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New Units of Crusher Capacity and Crusher Efficiency

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(New York Meeting, February 1941)

THIS paper proposes two units (believed to be new) for designating, respectively, capacity and efficiency for primary and intermediate crushers.

CAPACITY

Operators know that the tonnage of rock that can be put through a particular crusher in a given time depends primarily upon the crusher setting (open setting for jaw and primary gyratory types; closed setting for cones and rolls), and secondarily on the lithological character of the feed, its maximum size, the amount of fines that it carries, its moisture content, and the method of feeding. The proposed capacity formula states a relationship between all of these factors in one empirical equation, as follows:

$$T_R = TR_{80}KK'K'' \qquad [1]$$

in which T_R, called "reduction tons per hour" is the new capacity unit; T = tons of feed per hour *requiring crushing;* R_{80} = "80-per cent reduction ratio"; and K, K', and K'' are constants that approximate quantitatively lithological character, moisture content and method of feeding, respectively.

T is the hourly weight of material in the feed that is coarser than the coarsest discharge of the crusher. (For marked departures from the normal specific-gravity range—2.5 to 2.7—T should be adjusted to corresponding volumes.) In the crushers under consideration, discharge from the crushing zone is limited by the minimum dimension of a particle. Passage of the particle through a square-mesh screen is limited by its intermediate dimension. Sheppard[1] has shown that the ratio of maximum:intermediate:minimum average dimensions of broken rock fragments may be taken as about 1.5:1:0.6 for nonslabby rocks such as are normally crushed, while for slabby rocks, such as thin-bedded sedimentaries, foliated schists and gneisses, and the blocky metamorphics, the ratio is of the average order 1.3:1:0.3, with the possibility of an even smaller relative value for the thickness. Gaudin[2] finds the corresponding ratio for $-150 + 200$-mesh pyrite to be 1.4:1:0.6. The hourly tonnage of crusher-plant feed, therefore, should be diminished by the amount thereof that would pass a square-mesh screen of which the aperture is S times the crusher setting, where S is a shape factor with the value 1.7 for rocks that break exceptionally cubic; 2 for ores and rocks that exhibit no appreciable tendency to slabbiness; and 2.5 to 3, or, in exceptional cases, even larger, when the tendency to slabbiness is marked.

R_{80} is the quotient of the theoretical square-mesh aperture that would pass 80 per cent of the feed, divided by the aperture that would pass 80 per cent of the product. It is best determined from screen tests.

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[1] References are at the end of the paper.

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If, as is usual with primary crushers, a sizing test of feed is not available, the 80 per cent size thereof (l_F) may be estimated* with sufficient precision from

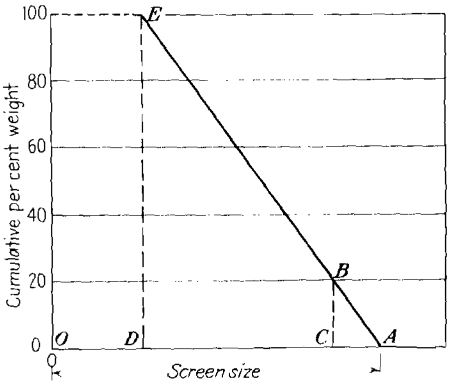

FIG. 1.—GRAPH OF FEED TO PRIMARY CRUSHERS.

the aperture of the feed-guard grizzly as follows:

$$l_F = \frac{S(4g + s_0)}{5} \qquad [2]$$

If the sizing curve of the product of a primary crusher is unknown, the 80-per cent size (l_P) may be estimated on the basis

$$l_P = Ms_0 \qquad [3]$$

where M has the values given in Table 1.

The reasons for using R_{80} instead of R_{Limiting} or $R = g/s_0$ is that R_{Limiting} may be < 1 for slabby feeds, that values for g are not available, and those for s_0 are not significant in intermediate crushers.

Choice of the 80-per cent point is based on the fact that the effect of stray coarse material on the curvature of the sizing-test graph has disappeared at that point, while the remaining segments of the sizing curves still have a substantially maximum spread.

TABLE 1.—*Values of M in Equation 3*

Character of Feed	Type of Crusher	M
Easily crushed rock, unscalped	Jaw	0.90
	Gyratory	0.75
Easily crushed rock, scalped	Jaw	1.15
	Gyratory	1.00
Average rock, unscalped	Jaw	1.15
	Gyratory	1.00
Medium tough or slabby rock, unscalped	Jaw	1.40
	Gyratory	1.25
Average rock, scalped	Jaw	1.40
	Gyratory	1.25

K is 1.0 for rocks such as medium hard to hard limestones and the like. When crushing harder, but clean-breaking stone, such as the general run of undecomposed igneous rocks, the same factor may be used provided the crusher is massive enough to stand up under the strain of such overpowering as is necessary to prevent slowing down under full load. For crushers not so sturdy, K should be taken as 0.85 for rocks of the general crushing behavior of hard, tough granites, and 0.75 for the tough basic igneous rocks such as traps and diabases. If rock is not clean-breaking, whether harder or softer than limestone, or if it tends to shatter on initial break to produce a choking tendency in the crushing zone, K should be taken as less than 1.0; probably around 0.75 is safe, unless the condition is aggravated.

K' will normally be 1.0, but may fall to 0.5 or less with crushers of the gyratory type, either primary or secondary, when set for relatively fine crushing, if undersize is fed containing just the intermediate amount of moisture that causes the fines to cake more or less when squeezed in the hand.

K'' will rarely exceed 0.75 for primary crushers, and probably averages nearer 0.5, although data for close quantification are lacking. Secondary crushers fed from surge

* The basis of estimate is that the feed to primary crushers (run-of-mine or run-of-quarry) approximates straight-line distribution in the coarser sizes. The graph of such a feed, scalped to the crusher open setting, is $\overline{AE}$ in Fig. 1, where $\overline{OA} = Sg$, when g = aperture of the grizzly limiting maximum size fed to the crusher; and $\overline{OD} = Ss_0$, where s_0 is the open setting of the crusher; S being the shape factor. Then

$$\overline{OC} = Sg - \overline{AC} = l_F$$

and, by similar triangles, and substitution as above, equation 2 develops.

bins by accurately controllable feeders can be operated with $K'' = 0.9$ to 1.0, but such feeding is rare in mill practice.

The product $KK'K''$ may be approximated in most cases by comparing T_R in average mill performance with values derived from the manufacturer's rating for the same machine. Manufacturers' ratings are usually based on thick-bedded, moderately hard, blocky limestone ($K = 1$); limited as to upper size so that it will enter the crusher and nip without delay due to bridging or nip failure, and scalped to remove all material finer than crusher open setting ($K' = 1.0$); fed by an attendant at the maximum rate—i.e., so that the crushing zone is kept full at all times ($K'' = 1.0$). The best data at hand indicate that for jaw crushers values of $KK'K''$ in the mills range from 0.3 to 1.8 times the values based on the manufacturer's ratings, averaging 1.0 for 20 installations; while for gyratory crushers the range is 0.3 to 2.3, averaging 0.88 for 17 reports. Two out of three field gyratories show values of $KK'K''$ below 1.0, the discrepancy being greater for the large crushers. So far as it is possible to evaluate the large discrepancies, the low field performances correspond to crushers that are oversize because of reception demands, while the high apparent performances are due to crediting the crusher with finished size in the feed, whether this was actually scalped or not.

Efficiency

Efficiency, in the usual mechanical sense of the ratio of work output to energy input, is not determinate for crushing operations, because of our inability to measure work output. Attempts to estimate output in terms of an equivalent unit, based on screen analyses, have been made,[3] but despite considerable experimental investigation and a large amount of contentious writing, no useful method has evolved. Millmen now, therefore, make a statement of effectiveness in terms of "tons per hp-hr. (or kw-hr)," with, usually, an accompanying statement of the limiting sizes of feed and product. The inadequacy of such a state-

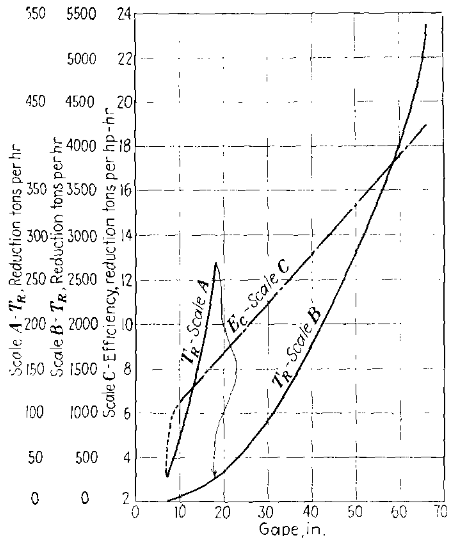

FIG. 2.—CAPACITY AND EFFICIENCY OF STANDARD JAW CRUSHERS.

ment is immediately apparent upon examination of a table of estimated crusher performances such as is found in any manufacturer's catalogue. Thus, for example, a 15 by 24-in. Blake crusher is rated for 15 to 33 tons per hour at settings of 1.5 to 3 in., respectively, at one power draft, say 30 hp., and will show therefore from 0.5 to 1.1 tons per hp-hr., according to the setting. Yet the same machine is actually doing substantially the same amount of effective primary crushing per unit of energy input throughout this range of tonnages and settings, and should, if the proper unit is chosen, show the same efficiency.

The efficiency unit proposed—using the word efficiency in the broad sense of relative effectiveness—is the "reduction ton per hp-hr." (E_c). It is derived from the equation:

$$E_c = \frac{T_R}{P} \qquad [4]$$

where P = horsepower "consumed," which is actually energy input, since the time factor is included in T_R.

Values of E_c computed from manufacturer's catalogue data are substantially constant for a given crusher throughout its range of settings (for the 15 by 24-in. crusher cited above the values are 4.8 and 5.4 for the respective instances); they increase for a given type of crusher with increase in size of machine (Fig. 2); they are lower for the jaw than for the gyratory over the same feed-size range; and are higher for cone-type or high-speed gyratory crushers than for jaw or primary slow-speed gyratories in the fine-product sizes; all of which facts are well in accord with operators' experience.

Fig. 2 gives values of T_R and E_c for jaw crushers with standard straight-element plates, according to manufacturer's ratings, adjusted to a ratio of length of receiving opening L to gape G of 1.5. If values are desired for crushers of different L/G ratios, the values of T_R as read from the chart should be multiplied by the quantity $\frac{L}{1.5G}$. Values of E_c are the same irrespective of L/G, for the reason that power consumption for crushers of the same gape and different lengths of receiving opening is subject to the same adjustment as T_R and the value of the ratio T_R/P is not, of course, affected when numerator and denominator are both multiplied by the same factor.

In calculating the values for plotting Fig. 2, average values for P taken from manufacturer's catalogues were multiplied by the factor 0.85. This is an average factor from some 20 to 25 reports from the field on power drafts by fully loaded jaw crushers.

The use of Fig. 2 for estimating can best be explained by an example. Assume that a crusher is wanted to break 50 tons per hour of run-of-mine granite that has passed through an 18-in. grizzly. It is desired to have a product that will pass a 4½-in. grizzly.

T

Normal run-of-mine granite will break roughly to a straight-line cumulative curve. Hence the tons of −4½-in. fines in the feed will be 12.5 tons per hour, leaving a net scalped feed of 37.5 tons per hour.

K for granite = 0.85

K' for scalped feed = 1.0

K'' may be taken as 0.75 on the assumption of a headframe surge bin with pan feeder on push-button control. Then

$$T = \frac{37.5}{0.85 \times 0.75} = 59$$

R

Feed sledged through an 18-in. grizzly may be taken to have 18-in. thickness of largest particle.

The square-mesh size (intermediate dimension) of such material will average about $\frac{18}{0.6} = 30$ in. This sets a minimum length of receiving opening, if sledging at the crusher is to be avoided.

If the product is to pass a 4½-in. grizzly, the *open* setting should be 4 to 4¼ in., say 4. The square-mesh limiting screen for the product will be about $4 \times 1.7 = 6.8$ inches.

The 80-per cent point of the feed, from equation 2, is 25.8 in. The 80-per cent point for product, from equation 3 and Table 1, is 5.6. Hence

$$R_{80} = \frac{25.8}{5.6} = 4.6$$

and

$$T_R = 59 \times 4.6 = 272$$

From Fig. 2, the corresponding gape is 18 inches.

A crusher to receive feed of 18-in. maximum thickness should have $G = \frac{18}{0.85}$

= 21.4-in. gape, in order to assure ready nip.

The receiving opening required is, therefore, 21.4×30-in. minimum. A 24-in. gape is the nearest larger standard. A 24×30-in. straight-element crusher will serve, but if a small safety factor for later plant expansion can be afforded, a 24×36-in. machine should be chosen to eliminate danger of necessity for sledging to permit entry of particles with particularly long intermediate dimensions.

P may be determined from equation 4. E_c from Fig. 2 for a 24 by 36-in. crusher is 9.6. $T_R = 272$. Hence

$$P = \frac{T_R}{E_c} = \frac{272}{9.6} = 28.3$$

Since Fig. 2 is plotted on the basis of 85 per cent of operating peak-load power draft, the minimum motor rating required $= 28.3 \div 0.85 = 33.3$ hp. The crusher chosen, however, was a 24 by 36, which has a reduction-ton per hour capacity of 575. The motor rating required for this is $\frac{575}{0.85 \times 9.6} = 70.4$ hp. Good practice would be to install a 75-hp. motor to permit of cold starting without excessive overload, and to give sufficient power if it should be desired to work the larger crusher at full capacity for a shorter daily period.

Similar curves can be drawn for other types of crushers, making suitable changes in the method of calculation to adapt them to the particular type of crusher.

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