the shape of tin canisters, indiarubber bags, or waterproof packing. Special appliances can be obtained of the manufacturers for this class of work. The following sizes of cartridges are always kept in stock, viz. : 1, $1\frac{1}{8}$, $1\frac{1}{4}$, $1\frac{1}{2}$, $1\frac{3}{4}$ and 2 inch.

125. *Blasting Amberite.*—This is a new and powerful explosive, consisting of thoroughly purified nitro-cotton, wood-meal and other ingredients, manu-

factured by Messrs. Curtis & Harvey, Clyde Mills, who claim for it the following advantages :—

1. It contains no nitro-glycerine.

2. It will not become frozen in any weather.

3. It is in no way affected by changes of temperature.

4. It contains no poisonous ingredient.

5. It is not affected by exposure to a damp atmosphere.

6. It does not deteriorate in keeping.

7. It emits no noxious fumes on explosion.

8. It is not liable to explode from friction or blows.

9. It may be used in wet boreholes, as the cartridges are waterproofed.

10. It requires no more stemming than the other higher explosives.

11. It is made up into cartridges of various diameters and convenient weights.

12. The cartridges may without risk be cut in two to give any desired length of charge.

The strength of the explosive is about equal to dynamite, and it may therefore be used to replace the latter in all kinds of work.

Cartridges are made of $\frac{7}{8}$, 1, $1\frac{1}{8}$, $1\frac{1}{4}$, $1\frac{7}{16}$ and $1\frac{3}{4}$ inches diameter, being issued in two forms of cartridges—one "plain" (viz. closed at both ends), and the other "primers," which are left open at one end to receive a detonator. It is important to have the cartridges of such size that they nearly fit the borehole, or there will be great loss of force; but they must not be too large to go to the bottom of the hole. The detonator used must not be less than No. 6.

The specific gravity of amberite is 1·102.

126. *Electronite.*—Electronite is a high explosive consisting of blasting amberite mixed or impregnated with carbonate of calcium, designed to afford safety in coal-getting. This important end has been obtained by using such ingredients, and so proportioning them, as will ensure, on detonation, a degree of heat insufficient under the conditions of a "blown-out" shot to ignite firedamp or coal-dust. In dry and fiery mines no explosive can probably afford greater security. But to be safe is not the only qualification required in a blasting agent for mining operations. There are others of more or less importance, according to the conditions under which the explosive is worked. Among these the manufacturers claim the following are possessed by electronite:—

1. It is smokeless.

2. It emits no noxious fumes on explosion.

3. It contains no poisonous ingredient.

4. It cannot become frozen in the coldest weather.

5. It is equal in strength to the nitro-glycerine compounds.

6. It does not shatter coal, as many of the high explosives do.

7. It cannot be exploded by friction or flame, and is consequently perfectly safe to carry, store and handle.

8. The cartridges are waterproofed.

Electronite is suitable for all kinds of blasting operations. In hard rock it is as effective as dynamite, and it gives very satisfactory results where a rending rather than a shattering action is required.

Electronite cartridges are supplied in three lengths, namely, $4\frac{1}{2}$, 6 and $7\frac{1}{2}$ inches; and in the following diameters: 1, $1\frac{1}{8}$, $1\frac{1}{4}$, $1\frac{7}{16}$ and $1\frac{1}{2}$ inches.

It is important to use a sufficiently strong detonator. Nothing less than No. 6 is strong enough. Better results will be given by a No. 7. Weak detonators will not develop the full power of the explosive.

The specific gravity of this explosive is 0·806.

127. *Ordinary Gunpowder.*—Gunpowder is a mechanical mixture of sulphur, nitre and charcoal, in the proportions of 62 to 75 parts of nitre, 9 to 20 parts of sulphur, and 9 to 18 parts of charcoal; the ordinary composition being 75 parts of nitre, 15 of charcoal and 10 of sulphur. Blasting powder is a cheaper and inferior variety containing less nitre and more sulphur and charcoal. The

properties of gunpowder depend largely on its physical characteristics, namely, the thoroughness of the mixture of the ingredients; its gravimetric density; size and shape and glazing of the grains or pellets; and the amount of moisture it contains. An inferior and cheap powder is in some cases advantageous for quarrying (as when the chief object is to shatter the rock as little as possible), as, owing to its combustion being slower, its action in the rock is not so violent as the more perfect explosive. In all other cases the strongest and best powder should be used. In wet ground gunpowder must be used in watertight cartridges. Gunpowder is more economical than the high explosives for "bulling," which consists in filling the main crack or fissure in a mass of rock cracked from its bed by a previous blast with an explosive, and firing it to further loosen the rock. Gunpowder is generally preferred for quarrying rock in large blocks, and also for blasting coal.

The density of coarse gunpowder in bulk is about 0·7.

Gunpowder may be fired by ignition or detonation.

128. *Compressed Gunpowder in Pellet-Blasting Cartridges.*— These are cartridges of gunpowder moulded into solid cylinders and perforated to admit of their being strung on the safety fuse by which

they are to be ignited. In charging the shotholes these cylinders are in practice found to be very convenient, a number sufficient for the charge having only to be strung on to the fuse, the lower end of which, cut slanting, should be doubled back, which will retain it in its position, and the charge then inserted in the hole. The charging in this way may be effected as readily in ascending as in descending holes. Other advantages are that the fuse cannot be pulled out of the charge in tamping, and that but little smoke is produced.

Compressed pellets are not only convenient to handle and safe to use, but also very effective in their action. The force of explosion being concentrated upon a small surface, the rending effect is great, and more work is done by bringing out the rock to the bottom of the hole; moreover, these pellets being slow to absorb moisture, they retain their strength in damp holes. The central perforation allows the hot gases to pass down both on the inside and the outside, as well as through the interstices between the pellets, thereby igniting the latter simultaneously at every point of their surfaces, a condition favourable to the quick and complete combustion of the powder.

The remarkably gentle lifting action of compressed powder renders it very suitable for use in slate and other quarries where it is required to move masses of rock without shattering them.

This powder is made in two qualities, viz. :—

Compressed gunpowder pellet cartridges, 1, $1\frac{3}{16}$, $1\frac{1}{4}$, $1\frac{5}{16}$, $1\frac{3}{8}$, $1\frac{1}{2}$ and $1\frac{3}{4}$ inches in diameter, and

E.S.M. compressed gunpowder (extra strong) pellet blasting cartridges, 1, $1\frac{3}{16}$, $1\frac{1}{4}$, $1\frac{3}{8}$, $1\frac{1}{2}$ and $1\frac{3}{4}$ inches in diameter.

The specific gravity of the latter is 1·646.

129. *Safety Explosives,*—In order to prevent explosions of mixtures of coal-gas and air, and coal-dust and air, in gaseous mines, due to the effect of blasting with the ordinary explosives, so-called flameless or safety explosives are employed. Hitherto gunpowder has been most extensively used in coal-mining, as owing to its combustion being gradual, or comparatively slow, its action is rending or projecting, and not shattering, and therefore enables coal to be blasted with a minimum of slack ; but this sole advantage has been far outweighed by certain results which have proved fatal to life on account of the heated products of combustion produced setting fire to mixtures of coal-gas and air and coal-dust and air. Owing to the improved methods of ventilation, the danger due to accumulation of gas has been very much diminished, but on the other hand the air has been charged to a greater extent with fine particles of coal-dust, and it is now known that such a mixture is easily ignited by a blown-out shot of powder, and is capable of initiating a most disastrous explosion. The use of gunpowder has

therefore been prohibited in certain coal mines by order of the Secretary of State for the Home Department, under Section 6 of the Coal Mines Regulation Act, 1896, the permitted explosives being the following:—

Ammonite.
Ardeer powder.
Bellite No. 1 and No. 3.
Carbonite.
Dahmenite A.
Electronite No. 2.
Roburite No. 3.
Westfalite.

These explosives may be divided into two classes, firstly, the nitro-glycerine class comprising Ardeer powder and carbonite; and secondly, the nitrate of ammonia, class ammonite, bellite Nos. 1 and 3, dahmenite A, electronite No. 2, roburite and westfalite.

Ardeer Powder consists of 31 to 34 parts nitro-glycerine, 11 to 13 parts kieselguhr, 49 to 51 parts magnesic sulphate, and a little nitre.

The high temperature produced by the explosion of the nitro-glycerine is lowered by the presence of magnesic sulphate, which salt contains a large amount of water of crystallisation, the evaporation of which, when the explosion takes place, absorbs a considerable amount of heat. The proportion of magnesic sulphate present exerts, therefore, a most decisive influence upon the degree of safety obtainable by the use of this explosive.

This explosive is intended to afford to colliery owners and miners a cheap and reliable means of coal blasting. In addition to being free from any complication of appliances, its safety qualities, especially the low temperature of the resulting products, and the absence of any flame that can ignite fire-damp or coal-dust, appear to give it an advantage over other explosive compounds such as black powder or tonite, and possibly explosives of the nitro-benzol or ammonia class, such as roburite, securite, bellite, &c., which are liable to change their character from atmospheric influences.

When immersed in water this explosive possesses the advantage of being practically unchanged and non-exuding. In practice it is found to be more than 50 per cent. stronger than blasting powder.

As Ardeer powder is intended for coal-getting purposes only, care has been taken that the disruptive power should be so modified as to secure the essential advantage of bringing down coal in a round, lumpy and marketable condition.

It is usually supplied by the manufacturers in cartridges $5\frac{1}{4}$ inches long by $1\frac{1}{8}$ inch in diameter, but may be obtained when required in other sizes. Electricity should be used for firing with Nobel's No. 3 electric detonator fuses.

The specific gravity of Ardeer powder is 1·16.

It is important to note that Ardeer powder should never be used in a hard or frozen condition.

Carbonite consists of under 27 parts nitro-glycerine and not less than 73 parts of a mixture of wood-meal not less than 40 parts, with not more than 36 parts of the nitrates of potash, soda or baryta. The average composition is nitro-glycerine 25 parts, wood-meal 40 parts, and 35 parts of nitre (with some nitrate at times).

It is a good substitute for blasting powder in collieries, &c. H.M. Inspectors of Mines strongly condemn the use of blasting powder in coal mining as highly dangerous, and they recommend that a high explosive practically free from flame, which property is claimed for carbonite, should be substituted. It differs from the high explosives in being slow-rending, and in this respect closely resembles blasting powder, but is from 2 to 2½ times stronger than blasting powder. The manufacturers supply it in cartridges of any size and weight, but the usual sizes which are always kept in stock are 1¼ inch weighing 2 ounces, and 1½ inch weighing 4 ounces.

Nobel's No. 6 Detonators are required to effect its complete detonation, and in collieries the detonators should be always fired by electricity.

The gravimetric density of carbonite is 1·12.

When hard or frozen it should be thawed in special warming-pans, as in the case of dynamite and all other nitro-glycerine compounds.

The great danger of incomplete detonation of explosives cannot well be exaggerated, especially in dusty or fiery collieries, where there is always the great risk of unexploded cartridges, or portions of cartridges left unexploded, continuing to flame after the detonator has exploded. It only requires the presence of inflammable gas or coal-dust in order to have all the conditions necessary for a serious accident. See 'Reports of Flameless Explosives, Committee of North of England Institute of Mining and Mechanical Engineers.'

Settle's patent gelatine-water cartridge is considered to be the best and safest system of blasting for collieries, and may be obtained from Nobel's Explosives Co., Glasgow, or their local agents.

The other safety explosives, according to the official pronouncements of Her Majesty's Chief Inspector of Explosives, are defined as consisting of the following components :—

Ammonite.—87 to 89 parts ammonic nitrate and 11 to 13 parts di-nitro-naphthalene.

Bellite.—Probably about 80 parts ammonic nitrate and 20 parts meta-di-nitro-benzol. In the schedule, No. 1 is stated to contain 79 to 81, and No. 3 from 92 to 94 parts of ammonia nitrate.

Roburite No. 3.—Defined as containing a mixture of ammonic nitrate, di-nitro-benzol and chloronaphthalene in such quantities that the chlorine shall form less than 1 per cent. of the total mass, which is practically identical with that licensed by the Inspectors of Explosives. Another formula gives it as consisting of 86 to 87 parts of ammonic nitrate, and 13 to 14 parts of chloro-di-nitro-benzol.

Dahmenite A.—Ammonic nitrate, naphthalene and potassic bichromate, the latter not to exceed 2·5 per cent. Its actual average composition, as used in practice, is said to be about 95 parts ammonic nitrate and 4·5 parts of naphthalene.

Electronite No. 2.—90 to 91 parts ammonic nitrate and 9 to 10 parts of wood-meal or starch.

Westfalite.—Ammonic nitrate 90 parts, resin 5 parts, and potassic bichromate 5 parts.

According to the Reports of the Royal Commission on Explosions from Coal-dust in Mines, "the so-called safety or flameless explosives are largely in use in all parts of the country, and as the results of practical experience, are generally pronounced to be effective substitutes for gunpowder, and certainly very much safer. Each of these compositions has its advocates, and each is said to be flameless, or practically so. As far as dust is concerned, the current opinion appears to be that they are perfectly safe, but there is a considerable doubt as to how far the small flash or scintillation which many witnesses say they display render them dangerous in the presence of gas."

The safety explosives should comply with the following special conditions:—

1. The temperature of detonation must be as low as possible.

2. The products of combustion must be non-combustible.

3. The products of decomposition must not be poisonous.

In using these explosives the cartridges should be exploded only by the detonators recommended for each of them, electric firing should be adopted, and 20 inches of clay stemming used in the holes to prevent flame, and thereby ensure the greatest safety.

In blasting weak rock or coal with these explosives, to obtain the best effect, and to prevent blown-out shots, it is very important to proportion the diameter of the hole inversely as the strength of the explosive, or the shearing force of the charge directly as the line of resistance, and also the length of charge as the section of rock or coal to be blasted, in accordance with the formula $\frac{F_1}{F} = \frac{S_1}{S}$.

CHAPTER XX.

INSTRUCTIONS FOR THE USE OF EXPLOSIVES.

130. *Directions for using Dynamite, Blasting Gelatine, Gelatine-Dynamite and other Gelatine Explosives.* — Unlike gunpowder, dynamite, blasting gelatine, gelatine-dynamite, and gelignite require a special mode of firing, which consists of a very strong percussion cap, called a "detonator," attached to a safety or electric fuse. The fuse explodes the fulminate in the detonator, which then explodes the cartridge.

A charge is made as follows :—

1*st Operation.*—A safety fuse is cut clean and inserted into a detonator, till it reaches the fulminate. The upper part of the cap is then squeezed with a pair of nippers (as shown in Fig. 81). The squeezing should not be neglected, as it not only secures the position of the fuse, but also serves to develop the power of the fulminate. For use under water great care should be taken to have the upper end of the detonator made watertight (with grease, tar or otherwise) where it joins the fuse, to prevent the fulminate from getting damp.

2nd Operation.—A primer or cartridge is opened at one end, and the detonator, with the fuse already attached to it, is pushed in so as to leave about one-

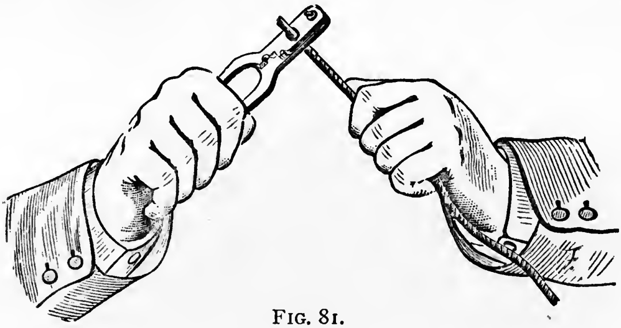

FIG. 81.

third of the copper tube exposed outside the cartridge (see Fig. 82). The detonator is then securely tied in that position. If the detonator is pushed too far into the cartridge the fuse may set fire to the

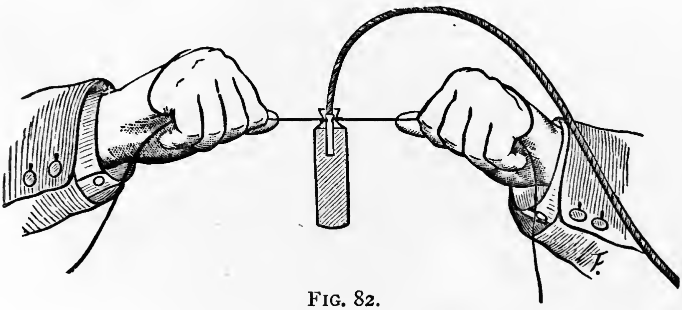

FIG. 82.

latter before the spark can explode the detonator, and unpleasant fumes may be the consequence.

3rd Operation.—One or more cartridges (as the height of charge may require) are inserted into the borehole, and each squeezed with a wooden rammer (as shown in Fig. 83) so as to completely fill out the borehole. Never use iron in squeezing home cartridges.

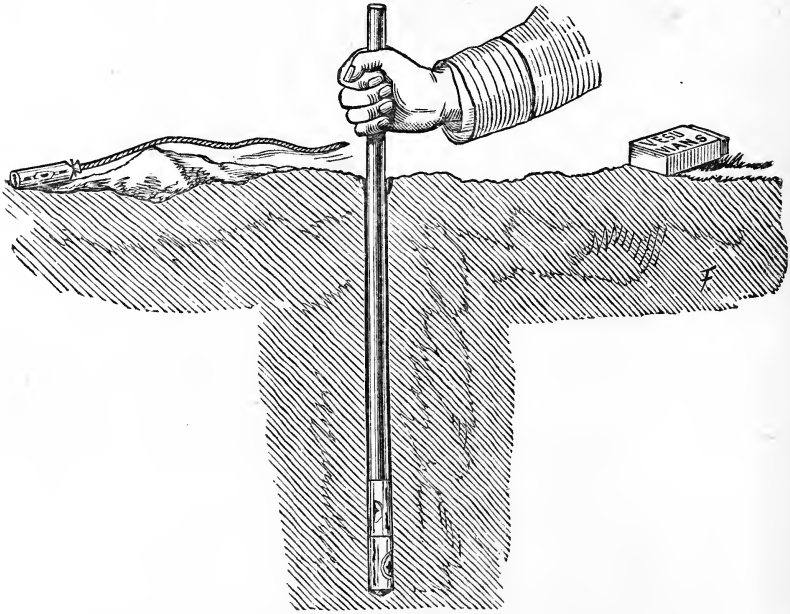

FIG. 83.

4*th Operation.*—Over the charge, as shown in the third operation, the cartridge, with detonator and fuse affixed, is inserted, but not squeezed, and loose sand or water is poured in as tamping (as shown in Fig. 84). The charge is then ready for firing.

131. *Directions for using Tonite.*—For tonite, the fixing of the detonator should be done as de-

scribed above under first operation, which is then pushed down into the cartridge as far as possible. The neck of each cartridge is furnished with a piece of wire, which must be twisted firmly round the fuse so as to make both fast together. The cartridge is then ready for use. Make sure that the borehole is large enough to let the charge to the very bottom, but it must not be too large, or else power will be wasted. When used in wet holes, the neck of the cartridge should be protected by tar or grease, to prevent water getting to the detonator.

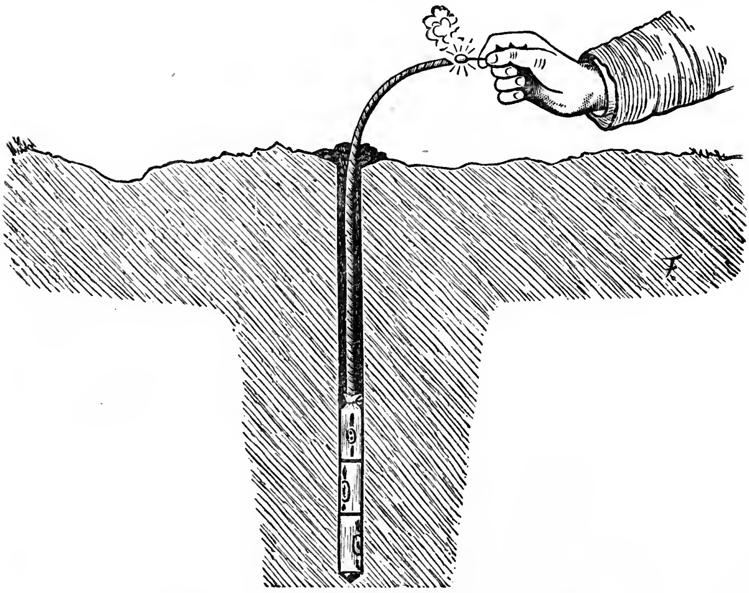

FIG. 84.

Where more than one cartridge is necessary to charge a mine, put in the hole as many cartridges as necessary (without detonators) and press them gently, one after the other, so as to leave no space between them; then introduce the cartridge containing the detonator, press it down carefully on account of the detonator inside, and tamp with clay or sand in the ordinary way. Cartridges without detonator holes are made for this purpose, but it is preferable to use one large cartridge when possible.

In case the cartridges which a miner has in stock do not fit the borehole, he can cut or break them in pieces, and press them down the hole if it is quite dry, reserving one of the top parts of the cartridges for the detonator, which with such part of cartridge is put in the hole last. The paper casing of the cartridges need not be removed.

132. *Directions for using Electronite and Blasting Amberite.*—Be careful to use a sufficiently strong detonator. Nothing less than a No. 6 is strong enough. Better results will be given by a No. 7 detonator. It is false economy to use weak detonators, for they are incapable of developing the full power of the explosives.

Electronite.—Cut the string with which the neck of the cartridge is tied, and having opened the neck, make a hole in the explosive with a pointed piece of metal rod or a stick, a little larger in diameter than the detonator. An ordinary lead pencil is a handy and efficient instrument for the purpose. Insert the detonator (previously fixed to the fuse as before described) into this hole, to a depth equal to at least half the length of the detonator, and tie the neck firmly to the fuse. When there is water in the borehole it will be necessary to put grease round the neck to keep out the water. If more than one cartridge is to be used for the shot, cut off the neck of that one which is not to have the detonator, about $\frac{1}{8}$ inch above the point where it is tied, and put it into the borehole neck end downwards. Grease the neck if there is water in the borehole. Then push the cartridge in which the detonator has been placed into the borehole till it comes in contact with the other cartridge. Stem lightly for about 3 inches on account of the detonator; then more firmly. Use cartridges which nearly fit the borehole, but see that they are not too large to go to the bottom.

Blasting Amberite.—This explosive is issued in two forms of cartridges, one "plain," being closed at both ends, the other "primers" which are left open at one end to receive the detonator. For small shots a primer will often be sufficient, for example, for breaking up boulders more will seldom be required. For larger shots, put into the borehole one or more of the plain cartridges, sufficient to make up, with the primer, the required length of charge, or height of explosive in the borehole.

Then having securely fixed the detonator to the fuse in the usual way, insert the detonator in the hole provided for that purpose in a primer, taking care to put it well down into the explosive, and tie the neck of the primer firmly to the fuse with a piece of wire or string. When there is water in the borehole, tallow or some other kind of grease should be put round the neck to make it water-tight. The primer thus prepared is to be pushed gently, on account of the detonator, down the bore-hole till it comes into contact with the plain cartridges. Stem lightly for the first 2 or 3 inches; then with more force. Hard stemming is not required.

When a whole plain cartridge would give too heavy a charge, it may be cut in two, and only a portion of it used. This should, however, not be done when there is water in the borehole. Use cartridges which nearly fit the borehole; but see that they are not too large to go to the bottom.

133. *Directions for Loading a Borehole with Miner's Coarse Ordinary Blasting Powder.*—The sludge from the boring of the hole should first be removed by a swopstick, and the hole dried by means of a wisp of hay, rag or tow passed through the eye of an iron rod, and forced slowly up and down the hole to absorb the moisture.

The powder is then poured into the hole through a copper or tin tube, so as to reach the bottom without touching the sides of hole above the limit of charge. If the hole be vertical or very steep the powder will drop in very freely to the bottom, but if the inclination be not very great, it must be pushed down with a wooden ramrod. If the hole be horizontal, a scoop or spoon is used which is filled with powder, inserted gently into the hole, and turned round at the end to deposit the powder at the bottom. The powder is pushed compactly to the bottom with a wooden ramrod. If the hole is inclined upwards, a paper cartridge is employed to hold the powder. In wet holes, watertight cartridges must be used with fuse attached. Before completing the charge, a Bickford safety or electric fuse is inserted into the hole sufficiently long to extend a few inches out of the hole, and the rest of charge then added so that the fuse is fixed well down in the powder. The charge is covered with a little dry clay, and the rest of the whole filled and tamped with clay or rotten stone crushed to a powder. The first 3 inches of tamping should be merely pressed down strongly with a wooden rammer, and the rest tamped strongly as it is filled into the hole with an iron rod tipped with copper, which is struck gently with a hammer to make the tamping as compact as possible. Care must be taken not to cut the fuse in carrying out this operation, and not to use materials that may strike fire.

134. *Directions for using the Nitrate of Ammonia Class of Safety Explosives.*—The detonator should always be inserted in the end of the cartridge nearest the mouth of the hole, and should only be planted deep enough to be just covered by the explosive. The end of the primer cartridge should be opened to admit the insertion of the detonator, which should then be secured by tying the mouth of the cartridge up again with a piece of string or wire. When possible, single cartridges should be used for the charge; these are made in weights varying from 1 to 16 ounces of various diameters. When having two or more cartridges to make up the charge, great care should be taken that they are in perfect contact without any dust between them. Cartridges should never be opened, except for the insertion of detonators; if in proper contact the whole charge should explode.

In charging holes the cartridges should be simply pressed home, and not rammed at all, as when rammed hard, not only is the cartridge broken but the explosive is compressed, in which condition it is more difficult to detonate. Any cartridge which appears to be hard should be rolled or squeezed to soften it before inserting in the holes. The stemming or tamping used should be of a soft not gritty nature, for fear of damaging the electric wires. The first six inches should be rammed tightly, and then the remainder of the hole rammed firmly. The total length of stemming should in all cases be not less than 20 inches. In using ordinary paper cartridges in wet holes it is necessary that the cartridge should be again sufficiently protected against wet where the detonator has been inserted, care being taken not to damage the cartridge case in charging.

CHAPTER XXI.

135. RECAPITULATION AND NOTATION OF THE MOST IMPORTANT FORMULÆ.

d = Diameter of borehole in inches.

D = Depth of borehole in inches for shearing resistance

$$= \frac{m}{2} + W = 6\,d + W.$$

m = Length of charge in any borehole for shearing $= 12\,d$.

N = Number of similar shotholes spaced a distance k apart in line parallel to a free face, and fired simultaneously.

k = Distance between shotholes.

θ = Chamber coefficient $= \frac{A}{S} = C_a\,W$.

$e\,W$ = Maximum distance that similar shotholes should be placed apart when in line parallel to a free face, and fired simultaneously, $e = \sqrt{2 \cdot 84\,\frac{S}{W}}$.

C_a = Coefficient of rock $= \frac{A}{S \times W}$.

C_v = Charging coefficient $= \dfrac{L}{W^3} = 3{\cdot}454\, g\, C_a^{\ 3}$.

N.B.—If W be taken in feet $C_v = 5969\, g\, C_a^{\ 3}$.

W = Line of resistance in inches

$$= \frac{A}{C_a S} = \frac{6}{13\, C_a} d.$$

L = Weight of charge in lbs. $= C_v W^3 = {\cdot}5434\, d^3$
$= 3{\cdot}454\, g\, C_a^{\ 3} W^3$.

N.B.—If W be taken in feet and d in inches $L = 5969\, g\, C_a^{\ 3} W^3$.

S = Periphery of charging chamber at right angles to line of resistance $= \dfrac{A}{C_a W}$.

A = Projection of charging chamber $= C_a S W$.

G = Number of similar shotholes required in a single exposed surface of rock when angled to meet at a point to "unkey" a line of resistance p W and placed so that they make an angle b with the latter, W being the line of resistance for one such shothole parallel to a free face in the same kind of rock

$$= \frac{12 + \operatorname{cosec} b}{13} p.$$

p_1 = Line of resistance for a shothole whose diameter is d, and angle to line of resistance b, the line of resistance being unity, for a shothole whose diameter is 1 inch, and angle to the line of resistance 90°

$$= \frac{13\,d}{12 + \operatorname{cosec} b}.$$

q = Line of resistance for the combined shearing force of N similar shotholes spaced a distance k apart in line parallel to a free face, when fired simultaneously, the line of resistance for one such shothole being unity

$$= \frac{\mathrm{N\,S}}{2\,\{(m + \mathrm{N}\,d) + k\,(\mathrm{N} - 1)\}}.$$

F = Sectional area of rock equal to resistance to shearing $= \dfrac{\cdot 032}{\mathrm{C}_a} \times 7\cdot 38\,d^2$.

N_0 = Number of holes required for headings or shafts $= 6\cdot 67\,\mathrm{C}_a = 4\cdot 24\,\mathrm{C}_a\,\mathrm{V}$.

m_1 = Length of charge for boreholes for a bed of rock whose thickness t is less than $e\,\mathrm{W}$

$$= \frac{t\,m}{e\,\mathrm{W}}.$$

M = Maximum pressures developed by different explosives. In similar holes charged with the same volume of explosive $\dfrac{\mathrm{M}_1}{\mathrm{M}} = \dfrac{\mathrm{W}_1}{\mathrm{W}}$.

L_1 = Charge required when $\dfrac{1}{n}$th more explosive is required for a blast whose angle is b, than for a horizontal blast

$$= \frac{1 + \sin b}{n}\,\mathrm{C}_v\,\mathrm{W}^3.$$

CHAPTER XXII.

EXAMPLES OF ALL THE MORE USEFUL AND IMPORTANT CALCULATIONS THAT ARE LIKELY TO OCCUR IN THE DAILY PRACTICE OF ROCK BLASTING, AND OF THE USE OF THE TABLES FOR FACILITATING THE CALCULATIONS.

136. *Example* 1. *Line of Resistance, Depth of Borehole, and Charge.*—For a blast in hard rock, find the line of resistance, depth of borehole, weight and length of a suitable charge of gelatine-dynamite, and approximate volume of rock which should be dislodged if the chamber coefficient be ·018, diameter of hole $1\frac{1}{8}$ inch, and there are three free faces at right angles to each other.

By reference to Table I. it will be found that the line of resistance is 2 feet $4\frac{3}{4}$ inches.

According to Table III. the depth of $1\frac{1}{8}$ inch diameter borehole required for a line of resistance 2 feet 3 inches is 2 feet 10 inches, and for a line of resistance 2 feet 6 inches it is 3 feet 1 inch, and as the mean of these lines of resistance is

$$\frac{2 \text{ ft. } 3 \text{ in.} + 2 \text{ ft. } 6 \text{ in.}}{2} = 2 \text{ ft. } 4\tfrac{1}{2} \text{ in.},$$

the depth of borehole required for a line of resistance 2 feet $4\frac{1}{2}$ inches is

$$\frac{2 \text{ ft. } 10 \text{ in.} + 3 \text{ ft. } 1 \text{ in.}}{2} = 2 \text{ ft. } 11\tfrac{1}{2} \text{ in.}$$

Therefore in practice we should bore the hole, say 3 feet deep.

The weight of gelatine-dynamite is given in Table II., viz. ·749 lb., or say $\frac{3}{4}$ lb., which should have a length of $12 \times 1\frac{1}{8}$ inch = $13\frac{1}{2}$ inches in the hole.

Table VI. gives the approximate volume of rock which would be blasted for a line of resistance of 2 feet 6 inches, viz. 36·46 cubic feet.

137. *Example* 2. *Blasting a Bench of Rock.*—What depth and diameter of borehole and line of resistance will enable the whole height of a bench of rock, 6 feet high, to be blasted so as to maintain a level floor, gelatine-dynamite being employed, and the coefficient of the rock with this explosive being ·016?

The depth of borehole must be the height of bench, viz. 6 feet, as any shorter hole would result in a sloping floor.

The diameter of hole, and line of resistance required for the 6-feet hole can be found from Table I. by making the line of resistance plus 6 times the diameter of the hole equal to the depth of borehole or height of bench, in accordance with the formula

$$D = W + \frac{m}{2} = W + 6\,d.$$

In Table I. we cannot find any line of resistance with its corresponding diameter to agree exactly with the above formula for the coefficient ·016, therefore we must take the mean of the nearest figures above and below, as follows:—

Line of resistance + 6 times diameter of hole = height of bench = 72 inches.

	Inches.	Inches.	Inches.
(1)	57·50	+ 6 × 2	= 69·50
(2)	61·09	+ 6 × $2\frac{1}{8}$	= 73·84
	2)118·59	2)$4\frac{1}{8}$	2)143·34
	mean 59·30	mean $2\frac{1}{16}$	mean 71·67

It is therefore clear that a line of resistance of 59·30 inches (or say 5 feet), and a borehole having a length of 6 feet and a diameter of $2\frac{1}{16}$ inches, meets the conditions of this case.

138. *Example* 3. *Height of Step or Bench of Rock.*—What height of bench should be worked by $1\frac{1}{2}$ inch diameter boreholes when the chamber coefficient of the rock for the explosive to be used is ·02?

The height of bench should be equal to the sum of the line of resistance and half the length of charge, and, as in Table I. the line of resistance for a $1\frac{1}{2}$ inch borehole when C = ·02 is $34\frac{1}{2}$ inches, the height of the bench should be W + 6 *d*, or

$$34\tfrac{1}{2} + 9 = 43\tfrac{1}{2} \text{ inches.}$$

139. *Example 4. Charging Coefficient and Volume of Rock.*—Find the charging coefficient of a $1\frac{1}{2}$ inch borehole, and the volume of rock which will be blasted for one, two, three, four or five free faces when gelignite is used, and the chamber coefficient is ·016.

Table I. gives the line of resistance 43·12 inches = 3·6 feet, and Table II. the weight of charge 1·775 lb. Hence the charging coefficient is

$$C = \frac{1 \cdot 775}{(3 \cdot 6)^3} = \cdot 038.$$

According to Table VI. the approximate volumes of rock which will be blasted are—

For one free face, $1\frac{1}{3}\,W^3 = 62 \cdot 10$ cubic feet.
For two free faces, $1\frac{2}{3}\,W^3 = 77 \cdot 76$ cubic feet.
For three free faces, $2\frac{1}{2}\,W^3 = 108 \cdot 70$ cubic feet.
For four free faces, $3\,W^3 = 139 \cdot 80$ cubic feet.
For five free faces, $4\,W^3 = 186 \cdot 40$ cubic feet.

It is important to note that these volumes have the following ratio to each other:

$$1 : 1\tfrac{1}{4} : 1\tfrac{3}{4} : 2\tfrac{1}{4} : 3,$$

and that the same quantity of explosive is used in each case to overcome the cohesive resistance of the rock; moreover that the charging coefficient will vary according to the number of free faces.

140. *Example 5. Economy of Proportioning Depth and Diameter of Borehole to Height of Bench of*

Rock.—Supposing that it has been found that 1 inch diameter boreholes 3 feet 6 inches deep so placed to have a line of resistance of 3 feet with two free faces, when charged with 1 foot or ·543 lb. of dynamite, and fired consecutively, will blast a bench of rock 3 feet 6 inches high; what then would be the relative economy if shotholes of the same diameter were used under like conditions for blasting a bench of rock 6 feet high?

It is clear that the depth of borehole required to blast a 6 foot bench is 6 feet, and that the line of resistance for this depth of hole should be the same as for the 3 foot 6 inch one, as it has the same diameter (namely 3 feet); as also that it is necessary to have the same length of tamping to blast away the whole length of the bench of rock, and maintain equilibrium of resistance on all sides of the charge.

The length of tamping required is

$$T = D - m,$$

T denoting the length of tamping, D the depth of hole, and m the length of charge.

Consequently,

$$T = 3 \text{ ft. } 6 \text{ in.} - 1 \text{ ft.} = 2 \text{ ft. } 6 \text{ in.}$$

The length of charge used in the 6 foot hole must be

$$m = T - D$$

or $$6 \text{ ft.} - 2 \text{ ft. } 6 \text{ in.} = 3 \text{ ft. } 6 \text{ in.}$$

This is three and a half times as long as that used in the 3 foot 6 inch hole; hence its weight will be

$$3\tfrac{1}{2} \times \cdot 543 = 1 \cdot 9 \text{ lb.}$$

On the contrary, the approximate volumes of rock blasted will be

(A) By means of the 1 foot charge, $3\frac{1}{2} \times 3^2 = 31\frac{1}{2}$ cubic feet.

(B) By means of the 3 foot 6 inch charge, $6 \times 3^2 = 54$ cubic feet.

Then, as $\dfrac{3\frac{1}{2}}{31\frac{1}{2}} = \dfrac{6}{54} \doteq \cdot 111$ foot of rock bored per cubic foot of rock blasted, there would be the same economy in boring; whereas the relative economy in explosive would be as follows:

For (A) $\dfrac{\cdot 543}{31 \cdot 5} = \cdot 01724$ lb. of dynamite per cubic foot of rock.

For (B) $\dfrac{3\frac{1}{2} \times \cdot 543}{54} = \cdot 0352$ lb. of dynamite per cubic foot of rock.

Consequently, the working of a 6 foot bench under the given conditions necessitates a consumption of

$$\frac{(\cdot 0352 - \cdot 01724)\, 100}{\cdot 01724}$$

= 104 per cent. more dynamite than the shorter bench.

As, however, the conditions of rock-boring necessitate the holes being bored more or less conical

—for instance, the 3 foot 6 inch hole would be, say, $1\frac{1}{4}$ inch diameter at top and 1 inch at bottom, and the 6 foot hole $1\frac{5}{8}$ inch at top and 1 inch at bottom —the relative economy in boring would be more favourable for the shorter holes than shown by the above calculation.

This example shows the importance of boring holes on correct principles to attain the greatest economy in explosive.

141. *Example* 6. *Maximum Distance between Shotholes Fired Simultaneously.*—What is the maximum distance two shotholes of $1\frac{1}{2}$ inch diameter, which are to be charged with dynamite and fired simultaneously, can be placed apart, parallel to a straight free face, so that the whole of the intervening rock will be carried away by the blast, if the chamber coefficient for the rock be ·02, and the length of charge 12 d?

According to Table I., the line of resistance should be $34\frac{1}{2}$ inches.

The maximum distance which the holes can be placed apart may be represented by eW in which $e = \sqrt{2 \cdot 84 \frac{S}{W}}$, and as for a $1\frac{1}{2}$ inch hole S = 39 inches, we have

$$e = \sqrt{\frac{2 \cdot 84 \times 39}{34 \cdot 5}} = 1 \cdot 8.$$

Hence the greatest distance between the shotholes should be

$$1 \cdot 8 \times 34 \cdot 5 = 62 \cdot 10 \text{ in. or } 5 \text{ ft. } 2\tfrac{1}{10} \text{ in.}$$

142. *Example 7. Line of Resistance for Two Shotholes Supporting Each Other.*—Calculate the line of resistance for two 2 inch shotholes which are to be bored 6 inches apart in line parallel to a free face and fired simultaneously, if the chamber coefficient of the rock be ·02.

Table I. shows the line of resistance for a single 2 inch hole to be 46 inches, and according to Table IV. the line of resistance for two such holes 6 inches apart is

$$1\cdot53 \times 46 = 70\cdot38 \text{ in. or } 5 \text{ ft. } 10\tfrac{2}{5} \text{ in.}$$

143. *Example 8. Economy of Shotholes Supporting Each Other.*—What is the relative economy in boring and in explosive by firing two 2 inch shotholes simultaneously under the conditions given in Example VII., as compared with firing a single 2 inch hole in the same kind of rock, assuming that there are three free faces in each case, and that the holes are fired in benches of rock with charges of dynamite?

For the single shotholes we have:

1. (*a*) The line of resistance given in Example 7

   46 inches = 3·83 feet.

   (*b*) From Table II., the weight of charge

   = 4·35 lbs.

   (*c*) The charging coefficient $= \dfrac{4\cdot35}{(3\cdot83)^3} = \cdot077.$

And on the contrary for the two 2 inch shotholes 6 feet apart fired simultaneously:

2. (*a*) The line of resistance given in Example 7, $70{\cdot}38$ inches $= 5{\cdot}84$ feet.

(*b*) The weight of the two charges $= 4{\cdot}35$ lbs. $\times 2 = 8{\cdot}7$ lbs.

(*c*) The charging coefficient $= \dfrac{8{\cdot}7}{(5{\cdot}84)^3} = {\cdot}044.$

Assuming then the relative economy in explosive to be as the charging coefficient there will be a saving of

$$\frac{({\cdot}077 - {\cdot}044)\ 100}{{\cdot}077} = 42\tfrac{6}{7} \text{ per cent.}$$

by the latter method of working.

It is evident that the length of the single hole should be

$$3{\cdot}83 \text{ feet} + 1{\cdot}00 \text{ feet} = 4{\cdot}83 \text{ feet,}$$

and of the double holes 6 feet apart,

$$5{\cdot}84 + 1{\cdot}00 = 6{\cdot}84 \text{ feet.}$$

Then, as the quantities of rock blasted may be taken as proportional to the cubes of the line of resistance, the comparative economy in boring will be as

$$\frac{4{\cdot}83}{(3{\cdot}83)^3} :: \frac{2\,(6{\cdot}84)}{(5{\cdot}84)^3},$$

or $20{\cdot}9$ less boring will be required per cubic foot of rock blasted by the second method of blasting with two holes simultaneously.

144. *Example* 9. *Economy of Firing Shotholes Simultaneously.*—What economy may be effected by simultaneous firing of 2 inch diameter shotholes in a bench of strong rock 6 feet high and 34 feet long, it having been found by trial shots that the chamber coefficient for gelatine dynamite is ·018, and that holes placed at a distance double the length of the line of resistance apart will carry away the whole of the intervening rock, as compared with the blasting of the same bench of rock by consecutive shots with holes of the same diameter? It is evident that the length of the shotholes should be 6 feet.

Table I. gives the line of resistance for a 2 inch borehole as 4 feet 3 inches; hence the number of holes required is

$$\frac{34}{2 \times 4\frac{1}{4}} = 4 \text{ holes},$$

as illustrated in Fig. 85.

Table III. gives the weight of gelatine dynamite required for a 2 inch hole, viz. 4·2 lbs., and therefore the total charge for the four holes will be

$$4{\cdot}2 \text{ lbs.} \times 4 = 16{\cdot}8 \text{ lbs.}$$

The volume of rock blasted will be

$$34 \times 6 \times 4\tfrac{1}{4} = 867 \text{ cubic feet.}$$

Consequently, each lb. of explosive will blast

$$\frac{867}{16{\cdot}8} = 51{\cdot}6 \text{ cubic feet of rock}$$

On the contrary, the best result obtainable in blasting with a single hole is that shown for the lines of rupture for the hole *h*, which is

$$1\frac{1}{2} \times 4\frac{1}{4} \times 4\frac{1}{4} \times 6 = 162 \cdot 54 \text{ cubic feet of rock.}$$

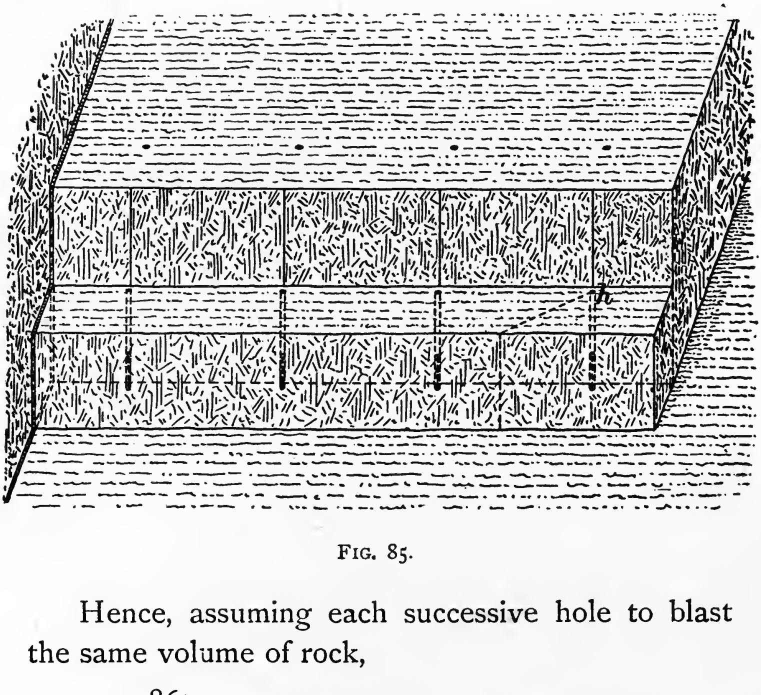

FIG. 85.

Hence, assuming each successive hole to blast the same volume of rock,

$$\frac{867}{162 \cdot 54} = 5 \cdot 3 \text{ holes would be required.}$$

Or each lb. of explosive will blast $\frac{162 \cdot 54}{4 \cdot 2} = 38 \cdot 7$ cubic feet of rock.

Therefore the economy effected in boring would be

$$\frac{(5 \cdot 3 - 4)\ 100}{5 \cdot 3} = 24 \cdot 53 \text{ per cent.},$$

and in explosive

$$\frac{(51 \cdot 6 - 38 \cdot 7)\ 100}{51 \cdot 6} = 25 \text{ per cent.}$$

145. *Example* 10. *Number of Shotholes required to Unkey a Face of Rock.*—How many boreholes 1 inch in diameter, bored at an angle of 10° with the line of resistance, should be employed to unkey the end of a tunnel for a length of 3 feet 6 inches, the chamber coefficient of the rock being ·02, and the holes being fired simultaneously ?

According to Table I., the line of resistance is 23 inches for a shothole 1 inch in diameter, placed parallel to a free face when the co-efficient of the rock is ·02.

Then, according to the formula

$$G = \frac{12 + \operatorname{cosec} b}{13} p,$$

in which G = number of holes, $b = 10°$, and $p = \frac{42}{23}$ inches.

$$G = \frac{12 + 5 \cdot 76}{13} \times \frac{42}{23} = 2\tfrac{1}{2}.$$

Therefore three holes would have to be bored to converge to a point, but as three shotholes are capable of taking out a greater length of core than 3 feet 6 inches, it would be more advantageous to bore the three holes to converge at a point 4 feet 2 inches from the face of the tunnel, and thus to make

$$G = \frac{12 + 5 \cdot 76}{13} \times \frac{50}{23} = 3.$$

146. *Example* 11. *Line of Resistance and Charge in a Bed of Rock.*—Find the line of resistance, length and quantity of charge for blasting a bed of rock whose thickness is 3 feet, if $1\frac{1}{2}$ inch diameter bore-holes and dynamite be employed, assuming the chamber coefficient of the rock to be ·02.

According to Table I., the line of resistance is $34\frac{1}{2}$ inches, and the length of charge $12\,d = 18$ inches, when there is equal resistance on each side of the charge. In the example this is not the case, owing to the thickness of the bed being less than 2 W, and consequently the length of charge should be—

$$m_1 = \frac{t\,m}{2\,W};$$

or as $t = 36$ in., $W = 34\frac{1}{2}$ in., and $m = 18$ in.

$$m_1 = \frac{36}{2 \times 34\frac{1}{2}} \times 18 = 9 \cdot 4 \text{ inches.}$$

Therefore, as Table III. gives the weight of 18 inches dynamite, in a $1\frac{1}{2}$ inch hole as 1·833 lb., the weight of the 9·4 inches of such charge is

$$\frac{9 \cdot 4}{18} \times 1 \cdot 833 \text{ lb.} = \cdot 957 \text{ lb.}$$

147. *Example* 12. *Position, Depth and Diameter of Boreholes in Jointed Rock.*—The granite in a quarry being jointed as represented in Fig. 86, and the distances between the "master" joints a, a_1 and $d\,b_1$ and the bedding joints b, b_1 and $a\,d$ being 10 feet and

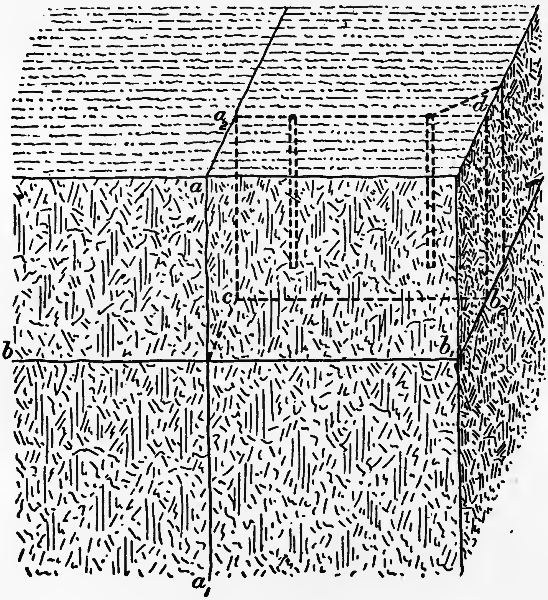

FIG. 86.

8 feet respectively, whereas the coefficient of the rock is, say ·014 for dynamite, and ·028 for powder, and there being practically no cohesive resistance along the joints; it is required to determine the position, depth and diameter of the holes to be bored for each explosive, and to adjust the same so as to shatter the rock as little as possible.

The best position of the main face of working is evidently at right angles to the "master" joints, as the blasts have to be arranged to fracture only the section of rock included by the "master" and bedding joints.

The holes should be vertical, and the line of resistance a length $\frac{eW}{2}$, or as this may be put equal to the distance between the "master" joints, we have

$$\frac{eW}{2} = \frac{10}{2} = 5 \text{ feet.}$$

For this line of resistance (see Table I.), a $1\frac{7}{8}$-inch borehole is necessary for the coefficient ·014, as for a charge of dynamite; and a $3\frac{3}{4}$-inch borehole for the coefficient ·028, as for a charge of powder. But as it is clear that there will be no shearing resistance to be considered under the given conditions, we may advantageously employ, say two $\frac{1\frac{7}{8}}{2} = \frac{15}{16}$ inch diameter boreholes for blasting with dynamite; and say two $\frac{3\frac{3}{4}}{2} = 1\frac{7}{8}$ inches diameter boreholes for blasting with powder instead, for the following reasons:—

1. The rupturing force will be practically the same, but the shock will be distributed, and its shattering effect greatly reduced.

2. There will be a great economy in the consumption of the explosive.

3. It is more economical to bore holes of small diameter.

4. The smaller quantity of explosive used will be sufficient to loosen the rock.

5. The ballistic force will be less.

According to the formula $F = \frac{\cdot 032}{c_a} \times 7 \cdot 38\, d^2$, the section F of rock which may be ruptured by a $\frac{15}{16}$-inch diameter hole is 14·82 square feet, if the coefficient of the rock is ·014, and 29·64 square feet for a $1\frac{7}{8}$-inch diameter hole, if the coefficient is ·028, that is when the length of charge is twelve times the diameter of hole; therefore, with this length of charge, as the section of rock to be fractured is a_2, d, b_2, $c = 10 \times 8 = 80$ sq. feet,

$\frac{80}{14 \cdot 82} = 6$ or six $\frac{15}{16}$ inch diameter holes are required for dynamite, and

$\frac{80}{29 \cdot 64} = 3$ or three $1\frac{7}{8}$ inch diameter holes are required for gunpowder, when the length of charge is $12\, d$.

But as there is no shearing resistance, two holes may be used in each case if the length of charge be increased, namely, to—

$$\frac{(12 \times \frac{15}{16}) \times 6}{2} = \frac{(12 \times 1\frac{7}{8}) \times 3}{2} = 33 \cdot 75 \text{ in.} = 2 \text{ ft. } 9\tfrac{3}{4} \text{ in.}$$

The depth of the holes should therefore be

$$4 \text{ feet} + \frac{2 \text{ feet } 9\frac{3}{4} \text{ in.}}{2} = 5 \text{ feet } 4\frac{7}{8} \text{ inches.}$$

The charge of dynamite for the two $\frac{15}{16}$ inch diameter holes will be

$$2 \times 1 \cdot 34 \text{ lb.} = 2 \cdot 68 \text{ lbs.}$$

And of gunpowder for the two $1\frac{7}{8}$ inch diameter holes,

$$2 \times 3 \cdot 36 \text{ lbs.} = 6 \cdot 72 \text{ lbs.}$$

The two holes in each case must be fired simultaneously. They should be placed as shown in the figure.

148. *Example* 13. *Number of Shotholes required for a Heading.*—If in rock whose coefficient is ·03 for gelatine-dynamite a 7 feet × 7 feet heading can be advanced 3 feet 6 inches with 20 $1\frac{1}{8}$-inch diameter holes, how many holes will be required to make the same advance in a heading of the same size in rock whose coefficient is ·016 ?

In this case we have

$$\frac{N_1}{N} = \frac{C_{a_1}}{C_a}.$$

Therefore the number of holes required is

$$N = \frac{\cdot 016}{\cdot 030} \times 20 = 11 \text{ holes.}$$

N.B.—The 11 holes must be so placed that the resistance to each is equal.

If in a heading 7 feet × 7 feet worked on the square cut system in rock whose coefficient of strength is ·024, 22 $1\frac{1}{8}$ inch diameter holes are required to advance the heading 3 feet 6 inches, what number of holes should sink 3 feet 6 inches in a rectangular shaft 14 feet × 8 feet in similar rock?

The number of holes are given by the formula,

$$N = 4{\cdot}24\, C_a\, V.$$

And as the volume of rock V which will be blasted is

$$14 \times 8 \times 3\tfrac{1}{2} = 392 \text{ cubic feet,}$$

$$N = 4{\cdot}24 \times {\cdot}024 \times 392 = 40 \text{ holes nearly.}$$

For the method of placing the holes see Figs. 57, 58, 59, 60 and 61.

149. *Example* 14. *Position of Chambers, and Charge for a Large or Giant Blast.*—If the coefficient of the rock is ·03 for coarse blasting powder which has a specific gravity of 0·7 in bulk, and one pound of such powder will give the required ballistic effect to three tons of rock (as found by trial blasts), the specific gravity of the rock being 2·62, what weight of charge and dimensions of chambers will be required to blast a line of resistance of 50 feet in a bench of rock, 150 feet long and 53 feet high, the ends of which are open as illustrated in Figs. 87, 88 and 89, if each chamber be given a square section at right angles to the line of resistance?

For shearing we should make the sectional area of chamber at right angles to line of resistance $A = C_a S W$, and we have for the chamber co-efficient (see Art. 38)

$$\theta = \frac{A}{S} = C_a W = \cdot 03 \times 50 = 1 \cdot 5.$$

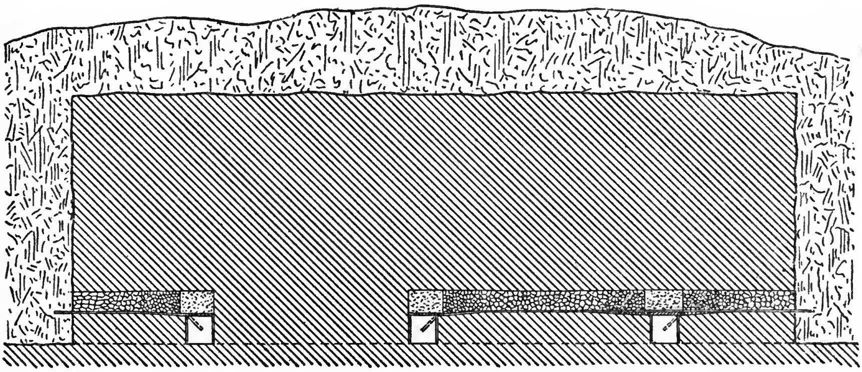

FIG. 87.

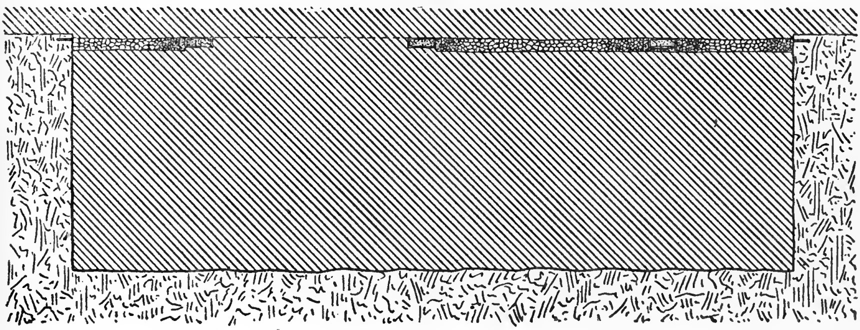

FIG. 88.

But we have for a square section of chamber whose side is l. (Art. 38.)

$$\frac{A}{S} = \frac{l}{4}.$$

Therefore $\frac{l}{4} = 1 \cdot 5$ and

$$l = 6 \text{ feet}.$$

Hence the length of the chamber is 6 feet, and

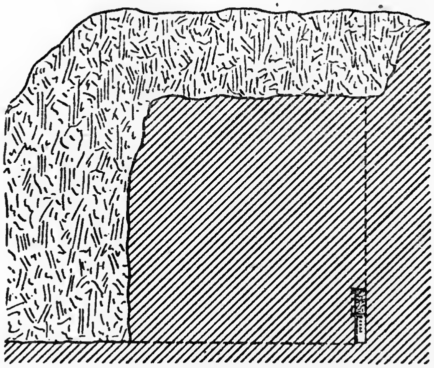

FIG. 89.

its depth 6 feet, and the area A = 6 feet × 6 feet = 36 sq. feet.

The width of the chamber will be determined by the volume of the charge as calculated below.

The number of chambers required, as they may be placed a distance of e W apart, is $\frac{150}{e\,W}$, and as

$$e = \sqrt{\frac{2 \cdot 84\, S}{W}}$$

$$e = \sqrt{\frac{2 \cdot 84 \times 24}{50}} = 1 \cdot 16 \quad \therefore \quad \frac{150}{eW} = \frac{150}{50 \times 1 \cdot 16}$$

$= 2 \cdot 6$, that is, three chambers are required.

The volume of rock to be blasted is

$$150 \times 53 \times 50 = 397500 \text{ cubic feet.}$$

For a specific gravity of $2 \cdot 62$ there will be $13\frac{1}{2}$ cubic feet in one ton, and consequently the weight of the 397,500 cubic feet is

$$\frac{397500}{13 \cdot 5} = 29444 \text{ tons.}$$

The weight of powder therefore required is

$$\frac{29444}{3} = 9815 \text{ lbs.}$$

which is to be placed in three chambers, or

$$\frac{9815}{3} = 3272 \text{ lbs. in each.}$$

The volume of 3272 lbs. of powder is

$$\frac{3272}{\cdot 7 \times 62 \cdot 4} = 75 \text{ cubic feet.}$$

Consequently each chamber should have a width of

$$\frac{75}{36} = 2 \cdot 08 = 2 \text{ feet } 1 \text{ inch.}$$

Therefore the cross section of each chamber will be 6 feet $\times$ 2 feet 1 inch, and the shearing force of the charge corresponding thereto equal to a line of resistance of

$$\frac{A}{C_a S} = 25 \cdot 8 \text{ feet.}$$

The chambers should be situated at the base of the bench of rock, and should be so placed that each is given a line of resistance of 50 feet towards the front face; the one midway between the end faces, and the others so that the one chamber is 25·8 feet from one end face, and the other chamber 25·8 feet from the other end face. Consequently, the distance between the central and each end chamber will be 40·2 feet or less than e W, which is a favourable condition for the rupture of the intervening rock.

When such large quantities of explosive are used great care must be taken to ensure an efficient blast. The chief consideration is that the charge should be able to overcome the resistance of the mass, and to ensure this the chambers should be dimensioned to enable the charges to overcome a considerably greater line of resistance than the actual. For example, if 15 per cent. margin be considered ample to meet all contingencies, this will be provided by increasing the side l of chamber 15 per cent., namely from 6 feet to 6·9 feet, and consequently making the section of chambers opposite the front face, or at right angles to the line of resistance, 6·9 feet × 6·9 feet. On the other hand, as the cubical contents of the chambers must be the same in each case, the width of each chamber will be reduced from 2·08 feet to 1·58 foot.

The positions of the chambers are given in the figures. An enlarged view of one of the chambers is shown in Fig. 90.

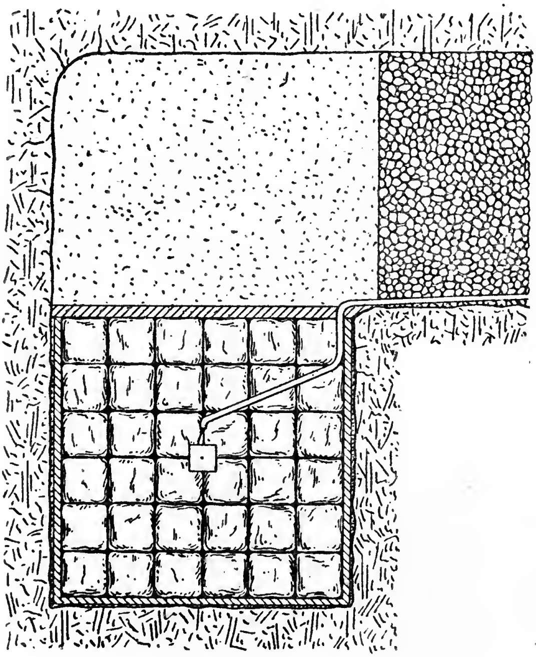

FIG. 90.

N.B.—If there were no shearing resistance, an elongated form of chamber would be preferable.

The charge (in suitable packages) should be placed in the chamber in a watertight deal box, of say $1\frac{1}{2}$ inch thick boards, and the open spaces between the rock and the sides of the box filled completely with fine sand; hence the dimensions of each chamber which has to be excavated in the rock are:

Length	6 ft. 11 in.	$+ (2 \times 1\frac{1}{2}$ in.)	$=$	7 ft. 2 in.
Depth	6 ft. 11 in.	$+ (2 \times 1\frac{1}{2}$ in.)	$=$	7 ft. 2 in.
Width	1 ft. 7 in.	$+ (2 \times 1\frac{1}{2}$ in.)	$=$	1 ft. 10 in.

In this case the chambers are best excavated by driving horizontal headings, which must turn at right angles at least once on their way to the powder chambers, to prevent the tamping being blown out. The size of the headings should be about 5 feet $\times$ 3 feet. If the ends of the bench of rock were not open a vertical shaft would have to be sunk to the central chamber, and headings driven from the bottom of the shaft to the other chambers.

150. *Range of Consumption of Explosive in Quarries, Tunnels and Mines.*—The results obtained in practice, except under the most exceptional conditions, are as follows:—

(*a*) For small blasts in open workings, a consumption of explosive ranging from $\frac{1}{4}$ to $\frac{1}{2}$ lb. of powder, and from $\frac{1}{16}$ to $\frac{1}{8}$ lb. of dynamite per ton of rock respectively.

(*b*) For large blasts in open workings, a consumption of explosive ranging from $\frac{1}{6}$ to $\frac{1}{2}$ lb. of powder, and from $\frac{1}{24}$ to $\frac{1}{8}$ lb. of dynamite per ton of rock respectively.

(*c*) For headings and tunnels, a consumption of explosive ranging from ½ to 2 lbs. of dynamite per ton of rock.

The above figures show to what extent the consumption of explosive may be influenced by the structure and tenacity, or cohesive resistance, of the rock, and the size and shape of the workings. The quantity of explosive required in any case, when the special conditions are given, may easily be calculated as explained in the preceding examples.

TABLE I.—MAXIMUM LINES OF RESISTANCE W in Inches for Dynamite
Length of

Direction of

Diameter of Chamber in inches.	Coefficients C_a.								
	·04	·038	·036	·034	·032	·03	·028	·026	24
	Very strong rock, $e = 2·38$.					Strong			
$\frac{3}{4}$	8·62	9·08	9·58	10·15	10·78	11·50	12·32	13·27	14·37
$\frac{7}{8}$	10·06	10·60	11·17	11·84	12·57	13·42	14·37	15·48	16·76
1	11·50	12·10	12·78	13·53	14·37	15·33	16·43	17·69	19·16
$1\frac{1}{8}$	12·94	13·61	14·37	15·22	16·16	17·24	18·48	19·90	21·55
$1\frac{1}{4}$	14·38	15·12	15·97	16·91	17·96	19·16	20·54	22·11	23·95
$1\frac{3}{8}$	15·81	16·63	17·57	18·60	19·75	21·08	22·59	24·32	26·34
$1\frac{1}{2}$	17·25	18·16	19·16	20·30	21·56	23·00	24·64	26·54	28·74
$1\frac{5}{8}$	18·69	19·67	20·76	21·99	23·36	24·92	26·69	28·75	31·15
$1\frac{3}{4}$	20·12	21·18	22·36	23·68	25·15	26·84	28·75	30·96	33·53
$1\frac{7}{8}$	21·56	22·70	23·95	25·37	26·94	28·75	30·80	33·17	35·92
2	23·00	24·20	25·56	27·06	28·74	30·66	32·86	35·38	38·32
$2\frac{1}{8}$	24·44	25·71	27·15	28·75	30·53	32·57	34·91	37·59	40·71
$2\frac{1}{4}$	25·88	27·22	28·75	30·44	32·33	34·49	36·97	39·80	43·11
$2\frac{3}{8}$	27·31	28·75	30·35	32·13	34·12	36·41	39·02	42·01	45·50
$2\frac{1}{2}$	28·75	30·26	31·94	33·82	35·92	38·32	41·08	44·22	47·90
$2\frac{5}{8}$	30·19	31·77	33·54	35·52	37·73	40·25	43·12	46·44	50·29
$2\frac{3}{4}$	31·62	33·26	35·14	37·20	39·50	42·16	45·18	48·64	52·68
$2\frac{7}{8}$	33·06	34·80	36·73	38·90	41·31	44·08	47·23	50·86	55·08
3	34·50	36·30	38·34	40·59	43·11	45·99	49·29	53·07	57·48

N.B.—This table is also applicable for any other explosive for which the
or coefficient of the rock is ·03 for gunpowder, the line of resistance for

Charges in Boreholes calculated from the formula $W = \frac{A}{C_a S} = \frac{\cdot 46\, d}{C_a}$ for Charge = 12 d.

blast horizontal.

Coefficients C_a.

·022	·02	·018	·016	·014	·012	·01	·008	·006
rock, $e = 2$.		Moderately strong rock, $e = 1\frac{1}{2}$.					Weak rock, $e = 1$.	
15·68	17·25	19·17	21·56	24·64	28·75	34·50	43·12	57·50
18·29	20·12	22·36	25·15	28·75	33·54	40·25	50·30	67·08
20·90	23·00	25·55	28·75	32·85	38·33	46·00	57·50	76·66
23·51	25·87	28·74	32·34	36·95	43·12	51·75	64·69	86·24
26·12	28·75	31·94	35·93	41·06	47·91	57·50	71·87	95·82
28·73	31·62	35·13	39·53	45·17	52·70	63·25	79·06	105·40
31·36	34·50	38·34	43·12	49·28	57·50	69·00	86·24	115·00
33·97	37·38	41·53	46·71	53·39	62·29	74·75	93·43	124·58
36·58	40·25	44·72	50·30	57·50	67·08	80·50	100·62	134·16
39·19	43·12	47·91	53·90	61·60	71·87	86·25	107·80	143·74
41·80	46·00	51·10	57·50	65·70	76·66	92·00	115·00	153·32
44·41	48·87	54·29	61·09	69·80	81·45	97·75	122·19	162·90
47·02	51·75	57·49	64·68	73·91	86·24	103·50	129·37	172·48
49·63	54·62	60·68	68·28	78·02	91·03	109·25	136·56	182·06
52·24	57·50	63·88	71·86	82·12	95·82	115·00	143·74	191·64
54·87	60·38	67·08	75·46	86·24	100·62	120·75	150·93	201·24
57·46	63·24	70·26	79·06	90·34	105·40	126·50	158·12	210·80
60·09	66·12	73·46	82·65	94·45	110·20	132·25	165·30	220·40
62·70	69·00	76·65	86·25	98·55	114·99	138·00	172·50	229·98

coefficient C_a of the rock has been found. For example, if the strength this explosive will be given under the coefficient C_a = ·03 in the table.

TABLE II.—WEIGHT IN LBS. OF CHARGE FOR BOREHOLES

Calculated from formula $L = \cdot 3396\, g\, d^3$ for a length of charge $12\, d$.

Name of Explosive.	Specific Gravity of Explosive.	Diameter of Boreholes in inches.												
		$\frac{3}{4}$	$\frac{7}{8}$	1	$1\frac{1}{8}$	$1\frac{1}{4}$	$1\frac{3}{8}$	$1\frac{1}{2}$	$1\frac{5}{8}$	$1\frac{3}{4}$	$1\frac{7}{8}$	2	$2\frac{1}{4}$	$2\frac{1}{2}$
Electronite	0·806	·116	·183	·274	·390	·535	·712	0·928	1·175	1·468	1·806	2·190	3·120	4·280
Blasting powder ..	1·000	·143	·228	·340	·484	·664	·884	1·148	1·459	1·822	2·240	2·720	3·872	5·312
Blasting amberite	1·102	·158	·250	·374	·532	·730	·972	1·262	1·604	2·004	2·465	3·000	4·256	5·840
Carbonite	1·120	·160	·254	·380	·541	·742	·988	1·280	1·630	2·036	2·505	3·040	4·328	5·936
Ardeer powder ..	1·160	·166	·264	·394	·561	·769	1·024	1·330	1·690	2·110	2·597	3·151	4·488	6·152
Cotton powder or tonite	1·500	·215	·341	·509	·725	·994	1·323	1·717	2·184	2·728	3·355	4·075	5·798	7·952
Blasting gelatine ..	1·550	·222	·352	·526	·749	1·027	1·367	1·775	2·256	2·819	3·467	4·211	5·991	8·218
Gelatine-dynamite	1·550	·222	·352	·526	·749	1·027	1·367	1·775	2·256	2·816	3·467	4·211	5·991	8·218
Gelignite	1·550	·222	·352	·526	·749	1·027	1·367	1·775	2·256	2·816	3·467	4·211	5·991	8·218
E.S.M. compressed gunpowder ..	1·646	·236	·374	·559	·796	1·092	1·453	1·886	2·398	2·995	3·685	4·472	6·368	8·736
Dynamite	1·600	·229	·363	·543	·773	1·060	1·412	1·833	2·329	2·910	3·579	4·347	6·184	8·480

Table III.

Depths of Boreholes for Shearing according to the formulæ

$$D = \frac{m}{2} + W \quad \text{and} \quad m = 12\,d.$$

Line of Resistance.	Diameter of Boreholes in inches.																					
	1		1⅛		1¼		1⅜		1½		1⅝		1¾		1⅞		2		2½		3	
feet.	ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.	ft.	in.
1·0	1	6	1	7																		
1·3	1	9	1	10	1	11	2	0														
1·6	2	0	2	1	2	2	2	3	2	3	2	4										
1·9	2	3	2	4	2	5	2	6	2	6	2	7	2	8	2	8						
2·0	2	6	2	7	2	8	2	9	2	9	2	10	2	11	2	11	3	0				
2·3	2	9	2	10	2	11	3	0	3	0	3	1	3	2	3	2	3	3				
2·6	3	0	3	1	3	2	3	3	3	3	3	4	3	5	3	5	3	6	3	9		
2·9	3	3	3	4	3	5	3	6	3	6	3	7	3	8	3	8	3	9	4	0		
3·0	3	6	3	7	3	8	3	9	3	9	3	10	3	11	3	11	4	0	4	3	4	6
3·3	3	9	3	10	3	11	4	0	4	0	4	1	4	2	4	2	4	3	4	6	4	9
3·6	4	0	4	1	4	2	4	3	4	3	4	4	4	5	4	5	4	6	4	9	5	0
3·9	..		4	4	4	5	4	6	4	6	4	7	4	8	4	8	4	9	5	0	5	3
4·0	..		4	7	4	8	4	9	4	9	4	10	4	11	4	11	5	0	5	3	5	6
4·3	..		..		4	11	5	0	5	0	5	1	5	2	5	2	5	3	5	6	5	9
4·6	..		..		..		5	3	5	3	5	4	5	5	5	5	5	6	5	9	6	0
4·9	..		..		..		5	6	5	6	5	7	5	8	5	8	5	9	6	0	6	3
5·0	..		..		..		..		5	9	5	10	5	11	5	11	6	0	6	3	6	6
5·3	..		..		..		..		6	0	6	1	6	2	6	2	6	3	6	6	6	9
5·6	..		..		..		..		..		6	4	6	5	6	5	6	6	6	9	7	0
5·9	..		..		..		..		..		6	7	6	8	6	8	6	9	7	0	7	3
6·0	..		..		..		..		..		..		6	11	6	11	7	0	7	3	7	6
6·3	..		..		..		..		..		..		..		7	2	7	3	7	6	7	9
6·6	..		..		..		..		..		..		..		7	5	7	6	7	9	8	0
6·9	..		..		..		..		..		..		..		..		7	9	8	0	8	3
7·0	..		..		..		..		..		..		..		..		8	0	8	3	8	6
7·3	..		..		..		..		..		..		..		..		..		8	6	8	9
7·6	..		..		..		..		..		..		..		..		..		8	9	9	0
7·9	..		..		..		..		..		..		..		..		..		9	0	9	3
8·0	..		..		..		..		..		..		..		..		..		9	3	9	6

TABLE IV.—LINES OF RESISTANCE for two, three and four shotholes of the same diameter, and having the same length of charge, situated a distance K apart from each other in line parallel to the free face, when the line of resistance for one such shothole is taken as unity, and the length of charge twelve times the diameter of the hole.

Distance K between holes. Inches.	Two Shotholes.				Three Shotholes.				Four Shotholes.			
	Diameter of Holes in inches.				Diameter of Holes in inches.				Diameter of Holes in inches.			
	1	1½	2	3	1	1½	2	3	1	1½	2	3
1	1·73	1·77	1·79	1·81	2·29	2·38	2·43	2·48	2·73	2·88	2·97	3·06
2	1·62	1·70	1·73	1·77	2·05	2·20	2·29	2·38	2·36	2·60	2·73	2·88
3	1·53	1·62	1·67	1·73	1·85	2·05	2·27	2·29	2·00	2·36	2·53	2·73
4	1·44	1·56	1·62	1·69	1·69	1·92	2·05	2·21	1·85	2·16	2·36	2·60
5	1·36	1·50	1·57	1·66	1·56	1·80	1·95	2·13	1·67	2·00	2·21	2·48
6	1·30	1·44	1·53	1·62	1·44	1·70	1·85	2·05	1·53	1·85	2·08	2·36
7	..	1·39	1·49	1·59	..	1·60	1·77	1·98	1·40	1·73	1·96	2·26
8	..	1·34	1·44	1·56	..	1·52	1·69	1·91	1·30	1·62	1·85	2·16
9	..	1·30	1·40	1·53	..	1·44	1·62	1·85	..	1·53	1·79	2·08
10	..	..	1·36	1·50	..	..	1·56	1·79	..	1·44	1·67	2·00
11	..	..	1·33	1·47	..	..	1·50	1·74	..	1·37	1·60	1·92
12	..	..	1·30	1·44	..	..	1·44	1·69	..	1·30	1·53	1·85
13	..	..	..	1·41	..	..	..	1·64	..	..	1·46	1·79
14	..	..	..	1·38	..	..	..	1·60	..	..	1·40	1·73
15	..	..	..	1·36	..	..	..	1·56	..	..	1·35	1·67
16	..	..	..	1·34	..	..	..	1·52	..	..	1·30	1·62
17	..	..	..	1·32	..	..	..	1·48	..	..	..	1·57
18	..	..	..	1·30	..	..	..	1·44	..	..	..	1·53

TABLE V.—RATIOS OF LINES OF RESISTANCE $\left(\frac{W_1}{W}\right)$ for a Shothole whose diameter is d_1, length of charge $12\,d_1$, and angle to Line of Resistance b, the Line of Resistance being taken as unity for a Shothole whose diameter is $d = 1$ in., length of charge $12\,d$, and angle to Line of Resistance 90 degrees, according to the formula $\frac{W_1}{W} = \frac{13\,d_1}{(12 + \operatorname{cosec} b)\,d}$.

Angle b of Shothole. Degrees.	Values of d_1.								
	1	$1\frac{1}{8}$	$1\frac{1}{4}$	$1\frac{3}{8}$	$1\frac{1}{2}$	$1\frac{5}{8}$	$1\frac{3}{4}$	$1\frac{7}{8}$	2
5	·55	·62	·69	·75	·82	·89	·96	1·03	1·10
6	·60	·67	·75	·82	·90	·97	1·05	1·12	1·20
7	·64	·72	·80	·88	·96	1·04	1·12	1·20	1·28
8	·67	·75	·83	·92	1·00	1·08	1·17	1·25	1·34
9	·70	·79	·88	·96	1·05	1·14	1·22	1·31	1·40
10	·73	·82	·91	1·00	1·09	1·18	1·27	1·36	1·46
11	·75	·84	·93	1·03	1·13	1·22	1·31	1·40	1·50
12	·77	·86	·96	1·06	1·16	1·25	1·35	1·44	1·54
13	·79	·89	·99	1·09	1·18	1·28	1·38	1·48	1·58
14	·80	·90	1·00	1·10	1·20	1·30	1·40	1·50	1·60
16	·83	·93	1·04	1·14	1·24	1·34	1·45	1·55	1·66
19	·86	·97	1·07	1·18	1·29	1·40	1·51	1·61	1·72
22	·89	1·00	1·11	1·22	1·33	1·44	1·56	1·67	1·78
26	·91	1·02	1·13	1·24	1·36	1·47	1·59	1·70	1·82
30	·93	1·05	1·16	1·28	1·39	1·51	1·63	1·74	1·86
35	·95	1·07	1·19	1·31	1·43	1·54	1·66	1·78	1·90
40	·96	1·08	1·20	1·32	1·44	1·56	1·68	1·80	1·92
45	·97	1·09	1·21	1·33	1·45	1·57	1·69	1·82	1·94

TABLE VI.

APPROXIMATE VOLUMES (V) IN CUBIC FEET OF ROCK, WHICH WILL BE BLASTED BY A CONCENTRATED CHARGE WHEN THERE ARE ONE, TWO, THREE OR FOUR FREE FACES (OPEN JOINTS MUST BE CONSIDERED AS FREE FACES).

Line of Resistance W in feet.	One Free Face, $V = 1\frac{1}{3} W^3$.	Two Free Faces, $V = 1\frac{2}{3} W^3$.	Three Free Faces, $V = 2\frac{1}{3} W^3$.	Four Free Faces, $V = 3 W^3$.	Five Free Faces, $V = 4 W^3$.
1	1·33	1·67	2·33	3·00	4·00
1¼	2·60	3·25	4·55	5·85	7·80
1½	4·50	5·62	7·87	10·12	13·50
1¾	7·15	8·94	12·51	16·08	21·44
2	10·66	13·36	18·66	24·00	32·00
2¼	15·18	18·97	26·57	34·17	45·56
2½	20·82	26·03	36·46	46·87	62·50
2¾	27·73	34·66	48·53	62·40	83·20
3	36·00	45·00	63·00	81·00	108·00
3¼	45·77	57·21	80·10	102·99	137·32
3½	57·17	71·46	100·04	128·62	171·50
3¾	70·31	87·89	123·04	158·19	210·92
4	85·33	106·67	149·33	192·00	256·00
4¼	102·35	127·93	179·10	230·28	307·04
4½	121·50	151·87	212·62	273·37	364·50
4¾	142·89	178·61	250·06	321·51	428·68
5	166·66	208·33	291·66	375·00	500·00
5¼	192·93	241·16	337·63	434·10	578·80
5½	221·83	277·29	388·21	499·12	665·50
5¾	253·48	316·85	443·59	570·33	760·44
6	288·00	360·00	504·00	648·00	864·00
6¼	325·52	406·90	569·66	732·42	976·56
6½	366·16	457·70	640·79	823·87	1098·50
6¾	410·06	512·57	717·61	922·65	1230·20
7	457·33	571·67	800·33	1029·00	1372·00
7¼	508·10	635·12	889·18	1143·24	1524·32
7½	562·50	703·12	984·37	1265·62	1687·50
7¾	620·64	775·80	1086·12	1396·44	1861·92
8	682·66	853·33	1194·66	1536·00	2048·00

N.B.—The faces are supposed to be at right angles to each other.

TABLE VII.

APPROXIMATE VOLUMES OF ROCK IN CUBIC FEET WHICH WILL BE BLASTED IN THE CASE OF STEPPED WORKINGS, TWO FREE FACES AND SINGLE BOREHOLES.

$$V = W^3 + \frac{m}{2} W^2. \qquad \frac{m}{2} = 6\,d.$$

Line of Resistance in feet.	Diameter d of Boreholes in inches.								
	1	1¼	1½	1¾	2	2¼	2½	2¾	3
1	1·50	1·62							
1½	4·50	4·78							
2	10·00	10·50	11·00						
2½	18·75	19·53	20·31						
3	31·50	32·63	33·75	34·88					
3½	49·00	50·53	52·06	53·59					
4	72·00	73·90	76·00	78·00	80·00				
4½	..	103·78	106·31	108·84	111·38				
5	..	140·62	143·75	146·88	150·00	153·12			
5½	..	..	189·06	192·84	196·63	200·41			
6	..	..	243·00	247·50	252·00	256·50	261·00		
6½	..	..	..	311·59	316·88	322·16	327·44		
7	..	..	..	385·88	392·00	398·13	404·25	410·37	
7½	..	..	..	..	478·13	485·16	492·19	499·22	
8	..	..	..	..	576·00	584·00	592·00	600·00	608·00
9	..	..	..	..	..	820·13	830·25	840·38	850·50
10	..	..	..	..	..	1112·50	1125·00	1137·50	1150·00
11	..	..	..	..	..	..	1482·25	1497·38	1512·50
12	..	..	..	..	..	..	1908·00	1926·00	1944·00
13	..	..	..	..	..	..	..	2429·38	2450·00
14	..	..	..	..	..	..	..	3013·50	3038·00
15	..	..	..	..	..	..	..	..	3712·05
16	..	..	..	..	..	..	..	..	4480·00

TABLE VIII.

APPROXIMATE VOLUMES OF ROCK IN CUBIC FEET WHICH WILL BE BLASTED IN THE CASE OF STEPPED WORKINGS, THREE FREE FACES AND SINGLE SHOTHOLES.

$$V = \tfrac{3}{2} W^3 + \tfrac{3}{4} m W^2. \qquad \tfrac{3}{4} m = 9 d.$$

Line of Resistance in feet.	Diameter of Boreholes in inches.								
	1	1¼	1½	1¾	2	2¼	2½	2¾	3
1	2·25	2·44	2·63	2·81					
1½	6·75	7·17	7·59	8·02	8·44				
2	15·00	15·75	16·50	17·25	18·00				
2½	28·12	29·30	30·47	31·64	32·81	33·98			
3	47·25	48·94	50·63	52·32	54·00	55·69			
3½	73·50	75·80	78·09	80·40	82·69	84·98	87·28		
4	108·00	111·00	114·00	117·00	120·00	123·00	126·00		
4½	151·88	155·67	159·47	136·27	167·06	170·86	174·66	178·46	
5	..	210·94	215·63	220·33	225·00	229·69	234·38	239·08	
5½	..	277·92	283·59	289·28	294·94	300·61	306·27	311·94	317·61
6	..	..	364·50	371·27	378·00	384·75	391·50	398·25	405·00
6½	..	..	459·47	467·41	475·31	483·23	491·15	499·08	507·00
7	..	..	..	578·84	588·00	597·19	606·38	615·56	624·75
7½	..	..	..	706·67	717·19	727·73	738·28	748·83	759·38
8	..	..	..	..	864·00	876·00	888·00	900·00	912·00
8½	..	..	..	..	1029·56	1043·11	1056·66	1070·20	1083·75
9	..	..	..	..	..	1230·19	1245·38	1260·56	1275·75
9½	..	..	..	..	..	1438·37	1455·28	1472·20	1489·11
10	..	..	..	..	..	..	1687·50	1706·25	1725·00

Table IX.

Approximate Volumes of Rock in Cubic Feet which will be blasted in the case of Stepped Workings, Four Free Faces and Single Shotholes.

$$V = 2\,W^3 + m\,W^2. \qquad m = 12\,d.$$

Line of Resistance in feet.	Diameter of Boreholes in inches.								
	1	1¼	1½	1¾	2	2¼	2½	2¾	3
1	3·00	3·25	3·50	3·75	4·00				
1½	9·00	9·56	10·13	10·69	11·25	11·91			
2	20·00	21·00	22·00	23·00	24·00	25·00			
2½	37·50	39·06	40·63	42·19	43·75	45·31	46·87		
3	63·00	65·25	67·50	69·75	72·00	74·25	76·50		
3½	98·00	101·06	104·13	107·19	110·25	113·31	116·38	119·44	
4	144·00	148·00	152·00	156·00	160·00	164·00	168·00	172·00	
4½	202·50	207·56	212·62	217·68	222·75	227·82	232·88	237·94	243·00
5	..	281·25	287·50	293·75	300·00	306·25	312·50	318·75	325·00
5½	..	370·56	378·13	385·69	393·25	400·81	408·38	415·94	423·54
6	..	..	486·00	495·00	504·00	513·00	522·00	531·00	540·00
6½	..	..	612·73	623·29	633·85	644·41	654·97	665·54	676·10
7	..	..	..	771·75	784·00	796·25	808·50	820·75	833·00
7½	..	..	..	..	956·25	970·31	984·38	998·44	1012·50
8	..	..	..	..	1152·00	1168·00	1184·00	1200·00	1216·00
8½	..	..	..	..	..	1390·81	1408·88	1426·94	1445·00
9	..	..	..	..	..	1640·25	1660·50	1680·75	1701·00
9½	..	..	..	..	..	..	1940·38	1962·94	1985·50
10	..	..	..	..	..	..	2250·00	2275·00	2300·00

TABLE X.

APPROXIMATE VOLUMES OF ROCK IN CUBIC FEET WHICH WILL BE BLASTED IN THE CASE OF STEPPED WORKINGS, TWO FREE FACES AND SIMILAR SHOTHOLES PLACED A DISTANCE 2 W APART, AND FIRED SIMULTANEOUSLY.

Two shotholes, $V = 3\,W^3 + \frac{3}{2}\,m\,W^2$. $\frac{3}{2}\,m = 18\,d$.

Line of Resistance in feet.	Diameter of Boreholes in inches.								
	1	$1\frac{1}{4}$	$1\frac{1}{2}$	$1\frac{3}{4}$	2	$2\frac{1}{4}$	$2\frac{1}{2}$	$2\frac{3}{4}$	3
1	4·50	4·88	5·25						
$1\frac{1}{2}$	13·50	14·34	15·19	16·03					
2	30·00	31·50	33·00	34·50	36·00				
$2\frac{1}{2}$	56·25	58·59	60·95	63·29	65·63				
3	94·5	97·88	101·25	104·63	108·00	111·38	114·75		
$3\frac{1}{2}$	147·00	151·59	156·19	160·78	165·38	169·97	174·56	179·16	
4	216·00	222·00	228·00	234·00	240·00	246·00	252·00	258·00	264·00
$4\frac{1}{2}$	303·75	311·34	318·94	326·53	334·13	341·72	349·31	356·91	364·50
5	..	421·88	431·25	440·63	450·00	459·38	468·75	478·13	487·50
$5\frac{1}{2}$	..	555·84	567·19	578·53	589·88	601·22	612·56	623·91	635·25
6	..	..	729·00	742·50	756·00	769·50	783·00	796·50	810·00
$6\frac{1}{2}$	..	..	..	934·78	950·63	966·47	982·31	998·16	1014·00
7	..	..	..	..	1176·00	1194·38	1212·75	1231·13	1249·50
8	..	..	..	..	1728·00	1752·00	1776·00	1800·00	1824·00

Three shotholes, $V = 5\,W^3 + \frac{5}{2}\,m\,W^2$. $\frac{5}{2}\,m = 30\,d$.

Line of Resistance in feet.	1	$1\frac{1}{4}$	$1\frac{1}{2}$	$1\frac{3}{4}$	2	$2\frac{1}{4}$	$2\frac{1}{2}$	$2\frac{3}{4}$	3
1	7·5	8·13	8·75						
$1\frac{1}{2}$	22·5	23·91	25·31	26·72					
2	50·00	52·50	55·00	57·50	60·00				
$2\frac{1}{2}$	93·75	97·67	101·56	105·47	109·38	113·49			
3	157·5	163·13	168·75	174·38	180·00	185·63	191·25		
$3\frac{1}{2}$	245·00	252·66	260·31	267·97	275·62	283·28	290·94	298·59	
4	360·00	370·00	380·00	590·00	400·00	410·00	420·00	430·00	440·00
$4\frac{1}{2}$	..	518·91	531·56	544·25	556·87	569·53	582·19	594·84	607·50
5	..	703·13	718·75	734·38	750·00	765·63	781·25	796·88	812·50
$5\frac{1}{2}$	..	..	945·31	764·22	983·12	1002·03	1020·94	1039·84	1058·75
6	..	..	..	1237·50	1260·00	1282·50	1305·00	1327·50	1350·00
$6\frac{1}{2}$	..	..	..	1557·97	1584·38	1610·78	1637·19	1663·60	1690·00
7	..	..	..	..	1960·00	1990·63	2021·25	2051·88	2082·50

TABLE XI.—SECTIONS OF ROCK IN SQUARE FEET WHICH WILL BE RUPTURED BY CHARGES WHOSE LENGTH = 12 d, ACCORDING TO THE FORMULA

$$F = \frac{\cdot 032}{C_a} \times 7\cdot 38\, d^2.$$

Diameter d of Shot-holes. Inches.	Coefficient of Rock C^a.														
	·032	·03	·028	026	·024	·022	·02	·018	·016	·014	·012	·01	·008	·006	·004
$\frac{3}{4}$	4·15	4·43	4·74	5·11	5·53	6·04	6·64	7·38	8·30	9·49	12·03	13·28	16·60	22·14	33·21
$\frac{7}{8}$	5·65	6·03	6·46	6·96	7·53	8·22	9·04	10·05	11·30	12·92	14·47	18·08	22·60	30·13	45·20
1	7·38	7·87	8·44	9·08	9·84	10·74	11·81	13·12	14·76	16·87	19·68	23·62	29·52	39·36	59·04
$1\frac{1}{8}$	9·35	9·60	10·67	11·49	12·44	13·58	14·94	16·60	18·67	21·34	24·90	29·87	37·34	49·78	74·68
$1\frac{1}{4}$	11·53	12·30	13·18	14·19	15·37	16·78	18·45	20·50	23·06	26·36	30·74	36·90	46·13	61·48	92·25
$1\frac{3}{8}$	13·95	14·89	15·95	17·17	18·60	20·30	22·32	24·81	27·90	31·90	37·20	44·64	55·81	74·40	111·62
$1\frac{1}{2}$	16·61	17·72	18·98	20·44	22·13	24·16	26·57	29·52	33·21	37·96	44·26	53·14	66·42	88·52	132·84
$1\frac{5}{8}$	19·49	20·79	22·27	23·99	25·98	28·35	31·18	34·65	37·97	44·54	51·96	62·36	77·95	103·92	155·90
$1\frac{3}{4}$	22·60	24·12	25·83	27·82	30·13	32·88	36·16	40·10	45·20	51·66	60·26	72·32	90·40	120·52	180·81
$1\frac{7}{8}$	25·95	27·67	29·66	31·94	34·58	37·75	41·51	46·13	51·90	59·32	69·16	83·02	103·80	138·32	207·60
2	29·52	31·50	33·74	36·34	39·35	42·95	47·23	52·49	59·04	67·48	78·70	94·46	118·08	157·40	236·16
$2\frac{1}{8}$	33·33	35·56	38·09	41·02	44·42	48·49	53·32	59·25	66·66	76·18	88·84	106·64	133·32	177·68	266·64
$2\frac{1}{4}$	37·36	39·66	42·70	45·99	49·81	54·36	59·78	66·41	74·72	85·40	99·62	119·56	149·44	199·24	298·88
$2\frac{3}{8}$	41·63	44·41	47·58	51·25	55·51	60·57	66·61	74·01	83·26	95·16	111·02	133·22	166·52	222·04	333·04
$2\frac{1}{2}$	46·13	49·05	52·79	56·78	61·51	67·11	73·80	83·01	92·26	105·58	123·02	147·60	184·52	246·04	369·54
$2\frac{5}{8}$	50·85	54·24	58·12	62·60	67·80	73·99	81·36	90·40	101·70	116·24	135·60	162·72	203·40	271·26	406·80
$2\frac{3}{4}$	55·81	59·52	63·78	68·80	74·40	81·28	89·30	99·20	111·62	127·56	148·80	178·60	223·24	297·60	446·48
$2\frac{7}{8}$	61·00	65·08	69·72	75·09	81·33	88·76	97·60	108·44	122·00	139·44	162·66	195·20	244·00	325·32	488·00
3	66·42	70·87	75·91	81·76	88·56	96·64	106·27	118·08	132·84	151·82	177·12	212·53	265·68	354·24	531·32

Table XII.—Capacity of One Foot of Borehole in Cubic Inches.

Diam of Bore-hole in Inches.	Cubic Inches per Foot.	Diam. of Bore-hole in Inches.	Cubic Inches per Foot.	Diam. of Bore-hole in Inches.	Cubic Inches per Foot.	Diam. of Bore-hole in Inches.	Cubic Inches per Foot.
3/4	5·3	1 7/16	19·45	2 1/4	47·71	3 5/8	123·84
13/16	6·22	1 1/2	21·20	2 3/8	53·16	3 3/4	132·50
7/8	7·22	1 9/16	23·00	2 1/2	58·90	3 7/8	141·50
15/16	8·28	1 5/8	24·88	2 5/8	64·93	4	150·90
1	9·42	1 11/16	26·83	2 3/4	71·26	4 1/4	170·20
1 1/16	10·63	1 3/4	28·86	2 7/8	77·90	4 1/2	190·80
1 1/8	11·93	1 13/16	30·95	3	84·83	4 3/4	212·65
1 3/16	13·28	1 7/8	33·13	3 1/8	92·10	5	235·70
1 1/4	14·72	1 15/16	35·36	3 1/4	99·54	5 1/4	259·70
1 5/16	16·22	2	37·70	3 3/8	107·35	5 1/2	285·00
1 3/8	17·80	2 1/8	42·55	3 1/2	115·45	6	339·25

Table XIII.—Weight of a Lineal Foot of Round, Octagonal and Square drilling Steel.

Note.—The diameter of octagon steel is measured across the sides.

Diam. of Round and Octagon and Side of Square in Inches.	Round. lbs.	Octa-gonal. lbs.	Square. lbs.	Diam. of Round and Octagon and Side of Square in Inches.	Round. lbs.	Octa-gonal. lbs.	Square. lbs.
3/16	·094	·099	·120	1	2·673	2·819	3·403
1/4	·167	·176	·213	1 1/8	3·382	3·568	4·307
5/16	·261	·275	·332	1 1/4	4·176	4·405	5·317
3/8	·376	·396	·479	1 3/8	5·053	5·330	6·433
7/16	·512	·540	·651	1 1/2	6·013	6·343	7·656
1/2	·668	·705	·851	1 5/8	7·057	7·444	8·985
9/16	·846	·892	1·077	1 3/4	8·185	8·633	10·421
5/8	1·044	1·101	1·329	1 7/8	9·396	9·910	11·963
11/16	1·263	1·332	1·608	2	10·690	11·276	13·611
3/4	1·503	1·586	1·914	2 1/4	13·530	14·271	17·227
13/16	1·764	1·861	2·246	2 1/2	16·703	17·618	21·267
7/8	2·046	2·158	2·605	2 3/4	20·211	21·318	25·734
15/16	2·349	2·478	2·991	3	24·053	25·371	30·625

TABLE XIV.—USEFUL HYDRAULIC DATA.

One cubic inch of water . .	=	·0361 lb.
		·00361 gallon.
		·000016 cubic metre.
		·0164 litre.
		·0164 kilogramme.
One cubic foot of water . .	=	62·355 lbs. = ·557 cwt. = ·028 ton.
		6·236 gallons, or say 6¼ gallons.
		·0283 cubic metre.
		28·3 litres.
		28·3 kilogrammes.
One gallon of water . . .	=	10 lbs. of fresh water, or 10·272 lbs. of salt water.
		·16 cubic feet, or 277 cubic inches.
		4·54 litres.
		4·54 kilogrammes.
		·0454 cubic metre.
One ton of water	=	36 cubic feet of fresh water, or 35 cubic feet of salt water.
		224 gallons of fresh water, or 218 gallons of salt water.
		1016 kilogrammes.
		1016 litres.
		1·0165 cubic metre.
One lb. of water	=	·016 cubic foot = 27·72 cubic inches.
		·10 gallon.
		·4536 kilogramme.
		·4536 litre.
		·0004536 cubic metre.
One cubic metre of water . .	=	1000 litres.
		1000 kilogrammes.
		35·317 cubic feet = 61028 cubic inches.
		220 gallons.
One cubic centimetre of water	=	·001 litre.
		·001 kilogramme.
		·061 cubic inch.
		·00022 gallon.
		·0022 lb.
One litre of water	=	1 kilogramme.
		·001 cubic metre.
		·22 gallon.
		2·2046 lbs.
		·0353 cubic feet, or 61 cubic inches.

TABLE XV.—WEIGHT OF STONE AND MINERAL SUBSTANCES.

Description.	Specific Gravity.	Weight of 1 Cubic Foot.	Weight of 1 Cubic Yard.	Number of Cubic Feet in 1 Ton.
		lbs.	lbs.	c. f.
Basalt, Scotch . .	2·95	184	4970	12
Chalk . . .	2·33	145	3900	15
„ . . .	2·62	162	4370	13¾
Granite, Aberdeen grey	2·62	163	4400	13¾
„ „ red	2·62	165	4450	13½
„ Cornish .	2·66	166	4480	13½
Limestone, compact .	2·58	161	4340	13¾
Marble, Egyptian green	2·67	167	4500	13½
„ Carrara .	2·72	170	4590	13¼
Oolite, Portland stone	2·42	151	4070	15
„ Bath stone .	1·98	123	3320	18¼
Sandstone . .	2·51	157	4240	14¼
Slate, Cornwall . .	2·51	157	4240	14¼
„ Welsh . .	2·88	180	4860	12½
Trap . . .	2·72	170	4590	13¼
Quartz . . .	2·75	171	4620	13
Coal, bituminous .	1·29	80	2160	28
„ anthracite. .	1·60	100	2700	22⅖
Earth, from . .	1·52	77	2080	29
„ to . .	2·	125	3375	18
Mortar, average .	1·70	106	2860	21⅛
Mud . . .	1·70	105	2830	21⅓
Felspar . . .	2·62	165	4450	13½

NOTE.—Solid rock increases about one-fifth in bulk, and decreases correspondingly in weight when broken and loaded.

TABLE XVI.—COMPARISON OF IMPERIAL AND METRIC SYSTEMS.

Linear Measure.

—	Mm.	Cm.	Metres.	—	Inches.	Feet.	Yards.
1 inch . .	25·4	2·54	·0254	1 mm. .	·03937		
1 foot . .	304·8	30·48	·3048	1 cm. . .	·3937	·0328	·0109
1 yard . .	914·4	91·44	·9144	1 metre .	39·3704	3·281	1·093
1 mile . .	..	..	1609·3	1 kilometre	..	3281·	1093·6

Square Measure.

—	Square Mm.	Square Cm.	Square Metre.	—	Square Inches.	Square Feet.	Square Yards.
1 square inch	645·1	6·451	·000645	1 square mm.	·00155		
1 „ foot	92899·	929·	·0929	1 „ cm.	·155	·001076	
1 „ yard	83609·7	8360·97	·8361	1 „ metre	1550·03	10·7641	1·196

Solid Measure.

—	Cubic Mm.	Cubic Cm.	Cubic Metre.	—	Cubic Inches.	Cubic Feet.	Cubic Yds.
1 cubic inch	16387	16·387	·000016	1 cubic mm.	·000061		
1 ,, foot	28315300	28315·3	·0283153	1 ,, cm.	·06102524	·0000353156	
1 ,, yard	764513000	764513·	·764513	1 ,, metre	61025·24	35·3156	1·30802

Liquid Measure.

—	Litres.	—
1 pint . . .	·56793	1 litre = { 1·76 pint.
1 quart . . .	1·1359	·88 quart.
1 gallon . .	4·5434	·22 gallon.

Weights.

—	Grammes.	Kilogrammes.	—	Ounces.	Lbs.	Cwt.	Ton.
1 ounce . .	28·349	·02835	1 gramme .	·035	·0022		
1 lb. . .	453·59	·4536	1 kilogramme.	35·27	2·2046	·01968	·000984
1 cwt. . .	50802·	50·802					
1 ton . .	1016047·5	1016·047					

Table XVII.

Pressure of Atmospheres in Lbs. per Square Inch.

Atmospheres	= lbs. per sq. in.	Atmospheres	= lbs. per sq. in.	Atmospheres	= lbs. per sq. in.	Atmospheres	= lbs. per sq. in.
$\frac{1}{2}$	7·35	3	44·1	$5\frac{1}{2}$	80·85	8	117·6
1	14·7	$3\frac{1}{2}$	51·45	6	88·2	$8\frac{1}{2}$	124·95
$1\frac{1}{2}$	22·05	4	58·8	$6\frac{1}{2}$	95·55	9	132·3
2	29·4	$4\frac{1}{2}$	66·15	7	102·9	$9\frac{1}{2}$	139·65
$2\frac{1}{2}$	36·75	5	73·5	$7\frac{1}{2}$	110·25	10	147·0

One cubic foot of air, at 14·7 lbs. per square inch, or one atmosphere, weighs ·080728 lb. at 32° F.

One lb. of air, at 14·7 lbs. per square inch, 62° F., occupies 13·141 cubic feet.

Table XVIII.

Decimal Equivalents of an Inch.

$\frac{1}{32}$ = ·03125	$\frac{9}{32}$ = ·28125	$\frac{17}{32}$ = ·53125	$\frac{25}{32}$ = ·78125
$\frac{1}{16}$ = ·0625	$\frac{5}{16}$ = ·3125	$\frac{9}{16}$ = ·5625	$\frac{13}{16}$ = ·8125
$\frac{3}{32}$ = ·09375	$\frac{11}{32}$ = ·34375	$\frac{19}{32}$ = ·59375	$\frac{27}{32}$ = ·84375
$\frac{1}{8}$ = ·125	$\frac{3}{8}$ = ·375	$\frac{5}{8}$ = ·625	$\frac{7}{8}$ = ·875
$\frac{5}{32}$ = ·15625	$\frac{13}{32}$ = ·40625	$\frac{21}{32}$ = ·65625	$\frac{29}{32}$ = ·90625
$\frac{3}{16}$ = ·1875	$\frac{7}{16}$ = ·4375	$\frac{11}{16}$ = ·6875	$\frac{15}{16}$ = ·9375
$\frac{7}{32}$ = ·21875	$\frac{15}{32}$ = ·46875	$\frac{23}{32}$ = ·71875	$\frac{31}{32}$ = ·96875
$\frac{1}{4}$ = ·25	$\frac{1}{2}$ = ·5	$\frac{3}{4}$ = ·75	1 = 1·0

TABLE XIX.

PROPERTIES OF THE CIRCLE.

Circumference . .	= diameter × 3·1416, or, × $3\frac{1}{7}$.
Diameter . . .	= circumference × ·31831.
Diameter × ·8862 .	= side of an equal square.
Diameter × ·7071 .	= side of an inscribed square.
Diameter . . .	= 1·1283 $\sqrt{\text{area of circle}}$.
Area	= diameter squared × ·7854.
Diameter . . .	= $\sqrt{\text{area} \times 1{\cdot}1283}$.
Length of arc . .	= number of degrees × ·017453 radius.
Sphere, solidity of .	= diameter^3 × ·5236.

ADDENDA

(The Articles in this Addenda are numbered the same as those in the body of the Work, to which the subject matter pertains.)

6. *Crater Forms.*—From experiments principally made in earth, the following sectional forms of craters are given by different writers (Prof. H. Hōfer on Blasting and Military Mining).

(1) *Megrigny* in the year 1686. The form of a trapezium, which view was afterwards adopted by Proudhomme and Lebrun. In later times this form was abandoned.

(2) *Vauban* (1704). A right cone, in which the two opposite exterior lines enclose a right angle, the vertex of the cone being in the chamber. At first Vauban's cone found few adherents, but more recently this form has been almost universally adopted by writers on military mines.

(3) *Belidor* (about 1730). A frustrum of a cone, to the smaller base of which is joined a hemisphere (or more correctly a calotte, which nearly equals a hemisphere).

(4) *Valiere* believed himself justified from excavations conducted in earth mines, in assuming the section of the crater to be a parabola whose focus coincides with the centre of the chamber.

(5) *John Muller* (1757) combined the forms of Megrigny and Valiere.

(6) The Swede *Meldecreuz* (1749) contended that the crater is bounded below by a catenary and at the sides by trajectories.

(7) *E. Rziha* (1866) was the first to point out that we are able to establish, after excavations in an earth mine, almost any form of crater which does not offend common sense. He preferred, therefore, to study the craters in rocks of various solidity, and taking for the crater form only the figure left by the ejected mass, adopted the form given by Vauban. In the softer rocks he assumed a sphere of compression to be formed

around the explosion of the charge, and for such rocks the crater form to consist of a compound figure, consisting of a cone whose vertex is surrounded by a sphere. Hence, by adding the sphere of compression to the crater he obtained a bell-shaped form in compressible material, and a right cone in incompressible masses (as any solid rock), where no sphere of compression exists. In the sandstone formation of Saxo-Bohemian Switzerland, he observed the bell-shaped crater form at every blast in several hundred cases, while in the compact granite of the neighbouring mountain chain, extending from Meissen into the northern part of Bohemia, each discharge produced a right cone.

For the solid rocks our observations are quite in accordance with E. Rziha, viz. that they exhibit no sphere of compression. Further, we are satisfied from our observations that there is very little compression even in rocks having little cohesion, such as sandstone. The amount of disintegration which occurs around the chamber of a blast is dependent on the friability of the rock, and the heat and pressure caused by the explosion.

9. *Prof. Höfer's Theory of Blasting and Military Mining.*—This theory is based on the assumption that the normal crater produced by a blast is a right cone, and that the waves of concussion occur in rocks practically as in perfectly elastic media, and the following conclusions are arrived at:—

(1) That the cone of projection, or mass projected by an explosion in a chamber O (Fig. 1), with the same charge and in the same rock will exhibit very different dimensions, according to the line of least resistance—that is the shortest, or perpendicular distance from the chamber to the free surface. If a certain depth, or rather length, of this line is exceeded, the circular base lying in the free surface diminishes with the increase of the line forming the height of the cone of projection, and for a certain line of least resistance becomes zero. When this occurs, the throwing effect of the explosion against the free surface ceases—the charge then rending the rock without displacing the parts.

(2) That the shock of the explosion at O is transmitted in concentric spherical layers having O for a common centre. As this force is distributed over gradually increasing spherical surfaces, it decreases per surface unit as the spherical surface

increases. These are therefore, inversely proportional. The force p_1, acting upon a surface unit of a spherical shell $o\,b_1$, is to the force p, acting on the surface unit of the spherical shell $o\,b$ inversely as the surface, or

$$p_1 : p = o\,b : o\,b_1$$

If $o\,b_1$ has the radius R_1, and $o\,b$ the radius R, then

$$p_1 : p = 4\,\pi\,R^2 : 4\,\pi\,R_1{}^2 = R^2 : R_1{}^2$$

or

$$\frac{p_1}{p} = \frac{R^2}{R_1{}^2}$$

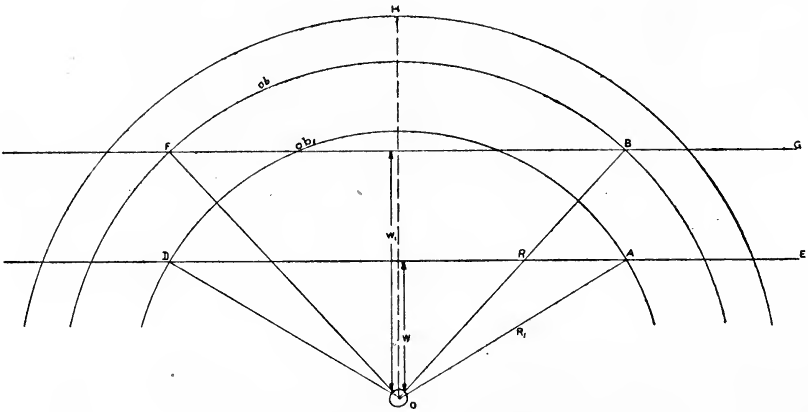

FIG. 1.

Hence, the intensities of the forces acting upon the surface units are to each other inversely as the squares of the corresponding radii (distances from the chamber).

(3) A normal charge L would in a given medium produce a cone of projection whose actual section A O B (Fig. 2), is a right angle. At A and B the vertical component of the radial force of the shock p would be just sufficient to displace one particle. In the same rock, using the same explosive, but a larger charge L_1, which is also a normal charge and produces a normal cone, L_1 will be m times as large as L or $\frac{L_1}{L} = m$.

When the impulse in the chamber is m times as great, a force m times as great as the former one (p), will act at the point B of the cone D O F; hence the radial force of shock at

B and at all other elements $o\,b$ of the spherical surface will be $m\,p$.

In order to obtain a normal cone, Prof. Höfer says we must increase the line of least resistance, which formerly was G O = W, to E O = W_1. A normal cone F D O then results, having its base-angle (a) equal to that of the former. The same force p will therefore act at F as at the first smaller cone of projection, so that its vertical component at F will be equal to the former. As at F, so at every corresponding element of the

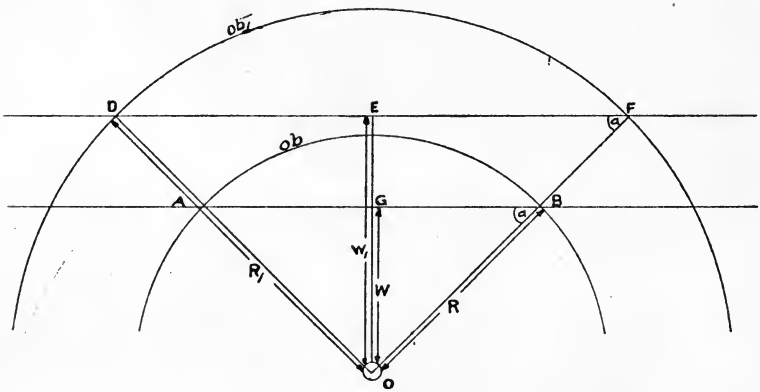

FIG. 2.

spherical envelope $o\,b_1$, the radial force of shock is equal to p.

The sums of all the forces of the charge L_1, acting on the spherical surfaces $o\,b$ and $o\,b_1$, must be equal; hence

$$4\,\pi\,R^2\,m\,p = 4\,\pi\,R_1^2\,p$$

or

$$R^2\,m = R_1^2$$

Since the cone sides R = O B and R_1 = O F belong to two similar triangles B G O and F E O, it follows that

$$R : R_1 = W : W_1$$

so that we may place in the above equation

$$W^2\,m = W_1^2$$

$$W_1 = W\sqrt{m}$$

That is: The normal lines of least resistance are to each other as the square roots of the charges.

Prof. Höfer, therefore, concludes that the normal cones of projection increase in a greater degree than the corresponding charges, and he says, if the charge in the second case was four times as large as in the first, a rock mass eight times as large would be projected in the cone, since $4\sqrt{4} = 8$.

As proof of the truth of these deductions from his premises, Prof. Höfer gives a few tests in special soils.

In the Appendix we give a special article to make clear the wrong premises on which this theory is based in regard to rock, from which it will be seen that it cannot be applied to rock blasting. Attention should, we think, be called to its fallacies, in view of the fact that a well-known mining journal states that the results and experiments are the best available at the present day.

12. *Combined Shearing and Flexure Resulting from the Application of a Force in a Chamber in Rock.*—Given sufficient free surface, E F (Fig. 3) the two opposite exterior lines E *m* and F *n* of the rock

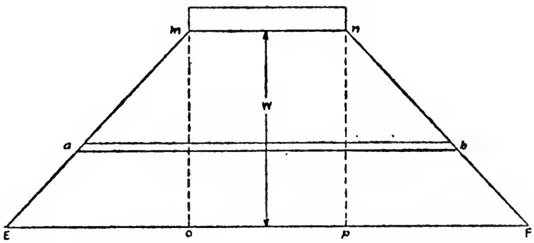

FIG. 3.

ruptured by the application of the pressure of a blast in the chamber *m n*, will enclose a right angle. The resulting pressure of the blast tends to shear the rock along the lines *m o* and *n p* perpendicular to the surface of the chamber, the shear zone *m o p n* therefore yields, and causes a bending or flexure of the surrounding rock E *m o* and F *n p*. Such flexure will be nil close to the chamber and increase proportionally to the distance of any layer *a b* from the chamber before the rupture occurs. Let us suppose the chamber to be circular, then the mass of rock E *m n* F E, which would be ruptured by sufficient pressure applied equally over the surface *m n*, is the frustrum of a cone. We may consider this frustrum to be composed of an infinite number N of circular layers $a\,b$ resting on one another, and that when bent they neither lose their parallelism nor slide upon one another. Then, according to the well-known theory of flexure for materials, those layers which are on the convex side are extended, and those on the concave side compressed, while a certain mean layer undergoes neither extension nor compression. The extension of the layers on one side and the compression of those on the other increase gradually, so that those most distant from the free surface undergo the maximum extension, and those nearest the free surface the maximum compression. (Fig. 4.)

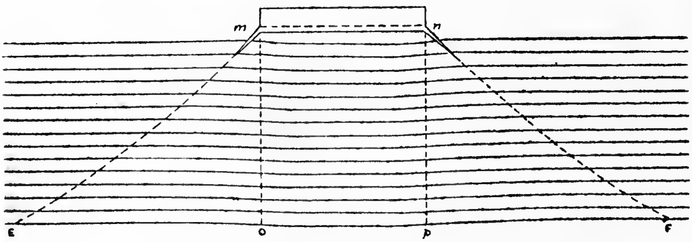

FIG. 4.

In consequence there is very little movement along the line of least resistance $m\,o = n\,p$ before the limit of ultimate strength is reached by the layer nearest the chamber, which layer therefore ruptures first so that opposite lines of fracture are started from m and n in the direction of m E and n F, which for all practical purposes enclose a right angle, the continued movement of the central part $m\,o\,p\,n$ successively bending each layer to the limit required to complete the rupture of the mass E $m\,n$ F. We, therefore, distinguish between the central portion $o\,m\,n\,p$ and that surrounding it by calling the former the shear-zone and the latter the flex-zone of the mass ruptured. For any sectional element of a layer $a\,b$ (Fig. 3) whose radius is r, depth d and breadth b, assuming the force p producing flexure to be applied at the centre, the moment of rupture by flexure is :—

$$p\,r = c\,b\,d^2$$

r being the arm of the force, c a coefficient and $b\,d^2$ the moment of flexure.

Substituting for b the circumference $2\,\pi\,r$ corresponding to the radius r we get the moment of rupture of the layer $a\,b$, namely :—

$$p\,r = 2\,c\,\pi\,r\,d^2$$

Therefore

$$p = 2\,c\,\pi\,d^2$$

That is, the force required to produce rupture by flexure is independent of the radii of the layers, and therefore N layers of the depth d will offer the same resistance to rupture as one of the thickness Nd, and the total force required to rupture N layers is

$$P = 2\,c\,\pi\,(N\,d)^2$$

But

$$N\,d = W$$

and consequently

$$P = 2\,c\,\pi\,W^2$$

or for any other line of resistance W_1

$$P : P_1 = 2\,c\,\pi\,W^2 : 2\,c\,\pi\,W_1^2 = W^2 : W_1^2$$

and

$$\frac{P_1}{P} = \left(\frac{W_1}{W}\right)^2$$

As the forces are proportional to the resistances, this ratio is the same as found for shearing (Art. 25).

In rock blasting, therefore, when the free surface is a plane and the force of the blast is insufficient to shear the lines of resistance $E\,m$ and $F\,n$, but equal to the line of resistance $W = m\,o = n\,p$, rupture results from both shearing and flexure, but the controlling factor is always the ratio of the shear-zone to the total pressure thereon tending to shear it.

In this connection it is important to note that the total pressures P and P_1 in the chambers acting to produce rupture depend on the sectional areas of the chambers at right angles to the axes of the shear-zones, whilst the resistance of each shear-zone is proportional to the product of the periphery of the sectional area of the chamber and the length of the line of least resistance, or $S \times W$. (Art. 37.)

25. *The Forces in Blasting are Proportional to the squares of the Lines of Least Resistance.*—Let P_e be the maximum pressure developed by the explosive in a spherical chamber whose radius is r and $A = r^2\,\pi$, the area of the section of the chamber at right angles to the line of least resistance W, then the resulting total force is

$$P = P_e\,A = P_e\,\pi\,r^2$$

The resistance, since the periphery of the chamber is $S = 2\pi r$, is :—

$$R = 2\pi r W K$$

K being the modulus of shearing.

Then making $P = R$, namely the force equal to the resistance, we have :—

$$2\pi r W K = P_e \pi r^2$$

or

$$W = \frac{r P_e}{2 K}$$

If the radius of the chamber is r_1, and the line of resistance W_1

$$W_1 = \frac{r_1 P_e}{2 K}$$

and

$$W_1 : W = \frac{r_1 P_e}{2 K} : \frac{r P_e}{2 K} = r_1 : r$$

or

$$\frac{W_1}{W} = \frac{r_1}{r} = \frac{d_1}{d}$$

d and d_1 being the diameters of the chambers.

Hence the lines of least resistance are to each other as the corresponding radii (or diameters) of the spherical chambers.

On the other hand, since

$$P = R$$

and

$$P_1 = R_1$$

we have

$$P = 2\pi r W K$$

and

$$P_1 = 2\pi r W_1 K$$

or

$$\frac{P_1}{P} = \frac{2\pi r_1 W_1 K}{2\pi r W K} = \frac{r_1 W_1}{r W}$$

and therefore as

$$\frac{r_1}{r} = \frac{W_1}{W}$$

it follows that

$$\frac{P_1}{P} = \left(\frac{W_1}{W}\right)^2$$

It is easily proved that the same relations exist for cylindrical or borehole charges, and therefore the resultant total forces (not the forces per surface-units (Article 9, Addenda) as stated by Prof. Höfer), acting at right angles to the free face, of properly proportioned charges for the blasting of rock are to each other as the squares of the corresponding lines of least resistance.

Let us now consider the transmission of the force per surface-unit through the rock. The pressure on the walls of the chamber neglecting the thermal conductivity of the rock and conditionally that the charge fills the chamber may be taken as constant and equal to the maximum pressure developed by the explosive. In a shear-zone it is evident that each unit of its length offers the same resistance to shearing and therefore the pressure transmitted decreases along the same directly as the distance from the chamber. If then P_w represents the total pressure on a shear-zone in the chamber for the whole line of resistance W and P_{w1}, is the total pressure transmitted to a section of the shear-zone, whose distance is W_1 from the free face, then

$$\frac{P_{w1}}{P_w} = \frac{\pi r^2 p_1}{\pi r^2 P_e} = \frac{2 \pi r W_1 K}{2 \pi r W K}$$

whence

$$\frac{p_1}{P_e} = \frac{W_1}{W}$$

P_e being the pressure of the explosive per surface-unit in the chamber and p_1 the pressure per surface-unit on the section of the shear-zone, whose distance is W_1 from the free face.

Since for any other distance W_2 we should have $\frac{p_2}{P_e} = \frac{W_2}{W_1}$, it is evident that

$$\frac{p_1}{p_2} = \frac{W_1}{W_2}$$

That is:—The intensities of the forces acting upon the surface-units of the sections decrease directly as their distance from the chamber.

Prof. Höfer, on the contrary, concludes that the intensities of the forces acting upon the surface-units are to each other inversely as the squares of the corresponding radii (distances from the chamber), which cannot be admitted.

28. *Maximum Pressures Developed by Explosives.*—If the maximum pressure developed by an explosive is P_e, K the modulus of

ultimate strength of the rock, r the radius of a spherical chamber, and W the line of least resistance, then according to Art. 25 (Addenda)

$$r = \frac{2\ K\ W}{P_e}$$

Suppose P_e = 12,000 kilogrammes per square centimetre, or 12,000 × 2·2 × 6·45 = 170,280 lb. per square inch and the modulus of ultimate strength of the rock to be 4000 lb. per square inch, then if the line of least resistance to be ruptured is 30 inches, the radius of spherical chamber, or charge required, is

$$r = \frac{2 \times 4000 \times 30}{170280} = 1\cdot41 \text{ inch}$$

The diameter of the charge would then be 2·82 inches, and the quantity of charge (if one of the dynamite compounds)—

$$\cdot5236 \times (2\cdot82)^3 \times \cdot03612 \times 1\cdot55 = \cdot66 \text{ lb.}$$

This example shows that the charge for a blast in rock could be directly calculated, if we knew the maximum pressures developed by the different explosives under ordinary working conditions, and the modulus of ultimate strength of the different rocks.

53. *Best Lengths for Borehole Charges.*—As the length of a borehole should be equal to the sum of the length of the line of least resistance and half the length of charge, it is evident that the cubic contents of a properly proportioned shot-hole, putting W for the length of the line of least resistance, m for the length of charge, and d for the diameter of bore-hole, should be

$$\cdot7854 \left[W + \frac{m}{2}\right] d^2$$

and if the borehole is drilled at an average cost of B pence per cubic inch, its cost will be

$$\cdot7854 \left[W + \frac{m}{2}\right] d^2\, B$$

Since the length of charge may be given in terms of the diameter of the borehole, by making $m = n\,d$, the cubic contents of the charge will be

$$\cdot7854\, n\, d^3$$

and its cost, putting E for the cost per cubic inch of the explosive (including fuse and detonator)

$$\cdot7854\, n\, d^3\, E$$

Therefore for the relative cost, p_v of unit-volume (cubic inch) of rock blasted we can put

$$p_v = \frac{\cdot 7854\left(W + \frac{n\,d}{2}\right) d^2\,B + \cdot 7854\, n\, d^3\,E}{W^3} \qquad \text{(eq. 1)}$$

But $W = \dfrac{A}{C_a\,S} = \dfrac{n\,d}{2C_a\,(n+1)}$,

and substituting in (eq. 1) we get

$$p_v = \frac{6\cdot 2832\ C_a^{\,3}\left[\frac{B}{2}(n+1)^3 + \frac{B\,(n+1)^2}{2C_a} + E\,(n+1)^3\right]}{n^2} \qquad \text{(eq. 2)}$$

A blast in a borehole will evidently give the greatest efficiency when p_v is a minimum in the above equation.

Differentiating therefore with reference to n we obtain—

$$\frac{B}{2}\left[n^3 - 3n - 2\right] - \frac{B}{C_a}\left[n + 1\right] + E\left[n^3 - 3n - 2\right] = 0,$$

$$\left[n^3 - 3n - 2\right]\left[\frac{B}{2} + E\right] = \frac{B}{C_a}\left[n + 1\right]$$

$$n^2 - n = \frac{2\,B}{C_a\left[B + 2\,E\right]} + 2$$

and $$n = \sqrt{\frac{2\,B}{C_a\left[B + 2\,E\right]} + 2\tfrac{1}{4}} + \tfrac{1}{2} \qquad \text{(eq. 3)}$$

If the ratio $\dfrac{B}{E} = b$

$$n = 1\cdot 1\sqrt{\frac{2\,b}{C_a\left[2 + b\right]}} + 2 \text{ nearly} \qquad \text{(eq. 4)}$$

Example.—If for gelatine dynamite the coefficient C_a of the rock is $\cdot 012$, the cost E of the explosive $\cdot 6d.$ per cubic inch and the cost B of drilling the rock $\cdot 27d.$ per cubic inch, what length of charge should be used to obtain the greatest economy in the blasting of the rock?

Since $b = \dfrac{B}{E} = \dfrac{\cdot 27}{\cdot 6} = \cdot 45$,

$$n = 1\cdot 1\sqrt{\frac{2 \times \cdot 45}{\cdot 012\left[2 + \cdot 45\right]}} + 2 = 1\cdot 1\sqrt{32\cdot 61}$$

$$n = 6\cdot 28.$$

Hence the length of charge in this case should be 6·28 times the diameter of the borehole.

The price of ·6*d.* per cubic inch for the explosive corresponds to a price of 100*l.* per ton for gelatine dynamite and gelignite.

Experience shows that the cost of drilling the rock is approximately proportional to its hardness and fineness of grain, and for different rocks whose coefficients are C_a and C_{a1} and costs of drilling B and B_1,

$$B : B_1 : : C_a : C_{a1}$$

or

$$\frac{B_1}{B} = \frac{C_{a1}}{C_a} \qquad \text{(eq. 5)}$$

The cost of drilling with a 10-drill plant may be taken under ordinary conditions as follows—

				£	s.	d.
Compressed air costs per machine drill with 10-drill plant—						
Fuel for compressing 75 cub. ft. of free air per minute to 70 lb. pressure, 2 cwt. at 20*s*.	£0	2	0			
Water, 18 gal. at say 4*s*. per 1000 gal.	0	0	1			
Oil and waste	0	0	9			
Labour, 2 men at 5*s*. per shift = per drill	0	1	0			
Depreciation on £2000 at 10 % = £200, or 13*s*, 4*d*. per day, or 16*d*. per drill, which apportioned over two shifts	0	0	8			
Piping, say	0	1	6			
Compressed air charges per drill per shift				0	6	0
Rock drill costs—						
Oil	0	0	9			
Labour, 2 men at 5*s*.	0	10	0			
Upkeep	0	3	0			
Steel waste	0	0	4			
Smithing	0	1	6			
				0	15	7
Administration				0	2	5
Total cost per drill per shift				£1	4	0

In hard rock, whose coefficient is say ·02 with gelatine dynamite, a good machine drill will bore 8 holes per shift, having an average depth of 54 inches, and finishing with a diameter of $1\frac{1}{2}$ inch at bottom. Then as $1\frac{1}{2}$ inch is the useful diameter of the bore hole for the cubic contents of the 8 holes, we have

$$8 \times 54 \times (1·5)^2 \times ·7854 = 763·4 \text{ cubic inches.}$$

The cost is therefore

$$\frac{1l.\ 4s.}{763·4} = ·3773d. \text{ per cubic inch.}$$

Substituting this value in Eq. 5

$$B = \frac{\cdot 3773}{\cdot 02} C_a = 18\cdot 865\ C_a$$

and consequently

$$b = \frac{18\cdot 865\ C_a}{E}.$$

Therefore, by (Eq. 4)

$$n = 1\cdot 1 \sqrt{\frac{2 \times 18\cdot 865}{2\,E + 18\cdot 865\ C_a}} + 2$$

for the given value of b.

Therefore, when $C_a = \cdot 015$ and $E = \cdot 6$

$$n = 1\cdot 1 \sqrt{\frac{2 \times 18\cdot 865}{2 \times \cdot 6 + [18\cdot 865 \times \cdot 015]}} + 2 = 5\cdot 76.$$

Allowing an increase of 25 per cent. in the cost of drilling

$$B = \frac{\cdot 3775 \times 1\cdot 25}{\cdot 02} C_a = 23\cdot 6\ C_a$$

and

$$n = 1\cdot 1 \sqrt{\frac{2 \times 23\cdot 6}{2 \times \cdot 6 + [23\cdot 6 \times \cdot 015]}} + 2 = 6\cdot 26.$$

If $C_a = \cdot 03$

$$n = 1\cdot 1 \sqrt{\frac{2 \times 23\cdot 6}{2 \times \cdot 6 + [23\cdot 6 \times \cdot 03]}} + 2 = 5\cdot 69.$$

As an example, taking different lengths of charges a, b, c for a $1\frac{1}{2}$ inch diameter bore hole in rock whose coefficient is ·015, we get the following figures :

Charge.	$n =$	Length of Charge	Line of Resistance.	Depth of Bore hole.	Total Cost of Drilling Holes at ·354*d.* per cub. in.	Cost of Charge at ·6*d.* per cub. in.
		in.	in.	in.	*d.*	*d.*
a	5·26	7·89	42·01	45·95	28·75	8·37
b	6·26	9·39	43·11	47·80	29·90	9·96
c	7·26	10·89	43·95	49·40	30·91	11·55
d	12·0	18·0	46·16	55·16	34·51	19·09

The length of charge is n times the diameter of bore hole : the line of resistance $= \frac{A}{C_a\,S}$; the depth of bore hole $= W + \frac{m}{2}$, and the cubic contents of bore hole $= \cdot 7854\ d^2\,D$.

Hence the relative cost per unit of rock blasted is

$$(a) \quad \frac{28 \cdot 75 + 8 \cdot 37}{(42 \cdot 01)^3} = \frac{37 \cdot 12}{74141} = \cdot 0005005d.$$

$$(b) \quad \frac{29 \cdot 90 + 9 \cdot 96}{(43 \cdot 11)^3} = \frac{39 \cdot 86}{80119} = \cdot 0004975d.$$

$$(c) \quad \frac{30 \cdot 91 + 11 \cdot 55}{(43 \cdot 95)^3} = \frac{42 \cdot 46}{84894} = \cdot 0005002d.$$

$$(d) \quad \frac{34 \cdot 51 + 19 \cdot 09}{(46 \cdot 16)^3} = \frac{53 \cdot 6}{98355} = \cdot 0005449d.$$

The cost per unit of rock blasted is therefore a minimum when $n = 6 \cdot 26$, the increased cost for the charge whose length is $12d$ being

$$\frac{[\cdot 0005449 - \cdot 0004975]\ 100}{\cdot 0004975} = 9 \cdot 53 \text{ per cent.}$$

Under the given conditions the greatest economy will be obtained when the length of charge is 6·26 times the diameter of bore hole.

Since for a length of charge $= 12d$. $W = \frac{\cdot 46\, d}{C_a}$, and for a length of charge $= 6d$. $W = \frac{\cdot 43\, d}{C_a}$, the lines of resistance given in Table I. must be reduced for the latter by multiplying by the factor $\frac{\cdot 43}{\cdot 46} = \cdot 935$.

With regard to the length of charge being limited as above, it must always be remembered that this appertains only to charges which are required to shear the rock, in other cases the length of charge should be proportioned as explained in Chapter XI.

The preliminary report of the Geological Survey of Canada gives the following as the costs per ton of ore extracted in Rossland, British Columbia.

—	1897.	1899.	1901.	1902.	1903.	1904.
	\$	\$	\$	\$	\$	\$
Drilling	0·94	1·53	0·43	0·73	0·64	0·46
Blasting	..	..	0·04	0·06	0·05	0·05
Explosives	0·27	0·25	0·13	0·26	0·22	0·16
Machine drills, fitting and expenses	0·23¼	0·05	0·07	0·14	0·07	0·08
Compressed air . . .	..	0·21	0·09	0·15	0·18	0·11
Totals . .	1·44¼	2·04	0·76	1·34	1·16	0·86
Ground stoped in tons .	..	45,810	17,910	20,327	58,683	53,084

This table clearly shows that the drilling cost is the chief item in rock blasting.

Table showing Variation of Lengths of Charges according to Coefficient of Rock and Cost of Drilling and Explosive.

Coef-ficient of Rock *Ca.*	Cost of Boring per cub. in. B	Cost of Explosives per cub. in. E	Multiple of Diam. for Length of Charge. *n*	Coef-ficient of Rock *Ca.*	Cost of Boring per cub. in. B	Cost of Explosives per cub. in. E	Multiple of Diameter for Length of Charge. *n*
	d.	*d.*			*d.*	*d.*	
0·01	0·3	0·4	8·0	0·02	0·4	0·6	5·7
0·01	0·4	0·4	8·8	0·02	0·6	0·6	6·5
0·01	0·5	0·4	9·4	0·02	0·5	0·6	6·1
0·01	0·6	0·4	9·9	0·03	0·3	0·4	5·0
0·01	0·3	0·5	7·5	0·03	0·4	0·4	5·5
0·01	0·4	0·5	8·2	0·03	0·5	0·4	5·8
0·01	0·5	0·5	8·8	0·03	0·6	0·4	6·0
0·01	0·6	0·5	9·3	0·03	0·7	0·4	6·3
0·01	0·3	0·6	7·0	0·03	0·8	0·4	6·4
0·01	0·4	0·6	7·7	0·03	0·5	0·5	5·5
0·01	0·5	0·6	8·3	0·03	0·4	0·5	5·1
0·01	0·6	0·6	..	..	..	..	..
0·02	0·3	0·4	5·9	0·03	0·6	0·5	5·7
0·02	0·4	0·4	6·5	0·03	0·7	0·5	6·0
0·02	0·5	0·4	6·9	0·03	0·8	0·5	6·2
0·02	0·6	0·4	7·2	0·03	0·3	0·6	4·5
0·02	0·3	0·5	5·5	0·03	0·4	0·6	4·9
0·02	0·4	0·5	6·0	0·03	0·5	0·6	5·2
0·02	0·5	0·5	6·5	0·03	0·6	0·6	5·5
0·02	0·6	0·5	6·8	0·03	0·7	0·6	5·[illegible]
0·02	0·3	0·6	5·2	0·03	0·8	0·6	5·9

129. *Safety Explosives.*—The following is a complete list of the names of permitted explosives as defined in the Schedules to the Explosives in Coal Mines Orders of the 17th December, 1906, of the 8th April, 1907, of the 26th May, 1908, and of the 20th August, 1908.

Explosives in First Schedule.

Abbcite	Dragonite	Permonite
Albionite	Electronite	Permonite II.
Ammonal	Excellite	Phœnix powder
Ammonal B	Extra carbonite	Pit-ite
Ammonite	Faversham powder	Rexite
Amvis	Fracturite	Ripping ammonal
Aphosite	Geloxite	Rippite
Arkite	Good luck	Roburite No. 3
Bellite No. 1	Haylite No. 1	Russelite
Bellite No. 3	Kolax	Saxonite
Bobbinite	Kynite	Stowite
Britonite	Kynite condensed	Thunderite
Cambrite	Minite	Titanite
Carbonite	Monobel powder	Tutol
Celtite	Negro powder	Victorite
Cliffite	Nobel carbonite	Virite
Clydite	Normanite	Westfalite No. 1
Colliery steelite	Oaklite No. 1	Westfalite No. 2
Cornish powder	Oaklite No. 2	Withnell powder
Curtisite	Odite	
Dahmenite A	Permitite	

EXPLOSIVE IN SECOND SCHEDULE.

Bickford's Igniter Fuse.

(1) In all coal mines in which inflammable gas has been found within the previous three months in such quantity as to be indicative of danger, or which are not naturally wet throughout, all explosives other than the above are prohibited under the explosives in Coal Mines Orders of the 17th December, 1906, 8th April, 1907, 26th May, 1908, and 20th August, 1908.

In all such mines, or parts thereof, the use of permitted explosives is prohibited unless the following conditions are observed :—

(*a*) Every charge shall be fixed by a competent person, called the shot-firer, appointed in writing for this duty by the owner, agent, or manager of the mine, and not being a person whose wages depends on the amount of mineral to be got.

(*b*) Every charge of the explosive shall be placed in a properly drilled shot-hole and shall have sufficient stemming, and each such charge shall consist of a cartridge or cartridges of not more than one description of explosive.

(*c*) No cartridge shall be used unless it is marked with the outline of a crown with the letter P in the centre, and in addition thereto the words " Permitted Explosive."

(*d*) No charge shall be fired except by means of an efficient electrical apparatus so enclosed as to afford reasonable security against the ignition of inflammable gas, or by a permitted igniter-fuse as defined in the Order.

(*e*) Where the charge is fired by an electrical apparatus, the shot-firer shall not use a cable for the purpose which is less than 20 yards in length. He shall himself couple up the cable to the charge and shall do so before coupling the cable to the firing apparatus. He shall also himself couple the cable to the firing apparatus. Before doing so he shall see that all persons in the vicinity have taken proper shelter. Should the charge miss fire, he shall immediately dis-connect the cable from the firing apparatus.

(*f*) Every electrical firing apparatus shall be provided with a removable handle or safety plug, or push button, which shall not be placed in position or operated until the shot is required to be fired and which shall be removed or released as soon as a shot has been fired. The removable handle or safety plug shall at all times remain in the personal custody of the shot-firer whilst on duty.

(*g*) Each explosive shall be used in the manner and subject to the conditions prescribed for the same.

(*h*) Where two or more shots are being fired in the same place, and such shots are not fired simultaneously, the shot-firer shall make an examination for gas immediately before the firing of each shot, and shall not fire the shot unless he finds the place where the shot is to be fired and all contiguous accessible places within 20 yards free from gas and safe for firing.

This order does not, however, prohibit the use of safety fuse in any mine in which inflammable gas has not been found within the previous three months in such quantity as to be indicative of danger.

In every coal mine the use of any explosive is prohibited in the main haulage roads and in the intakes unless all workmen have been removed from the seam in which the shot is to be fired, and from all seams communicating with the shaft on the same level, except the men engaged in firing the shot, and in addition such other persons not exceeding ten in number as are necessarily employed in attending to the ventilating furnaces, steam boilers, engines, machinery, winding apparatus, signals or horses, or in inspecting the mine; or unless a permitted explosive is used when the mines are not naturally wet throughout, and every part of the roof, floor, and sides of the main haulage road or intake, within a distance of 20 yards from the place where it is used, is, at the time of firing, thoroughly wet, either naturally or from the application of water thereto.

The above does not apply to such portions of the main haulage roads and intakes as are within 100 yards of the coal face.

Detonators are not allowed to be used in or taken for the purpose of use into any mine unless the following conditions are observed :—

(*a*) Detonators shall be under the control of the owner, agent, or manager of the mine, or some person or persons specially appointed in writing by the owner, agent or manager, for the purpose, and shall be issued only to shot-firers or other persons specially authorised by the owner, agent, or manager, in writing.

(*b*) Shot-firers and other authorised persons shall keep all detonators issued to them until about to be used in a securely locked case or box, separate from any other explosive.

In the case of a shaft being sunk from surface, primers for charges may be fitted with detonators on the surface before being taken into the shaft, provided the primers are so fitted in a workshop established under the Explosives Act, 1875 (section 47), and are only taken into the shaft immediately before use by the shot-firer or other authorised person and in a thick felt bag or other receptacle sufficient to protect them from shock.

Mines of clay or stratified or nodular ironstone are excluded from the regulations, also shafts in course of being sunk from the surface, or deepened, or drifts and other outlets being driven from the surface in so far as no inflammable gas (or indications thereof) has been found.

The owner, agent, or manager, must take all reasonable means to prevent deterioration of the explosive or igniter fuse while stored, and should obtain a written certificate from the maker that each explosive complies with the terms of the Act.

According to the report of the Inspectors of Explosives the consumption of safety explosives in Great Britain for the year 1907 was 7,764,122 lb.

Out of this total were used :—

	lb.		per cent.
Saxonite	1,721,193	or	22·17
Bobbinite	1,063,111	,,	13·69
Manobel Powder	711,691	,,	9·17
Ammonite	562,405	,,	7·25
Carbonite	551,948	,,	7·11
Roburite	510,438	,,	6·57
Arkite	437,780	,,	5·64
Westfalite	405,691	,,	5·22
Bellite	371,455	,,	4·78
Rippite	306,408	,,	3·95
Faversham powder	224,200	,,	2·89
Stowite	180,393	,,	2·32
Ammonal	114,806	,,	1·48

Saxonite, Manobel Powder, Carbonite, Arkite, Rippite and Stowite, contain large percentages of nitroglycerine. Bobbinite is a black powder mixture, the others are ammonium nitrate explosives.

From these practical results and the prices of the explosives, the engineer will be able to judge which explosive meets his requirements.

APPENDIX

A translation by Capt. C. W. Raymond, Corps of Engineers, U.S.A., of a paper on the Theory of Blasting and Military Mining, by Prof. H. Höfer, of Przibram, in 1879, was issued to the U.S. Corps of Engineers in 1881 on the recommendation of the Board of Engineers for Fortifications, River and Harbour Improvements, etc.

This paper reviews briefly the question of crater forms and their varieties dependent upon the nature of the rock or earth blasted, assuming the right-cone crater to result with normal charges, that is, with charges arranged with lines of resistance to produce the maximum volumes of displacement, also that by increasing the line of resistance of a charge beyond such length as will give the maximum displacement, the angle enclosed by the exterior lines of the resulting crater will be less than a right angle and for the maximum length of the line of resistance becomes zero.

The cone-shaped cavity formed by a blast with a concentrated charge is called a "crater of projection," and the mass displaced, a "cone of projection."

According to the theory, the cone of projection attains its maximum, or the blast ejects the largest volume, when the quotient of the line of least resistance by the base-radius is equal to 1·11805, or when the angle at the base of the cone is equal to 48° 11′ 22·8."

Prof. Höfer says, "It has been already accepted for centuries that in a crater of projection corresponding to a normal charge—and only such are considered by engineers—the line of least resistance is equal to the base-radius, or what is the same thing, the angle at the base is 45°. This value, confirmed by many experiments, agrees very well with the theoretical value, as the small difference of about 3° may be regarded as an error of observation. This may readily be accepted in military mines, as their craters have to be excavated in consequence of the falling back of most of the débris into the mine."

This conformity between theory and practice, Prof. Höfer claims, is of the highest importance for the whole theory of blasting or military mines, and that it is a confirmation of his fundamental assumption that the transmission of the waves of concussion occurs in rocks practically as in perfectly elastic media, or at all events this view is admissible for those spheres within which occur the destructive effects desired alike by miner and military engineer. Although military mines for which this conformity was determined are generally situated in loose ground, Prof. Höfer considers these laws of transmission are even more applicable to the rocks with which miners usually have to deal.

Although it is a well-known fact in the blasting of rock that the maximum displacement corresponds to the maximum line of least resistance that can be blasted, Prof. Höfer says, "With the same charge and in the same rock the cone of projection will exhibit very different dimensions, according to the line of least resistance, that is the shortest, or perpendicular distance from the chamber to the free surface. If a certain depth, or rather length, of this line is exceeded, the circular base lying in the free surface diminishes with the increase of the line forming the height of the cone of projection, and for a certain line of least resistance becomes zero. When this occurs, the throwing effect of the explosive against the free surface ceases." That is to say, the blasting of the maximum line of least resistance produces the minimum amount of displacement of the rock. Let us see what actually occurs if equal concentrated charges be fired, say, in any solid rock as indicated by Fig. 5.

Suppose B A C to be the maximum crater of projection, then any increase of the line of least resistance W, will have the result that there will be no displacement of any part of

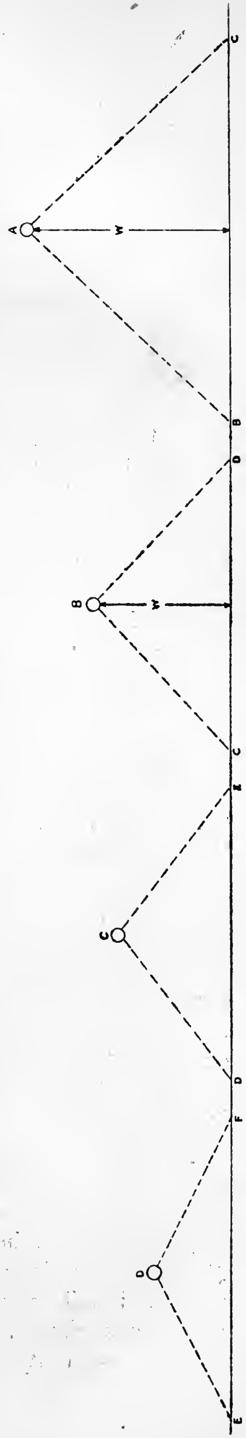

Fig 5

the rock or the throwing effect of the explosion against the free surface ceases. Reducing now the line of least resistance for the same charge, we obtain first a crater C B D similar in every respect to the crater B A C, and then, for the charges C and D, the craters D C E and E D F, in which the angles contained by the two opposite exterior lines are greater than a right angle. The explanation of this is that the length of the side of a crater cannot be less than the maximum line which can be sheared by the charge, viz. W, in the crater B A C. This, however, refers only to concentrated charges in spherical chambers in which the force of the blast is exerted equally in all directions, as its action is so sudden that there is no time for the less resistant part to yield before the full effect is felt on every part of the chamber. If, however, the chambers were disc-shaped, with their flat surfaces parallel to the free faces B C, C D, D E and E F, right cone-craters would be obtained in each case.

A crater, therefore, cannot be blasted in rock whose line of least resistance is greater than that which will produce the maximum volume of displacement, and consequently the cone of projection is a maximum when the line of least resistance is a maximum, which is quite contrary to Prof. Höfer's theory.

Prof. Höfer further states, " When the explosion takes place in a chamber O (Fig. 6), the shock will be transmitted in concentric

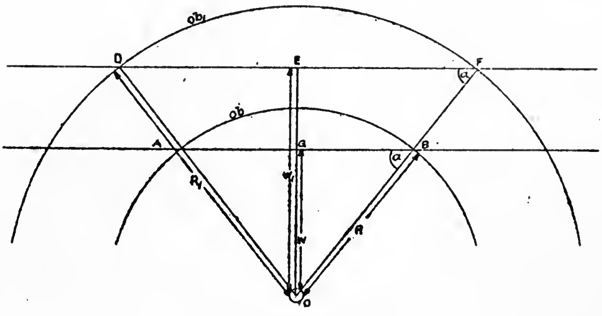

FIG. 6.

spherical layers having O for a common centre. As this force is distributed over gradually increasing spherical surfaces, it decreases per surface-unit as the spherical surface increases. These are, therefore, obviously proportional. The force p acting upon a surface-unit of the spherical shell $o\,b$, is to the force p_1 acting on the surface-unit of the spherical shell $o\,b_1$ inversely as the surfaces, or

$$p_1 : p = o\,b : o\,b_1$$

If $o\,b_1$ has the radius R_1 and $o\,b$ the radius R, then

$$p_1 : p = 4\pi R^2 : 4\pi R_1^2 = R^2 : R_1^2$$

$$\frac{p_1}{p} = \frac{R^2}{R_1^2}$$

from which the inference is drawn that the intensities of the forces acting upon the surface-units are to each other inversely as the squares of the corresponding radii (distances from the chamber).

This evidently is not in agreement with our deductions (Art. 25) that for the same explosive the resistances to rupture vary as the squares of the lines of resistance, and that the forces acting upon the surface-units decrease directly as the distances from the chamber (see Art. 25, Addenda). The explanation of this is that the resulting force of an explosion in a spherical chamber is not transmitted inversely as the surfaces of the spherical shells in accordance with Prof. Höfer's theory, but through shear zones whose shearing surfaces are proportional to the product of the line of resistance, and the periphery of chamber at right angles thereto.

If the free surface is spherical, every radial line will offer the same resistance to rupture, there will be no flexure of the mass, and rupture will result only from shearing action of the force.

If, however, a plane surface exists along A B, and the charge at O is only just sufficiently powerful to shear along the line of least resistance W, it is evident that it cannot shear along the lines of resistance O A and O B corresponding to the lengths of the sides of the crater, nevertheless, owing to the flexure of the rock around the line of least resistance, as explained in Art. 12 (Addenda), the whole mass of rock B O A will be ruptured.

In regard to the determination of the charges Prof. Höfer on the assumption that the outward forces of a blast upon units of surface vary inversely with the squares of the distances from centre of charge, arrives at the formula:—

$$\frac{W_1}{W} = \sqrt{\frac{L_1}{L}} = \sqrt{m}$$

in which L and L_1 are the weights of the charges, and W and W_1, the corresponding lines of resistance. No account is taken of the amount of work done or mass thrown out, and the influence of the form of the chamber is entirely neglected. Apparently concentrated or spherical charges are intended, and since the volumes of spheres are to each other as the cubes of their diameters, for two charges whose diameters are 1 and 2, their relative weights should be as 1 to 8, therefore, by Prof. Höfer's formula

$$\frac{W_1}{W} = \sqrt{\frac{8}{1}} = \frac{2 \cdot 83}{1}$$

that is, a charge whose weight is 8 oz. should blast a line of resistance 2·83 times greater than one whose weight is 1 oz. The formula that we give on page 45 requires on the contrary, for the given conditions

$$\frac{W_1}{W} = \frac{d_1}{d} = \frac{2}{1}$$

or W_1 must be twice and not 2·83 times W, and therefore, the given charges of 1 and 8 oz. are not suited for lines of least resistance found by Prof. Höfer's formula.

Since the pressure developed by the explosion of the charge depends on the form of chamber, it is clear that a formula for the calculation of a charge which does not regulate the size and form of chamber according to the line of resistance to be blasted, can be of no practical value. Not only does Prof. Höfer's formula fail in this, but also in proportioning the weights of the charges to the work to be done after the rock has been loosened from its bed.

Conformity between the blasting of rock and soil does not exist in a general law to suit all cases, but only in so far as we consider the cohesive resistance and work required to eject the mass from its bed separately.

For rock we have a very high cohesive resistance, and the walls of the chamber practically incompressible, and for soil the cohesive resistance so low that it may be entirely neglected, whilst the chamber walls are so compressible that the pressure producing rupture or displacement will depend on the weight of charge and compressibility of the material, and not on the initial pressure developed by the explosion, as in the case of rock. For soil, therefore, the calculation of the charge will depend on the compressibility of the material, whereas this is of minor importance in rock blasting.

The experiments at Olmutz, referred to by Prof. Höfer in his paper, were carried out in alluvial, compact ground; argillaceous sand and loess formation; natural stratified soil, tenacious clayey earth, and wet loam, and cannot be accepted as forming any basis of proof of his theory for rock blasting.

INDEX.

(The Figures give the number of the Pages.)

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