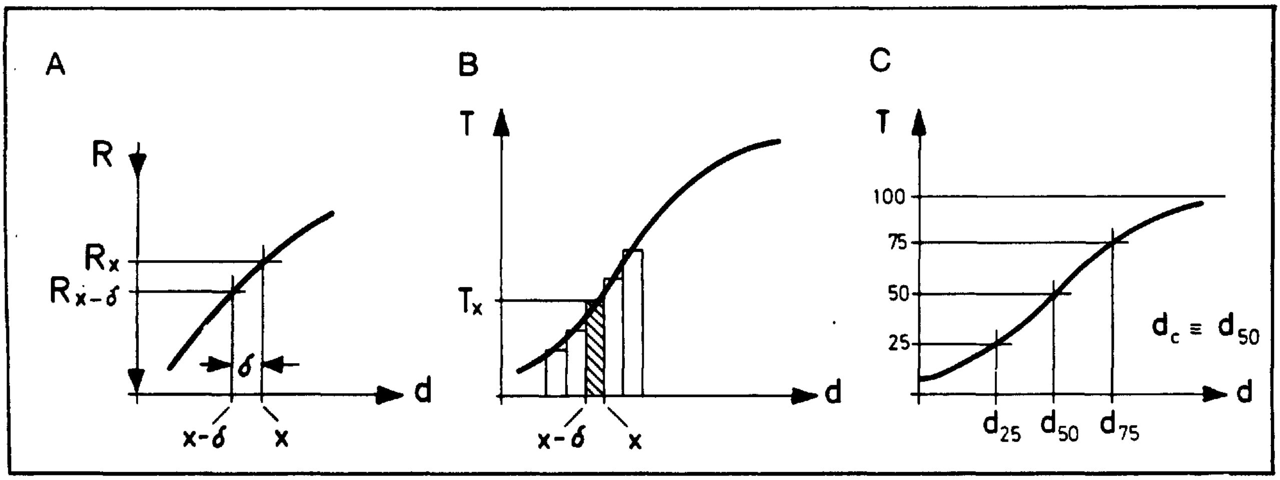


Fig. 3—Origin of Tromp curve: A) grain size distribution line (Rosin-Rammler-Bennett net); B) fractional mass recovery stair generates Tromp curve; C) points of Tromp curve determining separation mesh (cut point) and imperfection (Equation 10).

The Tromp curve is the plotting of the distribution numbers T_x (the differential mass recovery for the particle size range between x-δ and x) against the particle diameter d, calculated as follows:

$$T = \frac{\Theta \cdot \Delta R_c}{\Theta \cdot \Delta R_c + (1-\Theta) \cdot \Delta R_F} \tag{9}$$

The resulting step chart has to be approximated by a continuous curve. (See diagram B.)

The cut point (separation mesh) d_c is defined as that point on the Tromp curve for which T equals 0.5 or 50%, i.e., where the particles have an equal chance of going either with the overflow or the underflow. (See diagram C.) The sharpness of the cut depends on the particle diameters for $T = 0.75$ (d_{75}) and $T = 0.25$ (d_{25}), both of which may be taken from the Tromp curve. The so-called Imperfection I is calculated by the formula:[4]

$$I = \frac{d_{75} - d_{25}}{2 \cdot d_c} \tag{10}$$

Several formulas can be found in hydrocyclone literature for calculation of the separation mesh. The derivation of any such formula must start with Stokes' law describing the settling speed in laminar flow. Although the flow of the suspension in a hydrocyclone is turbulent (i.e., high Reynolds number), the flow surrounding the settling particle is laminar (small Reynolds number). The settling rate in a gravity field is

$$u_{so} = \frac{d^2 \cdot (\rho_s - \rho_l)}{18\,\eta} \cdot g \tag{11}$$

where ρ_s and ρ_l are the densities of solids and liquids, η the dynamic viscosity of the liquid, and g the acceleration due to gravity. Within centrifugal fields, the gravitational acceleration g is replaced by the centrifugal acceleration b or the product of g and the acceleration factor z. Therefore, the increased settling rate in the hydrocyclone amounts to

$$u_s = \frac{d^2 \cdot (\rho_s - \rho_l)}{18\,\eta} \cdot b = u_{so} \cdot z \tag{12}$$

The settling speed of particles having the cut point diameter d_c (i.e., u_s of Eq. 12) determines the capacity of the classifier area[6] at that point

$$q_F = \frac{Q}{F} \tag{13}$$

We can therefore write

$$u_s = q_F \tag{14}$$

We may now introduce the following relations:
For the separating area

$$F = \frac{2}{3} D \cdot \pi \cdot L_e \backsim \lambda D^2$$

wherein the "slenderness" figure λ is defined as the ratio of the effective length L_e and the cyclone diameter D. For the acceleration factor z, using r for $D/2$ and $v = \sqrt{2gH}$, with H as "pressure height" (gauge reading divided by slurry density, in meters):

$$z = \frac{v^2}{g \cdot r} = 4 \cdot \frac{H}{D} \tag{15}$$

For the volume flow capacity (semi-empirical):

$$Q = \mathrm{X} \cdot D^2 \cdot \sqrt{H} \tag{16}$$

X being the correction factor for the particular cyclone's geometry (cylinder length, cone angle, nozzle diameter, etc.). Finally, we get out for the cut point:[7]

$$d_c \backsim \sqrt{\frac{\eta}{g \cdot (\rho_s - \rho_l)}} \cdot \sqrt{\frac{\mathrm{X}}{\lambda}} \cdot \frac{\sqrt{D}}{\sqrt[4]{H}} \tag{17}$$

The first term collects the characteristic data of the suspension, and the second, those of the individual cyclone geometry, while the third term demonstrates that the size at the cut point is influenced by the square root of the cyclone diameter, but only by the fourth root of the pressure drop, and that inversely.

We may conclude from this statement that lower cut points could, at least in theory, be achieved with big cyclones, provided high enough pressures are applied. The economic restriction (power consumption, abrasion, etc.) is, however, of major importance, and in practice the cut point is determined primarily by the size of the cyclones. Fine separation undoubtedly requires small cyclones. As these have only a small capacity, several

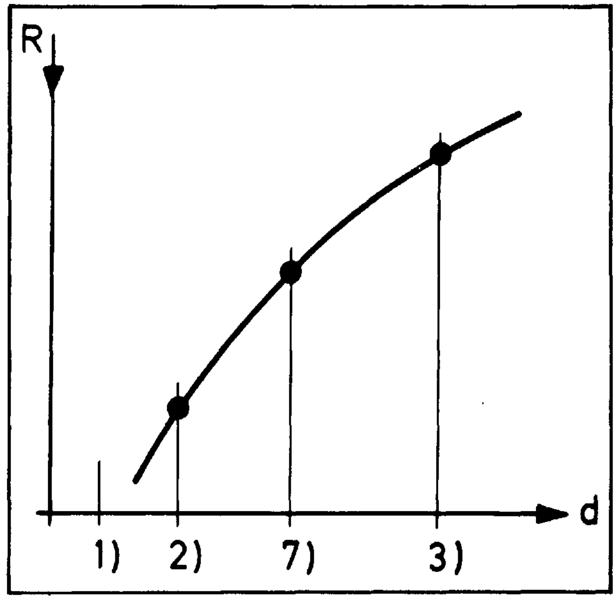
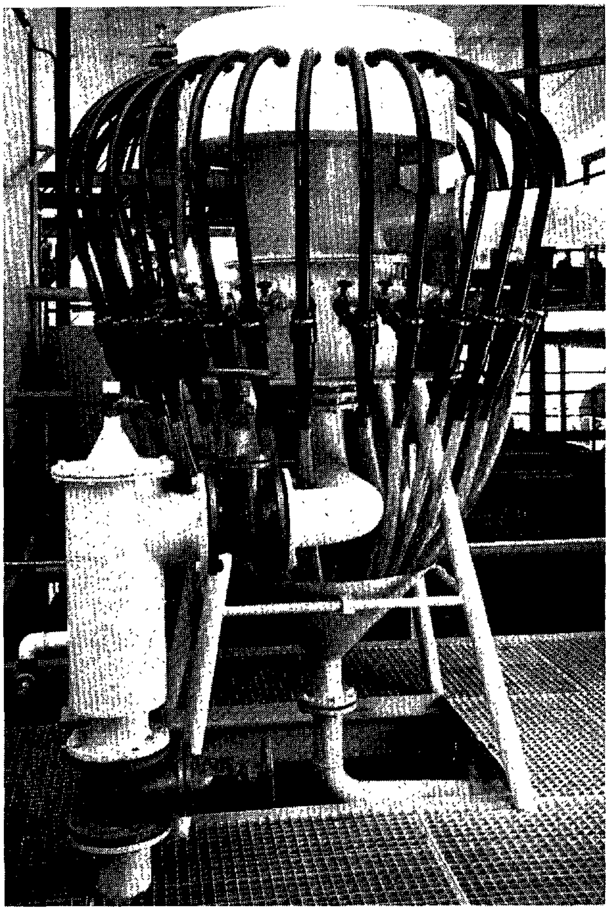


Fig. 4—Grain size distribution determines cyclone application: point 3 applies for degritting, point 7 for fractionation, point 2 for desliming, and point 1 for clarification.



Annular distributor with thirty-one 40-mm-dia hydrocyclones has an antiblocking filter in front. (See Fig. 20.)

cyclones have to be connected in parallel if high capacity or treatment rate is required.[8]

Hydrocyclone applications

There are 10 principal applications for hydrocyclones.

Thickening eliminates most of the water in a suspension to produce dewatered solids. True thickening aims at the recovery of all solids, resulting in the clarification of the liquid. However, flocculating agents cannot be used in hydrocyclones as settling aids, and in practice, the mass recovery is limited and a turbid overflow may be the result; i.e., desliming takes place in the cyclone. The building of tailing dams with hydrocyclones makes use of this phenomenon.

Desliming aims to eliminate fine particles from the overflow. This step is often necessary to improve the product for subsequent processes such as flotation, wet magnetic separation, filtration, etc. In chemical plants, desliming cyclones are often used for dewatering after a crystallization process, with the fine crystals being discharged with the overflow and recycled to the crystallizer, where they act as nuclei for crystal growth.

Degritting produces overflow as a product, and smaller amounts of oversize particles are rejected in the underflow. The difference between degritting and desliming is demonstrated graphically in Fig. 4 by using a grain size distribution line similar to that of Fig. 3A. Cut point 2 at the lower end of the curve refers to desliming, and cut point 3 at the upper end to degritting. Point 1, at the left of the curve, represents clarification or thickening, and point 7, in the middle, represents fractionation.

Closed circuit grinding processes often use cyclones for both degritting and desliming. Four possible circuits in which classifying hydrocyclones and wet mills can be employed together are illustrated in Fig. 7.

In case "a," the cyclone is installed ahead of the mill to deslime the feed. The fine fraction discharged with the overflow is blended with the ground mill product. The coarse underflow is ground in the mill and becomes the main component of the blended product. Such a circuit is called an "open" circuit.

In case "b," the feed goes directly to the mill, whose discharge is fed to a cyclone for degritting. The cyclone overflow is the product, and the underflow (oversize) goes back to the mill for more grinding, along with new feed. Such a circuit is called a "closed" circuit.

In case "c" (a combination of cases "a" and "b"), the circuit feed and the mill discharge are blended and classified together in the same cyclone. The cyclone deslimes the coarse feed and degrits the fine mill product. The cyclone overflow is the product, and the underflow is fed back to the mill. Because of the dual function of the cyclone, such circuits are called "double" circuits.

To optimize the performance of cyclones, desliming and degritting should be handled independently in separate cyclones of different sizes—as in case "d." Both overflows are then products and are fed to the mill. This is a more logical combination of cases "a" and "b" and is called an "improved" circuit.

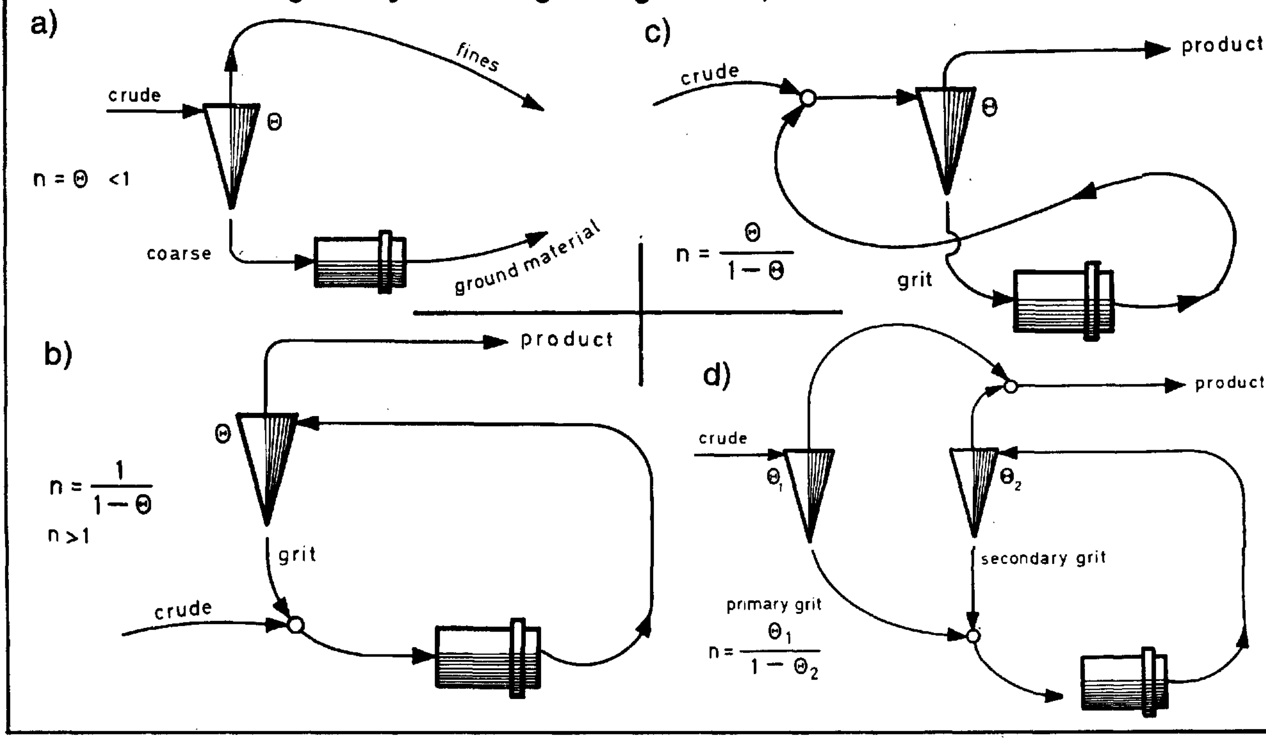
Fig. 6 illustrates some further modifications of the four basic circuits in Fig. 5, which may be of advantage in certain applications.

To reduce mill loading still further, a closed circuit can be further modified by employing two-stage degritting, as in case "e." The primary grit is deslimed in the second cyclone, and the intermediate product is blended with the primary feed. Without reducing the size quality of the final product, the rejects to the mill carry less fine residues.

If the primary overflow still carries too much oversize, it can be cleaned up in a second cyclone stage, as in case "f." Because the underflow of the secondary cyclone will carry too many fines, it is returned to the primary feed for further desliming.

Case "g" is a modification of circuit "d" that can be

Fig. 5—Cyclones in grinding circuits, standard cases



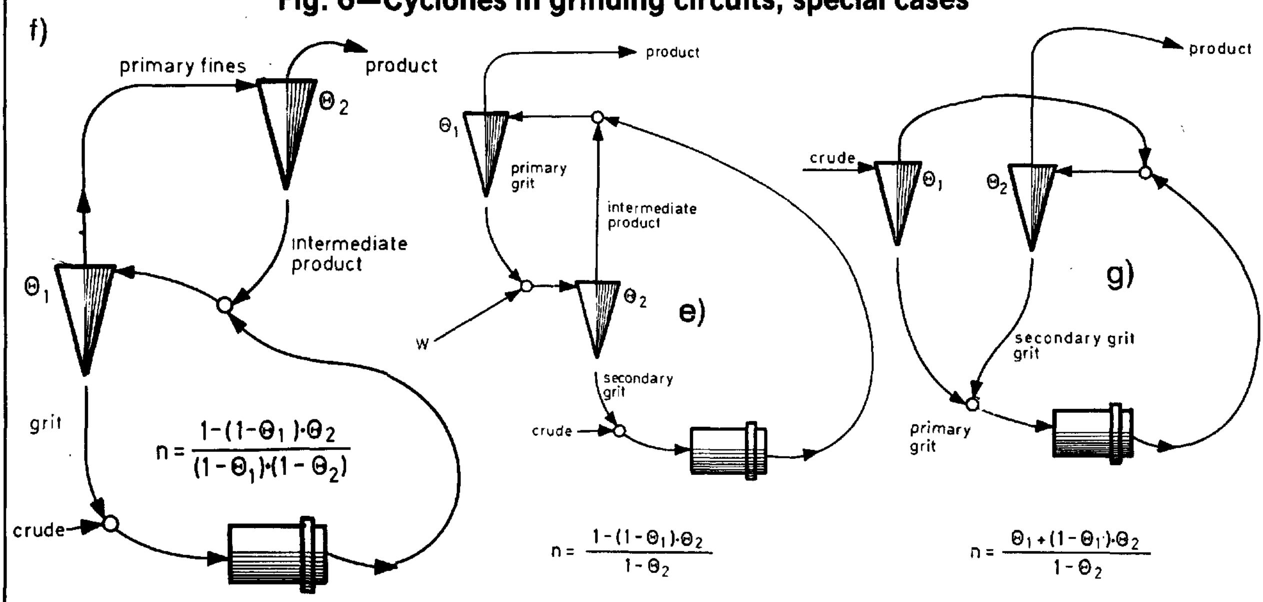
used if the overflow of the primary cyclone is not of the required quality. In this case, the primary overflow is blended with the secondary feed for a second cleaning operation.

The circulating load factor *n* compares the tonnage fed to the mill with the feed required to produce final tonnage of product. It can be calculated by using the given formulas when the mass recovery figures for the cyclones are known.[9]

Selective classification. The sorting of nonhomogenous feeds into their mineral components can be based on differing characteristics of the minerals. Specific gravity is used in heavy media separation, jigs, tables, and spirals; particle shape is used in tables and spirals; surface tension is used in flotation; electrical and magnetic properties are used in separators; and solubility is used in leaching processes. Sometimes, differences in grain size allow for a purely mechanical separation.

Antiparallel grain size distribution is illustrated in Fig. 7.[10] Kaolin is refined on the basis of such a grain distribution, with the finer product on the left being kaolin and that on the right being quartz. The cut at the size grain indicated in the diagram produces an enrichment of the fine kaolin in the cyclone overflow and of the coarse

Fig. 6—Cyclones in grinding circuits, special cases



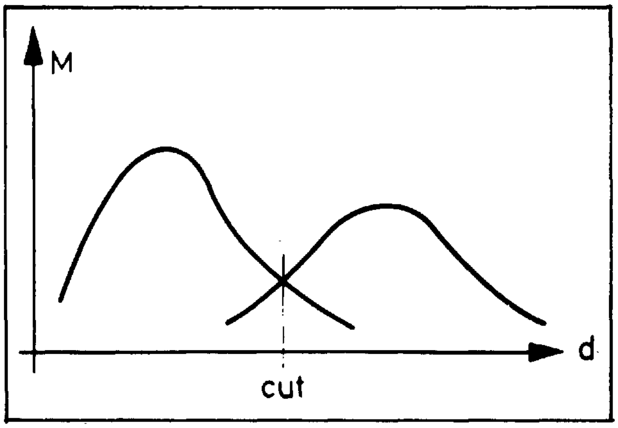


Fig. 7—Mineral distribution plotted against grain size for two minerals shows how selective classification works.

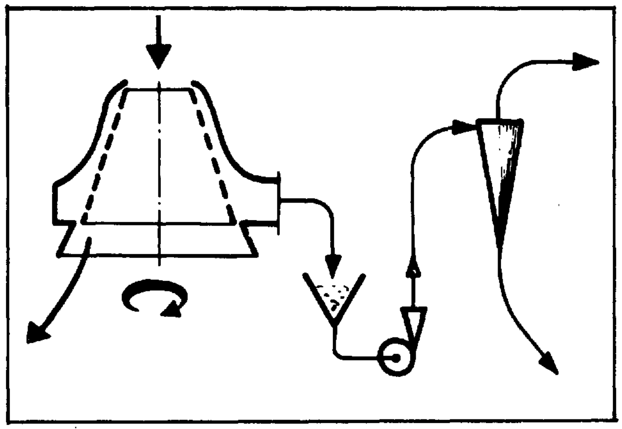


Fig. 8—A hydrocyclone can be used for the recovery of fine solids from the effluent of a screen-type centrifuge.

quartz in the underflow.

Solids recovery from turbid effluents (overflows or filtrates) of washing and dewatering equipment may prevent loss of fine product fractions. The recovery of these fine fractions from sand spirals, log washers, vibro screens and dewatering centrifuges (of the scroll discharge screen, vibro screen, or pusher type) is an attractive application of hydrocyclones.[12] A flowsheet for a scroll centrifuge and hydrocyclone combination is shown in Fig. 8.

Fractionation, which separates two fractions for further treatment in different processes, is another interesting application for hydrocyclones. Iron ore concentrate fractionation into sinter feed (coarse) and pellet feed (fine) is a typical example.

Preconcentration using hydrocyclones can be used to enrich the mineral components of ground ore if there are large differences in the specific gravity of the mineral components. This is a gravity concentration process—or perhaps more aptly, a centrifugal concentration process. No heavy medium is used in such cyclones.[13] Typical examples are the separation of heavy components such as metal sulphides (pyrite and zinc blende), metal oxides, and precious metals (gold, platinum, silver) from gangue.

Amberger Kaolinwerke cyclone plant beneficiates kaolin, making selective classifications on the basis of particle size.

Cyclones for such applications look different from standard cyclones; their flat bottoms and longer barrels have been proved to increase efficiency. Three cyclone configurations and typical applications are shown in Fig. 9.

Liquid recovery. If process water or parent liquids must be recycled, hydrocyclones may often produce satisfactory clarification. In coal washing plants, this is a major problem, especially when existing thickeners are overloaded. Hydrocyclones are often installed in parallel to keep the level of turbidity of the recycle water at a suitable value.

Countercurrent washing can eliminate adherent acids or lyes or fine particles from a product by periodical dilution and thickening in multiple cyclone stages. If X is the dilution factor of stage *i*, and *n* successive stages are used, the overall washing effect Ω amounts to

$$\Omega = \frac{1}{(1 - X_1)\ (1 - X_2)\ (1 - X_3) \cdots (1 - X_n)} \qquad (18)$$

A total of *n* times the amount of diluting water used per stage is required. Countercurrent washing can achieve high efficiencies by adding fresh water once only: to the last cyclone stage. The overall washing effect is somewhat lower than when fresh water is added several times, of course, but relative to the amount of fresh water added, the countercurrent system is much more effective.

A flowsheet for a four-stage CCW plant is shown in Fig. 10.

If the splitting factor of the cyclone (for the liquid only and not the volume of the suspension) is τ_1 for stage number *i*, the following formula gives the overall washing effect:[14]

$$\Omega = \frac{1 - \tau_4 (1 - \tau_3) - \tau_3 (1 - \tau_2) - \tau_2 (1 - \tau_1) + \tau_2 \tau_4 (1 - \tau_1)(1 - \tau_3)}{(1 - X_1)\ (1 - X_2)\ (1 - X_3)\ (1 - X_4)} \qquad (19)$$

Practical hydrocyclone operation

The determination of cyclone size is dependent on either the required mass recovery or the requirements of particle size split. Even for gravity concentration, there is a connection between enrichment, yield, and mass recovery. In no case should the cyclone size be determined by the desired total capacity. As the cut point is dependent on a lot of variables in addition to the cyclone size, practical tests are the only way to achieve the final layout.

Before discussing the various parameters influencing the cut point of a given cyclone, the test possibilities should be described. Cuts at small particle diameters (i.e., in the fine particle range) are tested with full size cyclones by running several sizes simultaneously in parallel or one after the other, while varying the pressure and/or the nozzles. Equation 17a shows the influence of cyclone diameter and pressure:

$$d_c \sim \frac{\sqrt{D}}{\sqrt[4]{H}} \qquad (17a)$$

A further approach would be the introduction of nozzle diameters using the characteristic factor, with d_o for overflow nozzle and d_e as equivalent diameter of the generally rectangular-shaped feed nozzle:

$$\psi = \frac{d_e \cdot d_o}{D^2} \qquad (20)$$

Multiplying Eq. 17a by $\sqrt{\psi}$ results in:[15]

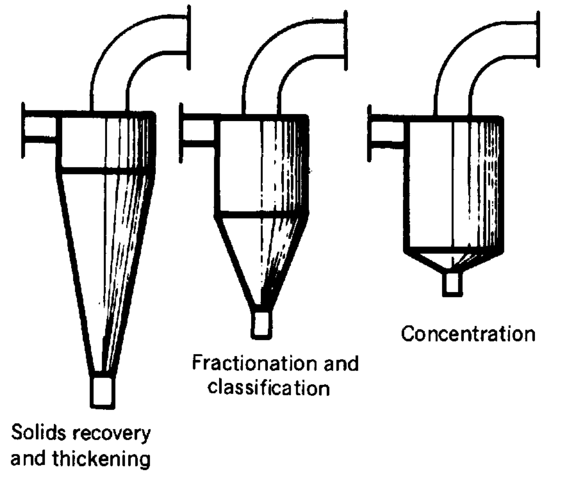
$$d_c \sim \sqrt{\frac{d_e}{D} \cdot \frac{d_o}{D}} \cdot \frac{\sqrt{D}}{\sqrt[4]{H}} = \frac{\sqrt{\dfrac{d_e \cdot d_o}{D}}}{\sqrt[4]{H}} \qquad (17b)$$

Because of the large numbers of other factors that influence the cut point, this equation cannot be used for numerical calculations, but it gives an idea of the direction in which one should continue to run tests after having obtained some preliminary results by arbitrary runs.

The real problem arises when coarse cutting cyclones are needed in the plant (for closed circuit grinding, etc.). Cutting at large particle sizes requires large diameter cyclones with consequent high capacities, but often only small amounts of material are available for the pilot plant or laboratory tests. Small test cyclones must therefore be employed, resulting—because of Eq. 17a/b—in lower cut points than desired. The limitations of such test work are very severe in terms of variations in the cyclone diameter. To fulfill the model law of Eq. 17a, we must consider, for constant cut d_c

$$\sqrt{D} \sim \sqrt[4]{H} \quad \text{or} \quad H \sim D^2 \qquad (21)$$

Fig. 9—Cyclone shapes and applications



This means that when running a test with a cyclone of half the diameter D, the pressure loss H employed should be a quarter of that of the plant. If the diameter of the original unit is 700 mm (28 in.) and it is to be run with 6-m liquid head pressure, then a test cyclone of 350-mm (14-in.) diameter requires 1.5-m pressure, which is close to the limit of air core stability. A cyclone with a diameter of 175 mm (7 in.) would require a mere 0.4-m pressure, which is not possible. Here, the corrected form of Eq. 17b is of help. If the big cyclone has smaller nozzles (e.g., a ψ-factor of about 0.05), a test cyclone with big nozzles may be used—e.g., $\psi = 0.12$, thus obtaining a conversion factor of $\sqrt{2.4} = 1.55$. This reduces the pressure factor from 1/16 to 1/10 for a test cyclone of one-fourth the full scale diameter. It must be determined whether the cyclone is stable in its operation with 0.6-m liquid head; e.g., 0.06 bar pressure. (The capacities may be as follows: 300 cu m per hr for the 700-mm-dia unit, 40 cu m per hr for 350 dia, and 5 to 12 cu m per hr for 175 dia, depending on nozzle sizes.)

There is no possibility of running a test with 3-in. or

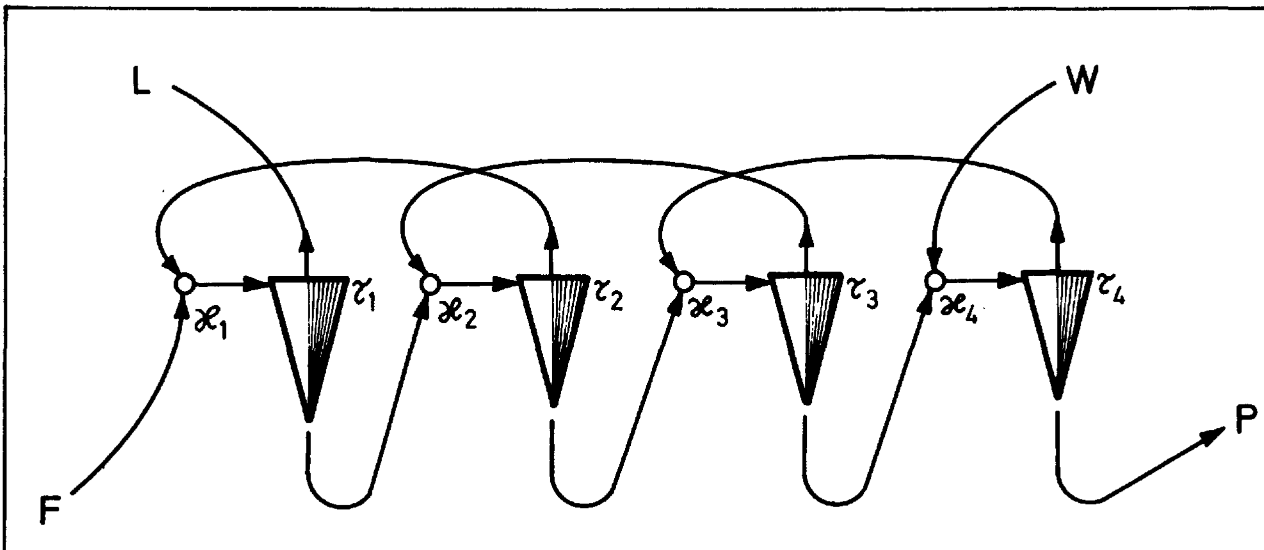


Fig. 10—Four-stage countercurrent washing (Equation 19): F = feed with lye content, P = washed product, L = residual washing liquid, W = wash water, τ = free liquid split ratio of cyclones, χ = wash liquid ratio at mix tank.

4-in.-dia cyclones—a mistake sometimes made in test institutes. For these cyclones, the equivalent pressure would be 0.1 to 0.15 m LH (equivalent to only 0.01-0.015 bar). No cyclone can function in this way. In addition, when the cyclone diameter is reduced by a factor of 8 (the cross sectional area by 64), the Reynolds Number will vary by the same factor, changing from transitional to laminar flow, and therefore disturbing any model law. In summary, satisfactory pilot plant tests can be made with a cyclone unit of half the diameter, using one-fourth of the pressure, and producing one-eighth of the throughput of a full scale plant. A reduction of 1:3 may be the optimum limit (1/10 of the pressure, approximately 1/30 of the capacity), but smaller cyclones will lead to incorrect results.

In addition to cyclone diameter, and feed and overflow nozzle diameter and pressure, the cut point is influenced by further geometric factors—e.g., the effective cyclone length $L_e = \lambda \cdot D$, which is determined by cone angle and barrel length. This factor is incorporated in

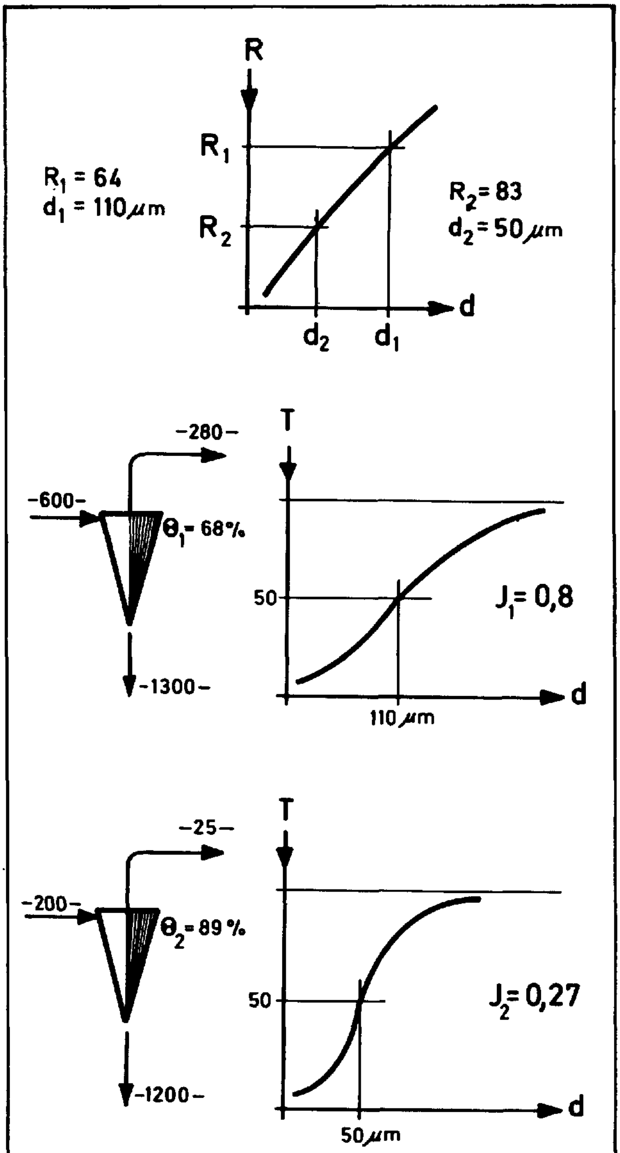


Fig. 11—Influence of feed solids content on separation mesh cut and imperfection: top, grain size distribution line (RRB); center and bottom, two Tromp curves.

Table 1—Influence of specific gravity and viscosity on cut point (separation mesh)

Solids	Liquid	ρ_s	ρ_L	η	Cutpoint factor
Sand	Water	2.6	1	1	1
Coal	Water	1.4	1	1	2
Iron ore	Water	5.0	1	1	0.63
Rock salt	Brine	2.1	1.2	6	3.26

Eq. 17a. Also important is the feed nozzle geometry. A rectangular-shaped feed duct in the cover plate, together with evolute entrance, has proved to be most satisfactory, but no data quantifying the influence of these design features are available.

The characteristics of the slurry that is fed to the cyclone also influence the cut point. The grain size distribution determines the relationship between the fractions plus and minus the separation mesh, and therefore the residual solids content in the overflow (the effective viscosity of the parent liquid). In the same way, the effective slurry viscosity, since it appears in the Stokes equation, influences the cut point. It is determined by the solids content in the feed. Higher slurry concentrations therefore generate coarser cuts than lower concentrations. This effect can also be described as hindered settling, because the movement of the coarser particles is hindered by the zone of smaller particles, through which the coarser ones must pass. The effect has also been measured in fluidized beds.[16] Fig. 11 explains the practical result of increasing separation mesh and imperfection by feeding slurries with a higher solids content.

The viscosity of the liquid itself acts in the same way. Furthermore, the difference in densities or specific gravity between the solids and the carrying liquid is important. Table 1 compares four examples, based on the assumption that the nominal cut at low solids content will be 100 microns. Again Eq. 17 should be considered —i.e., the first term of it:

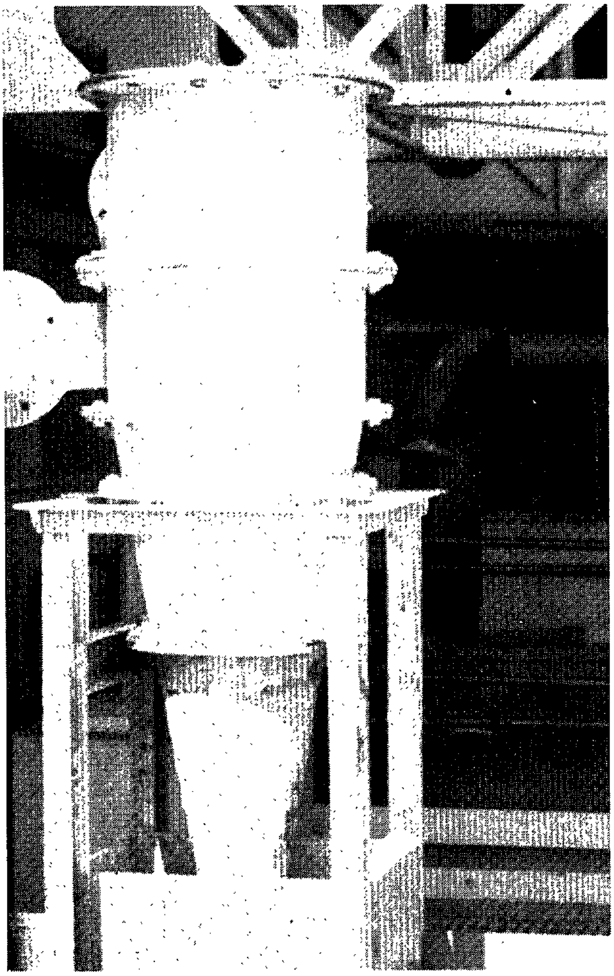
$$d_c \backsim \sqrt{\frac{\eta}{\rho_s - \rho_L}} \qquad (22a)$$

or

$$d_c = d_o \sqrt{\frac{\eta_1}{\eta_2} \cdot \frac{\rho_{so} - \rho_{Lo}}{\rho_{s1} - \rho_{L1}}} \qquad (22b)$$

The shape of the particles is also important. Very flat particles such as mica tend to go to the overflow even though they are relatively coarse. The definition of a shape factor based on specific surface is not of great help because the particle diameter itself is not sufficiently well defined. In any case, flat particles will become concentrated in the overflow. Surface-active fine particles will probably increase their diameter by hydration, and their density will decrease. Others will tend to float. Predictions of performance based solely on calculation are therefore not possible.

The separation mesh or cut point in terms of a formula such as Eq. 17 is the cut obtained inside the secondary vortex flow. Overflow and underflow of the cyclone may be influenced by outside forces that disturb the primary cut and make the simple formulas invalid. In most published work on separation mesh formulas, this point is ignored, but it is of such practical importance in the efficient operation of hydrocyclones that it



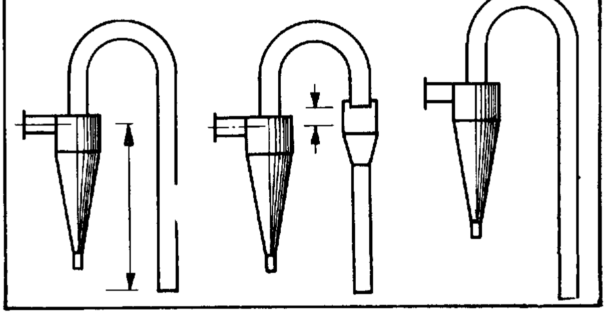
Overflow discharge chambers of good design may be beneficial, especially on bigger cyclones such as this 24-in.-dia unit.

is worthy of closer examination.

The overflow elbow of the cyclone should not be sharply curved or bent, because the rotation of the air core is continued inside the elbow.[17] Elbows of bigger radius are effective, as shown in Fig. 12. Overflow discharge chambers of good design are beneficial too, and are especially useful for bigger cyclone units. Leading the overflow from this chamber or from the hermetically closed elbow to lower levels, as shown on the left in Fig. 12, causes a siphon effect that disturbs the cut point. This is the case even when using the siphon for underflow regulation. To break the siphon effect, either the length of the elbow must be cut (Fig. 12, center), or a degassing pipe must be welded on (Fig. 12, right).

The throttling of the spigot causes other problems with cut point. High underflow concentrations are reached with a rope discharge, but in this case, some particles that have already been rejected inside the cyclone are returned to the overflow, disturbing the upper part of the Tromp curve (Fig. 13, left). A diluted underflow, called umbrella discharge, carries fine particles with the diluting water. This dead flux results in a disturbed lower part of the Tromp curve (Fig. 13, right). The imperfections of both cases are obvious. Improvement requires two cyclone stages if good degritting is required in the overflow simultaneously with good desliming of the underflow. For best results, as demonstrated in Fig. 14,

Fig. 12—Overflow siphon effect, and how to avoid it



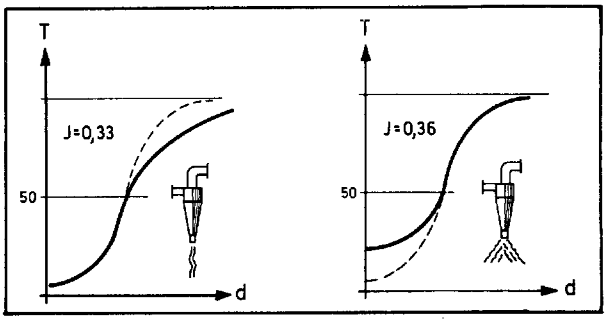


Fig. 13—Influence of underflow nozzle throttling on Tromp curve: left, rope discharge; right, umbrella discharge.

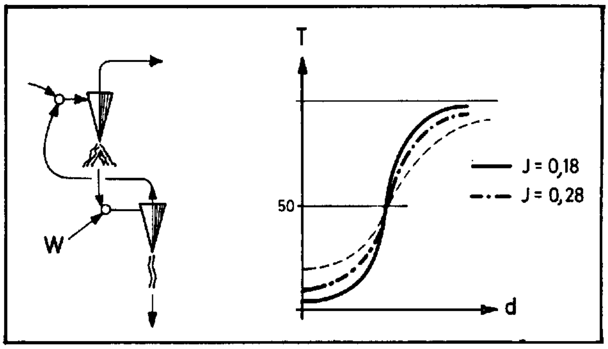


Fig. 14—Two stages in X-connection, with umbrella discharge on the first stage and rope discharge on the second stage, are applied to reach optimum Tromp curve, lowest imperfection.

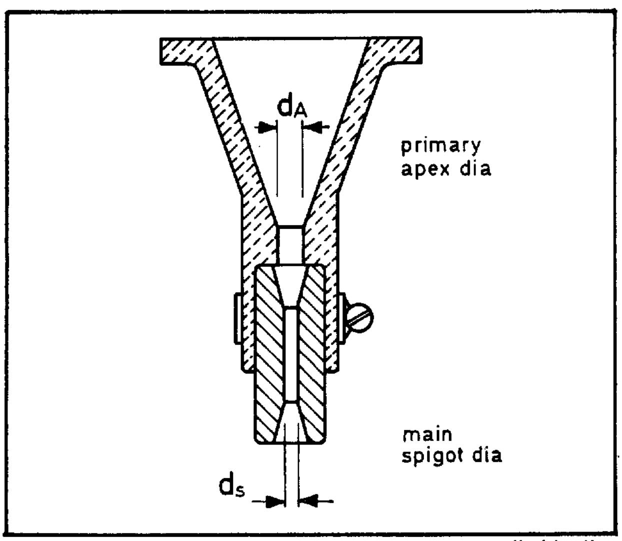
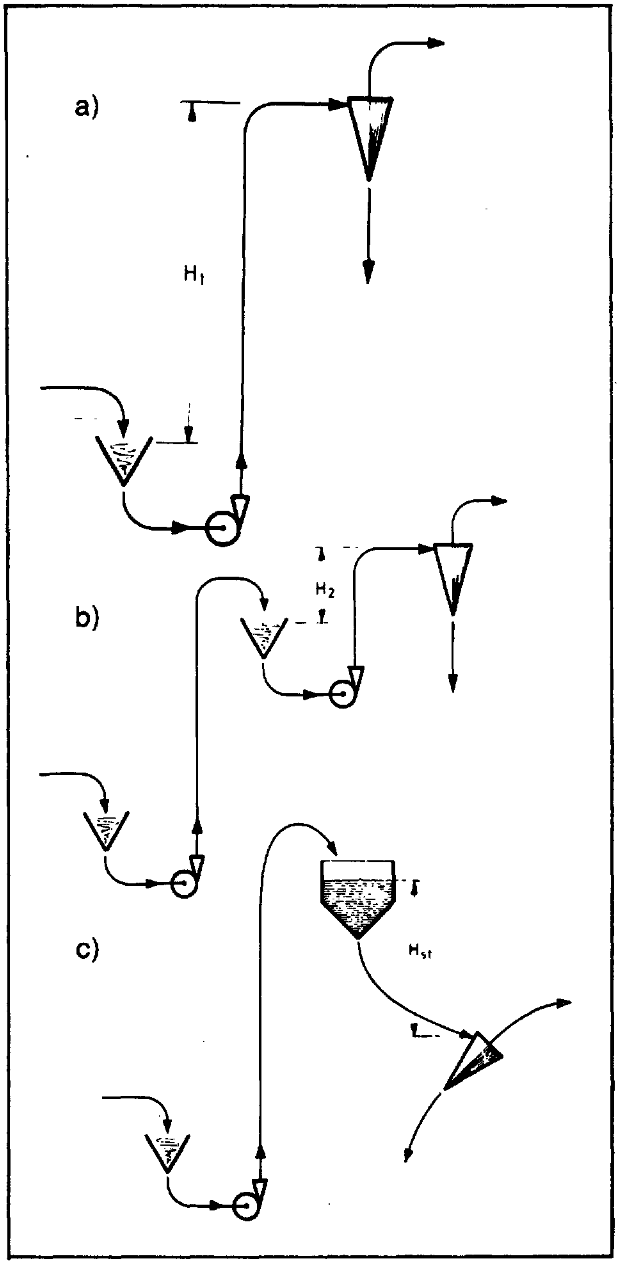


Fig. 15—Hydrocyclone spigot diameter can be controlled by the use of exchangeable apex stoppers.



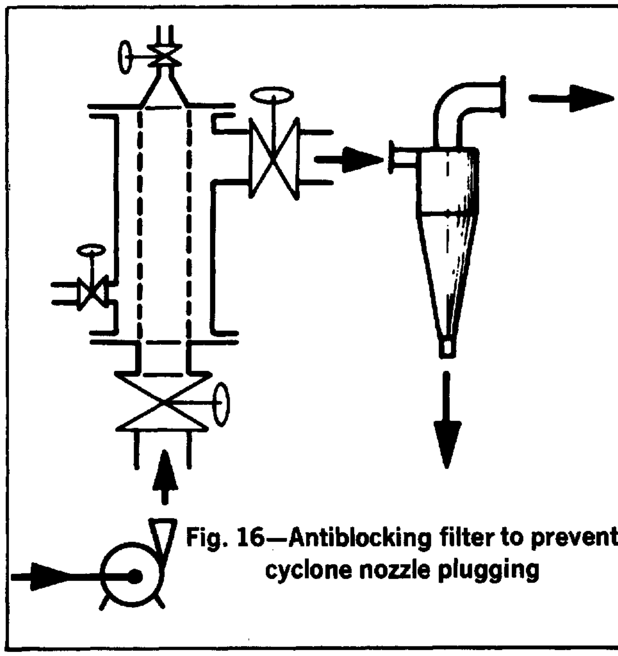


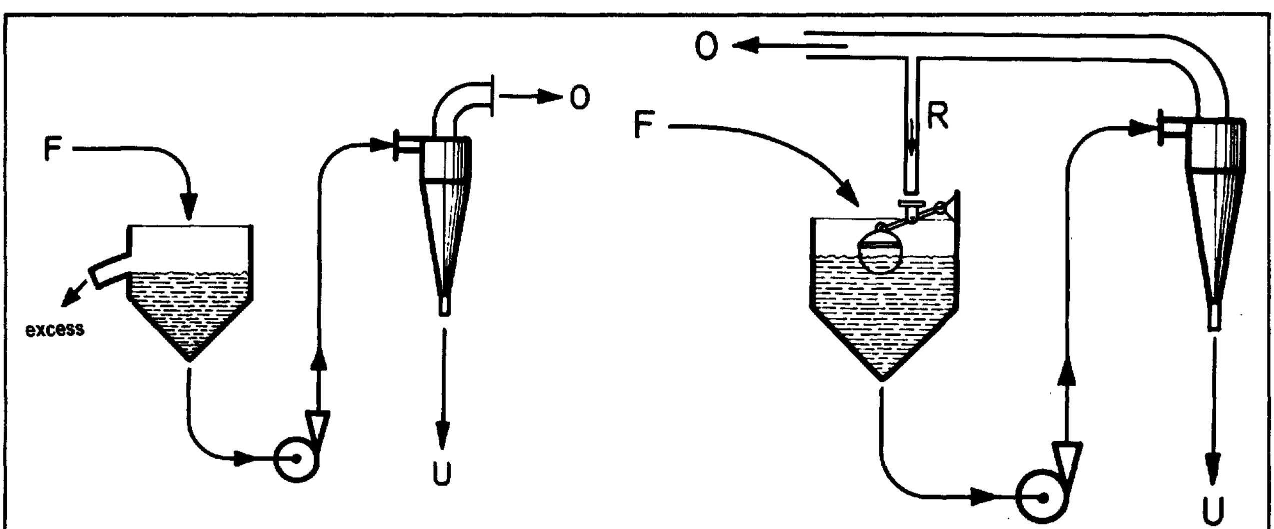
Fig. 16—Antiblocking filter to prevent cyclone nozzle plugging

the primary stage should be run with umbrella discharge and the secondary with rope discharge, with secondary overflow recycled to the main feed. This principle has been used with great success, as it also reduces the effect of fluctuating solids content and size distribution of the main feed. The result is an optimum Tromp curve with low imperfection (Fig. 14, right).

The medium Tromp curve in Fig. 14 refers to a single-stage operation where the cyclone spigot is regulated to optimize the underflow. Regulation can be done continuously by a hydraulic valve, which, of course, in-

Fig. 17 (below)—Sump level regulation: left, excess sump overflow; right, controlled partial hydrocyclone overflow recycle.

Fig. 18 (left)—Geodetic feed height and cyclone functioning: a) bad installation with too great a difference in height; b) good installation that makes use of intermediate pump; c) steady head generating gravity feed.



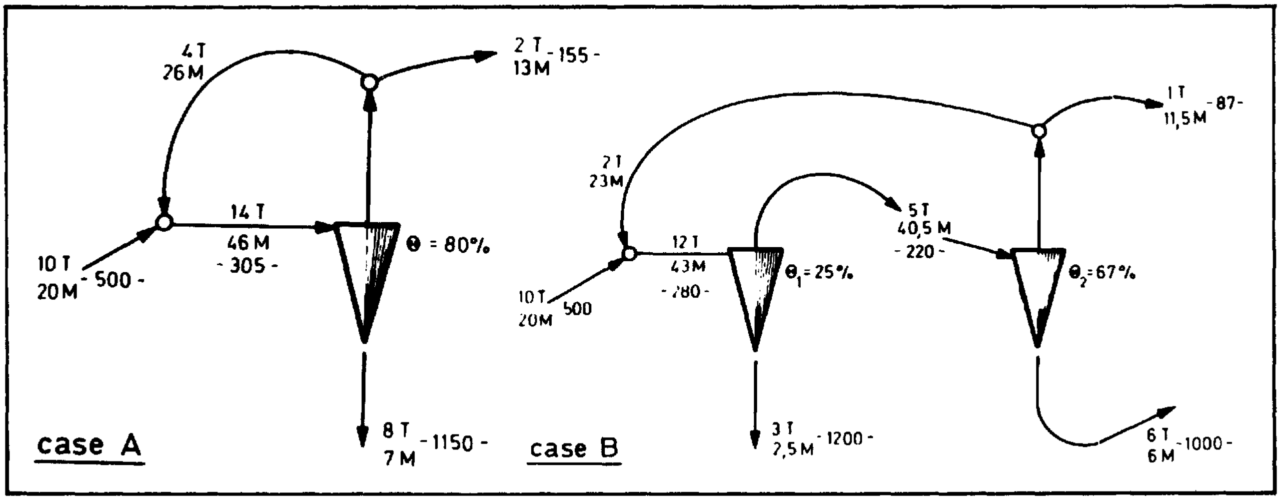


Fig. 19—Feed predilution by overflow recycle: case A, single stage installation (high mass recovery); case B, two-stage installation (low mass recovery in first stage). M = cu m per hr, T = tph, numbers between dashes = gpl.

creases capital and maintenance costs. But it can also be carried out in stages by changing apex stoppers manually, as shown in Fig. 15—a foolproof method. Such spigots with reduced diameters for higher underflow concentrations are, however, somewhat prone to plugging. To prevent plugging, the size of the biggest particles fed into the cyclone should be below one-fourth of the definite spigot diameter (in no case bigger than one-third). The anti-blocking-filter shown in Fig. 16 is a simple device to solve the problem. It is fitted with a main cleaning valve on top and with a rinsing device that will clean the perforated barrel when both the main valves are closed.

Optimum functioning of a hydrocyclone depends on constant conditions in the feed, especially the volumetric flow rate. To assure steady conditions, the level of the pump sump must be kept constant, and—above all—the level must not fall to a point where air is sucked through the pump. Fig. 17 shows two possibilities for the regulation of the feed level. In the left-hand sketch, the pump sump has a simple overflow system for the excess feed, but in many operations this principle can not be used. Much better is the principle of partial recycle of the overflow, shown on the right. The bypass line R recycling overflow suspension is closed by a simple butterfly valve connected with a float. The capacity of the cyclone is selected to be 10% or 15% bigger than the original feed F to the system.

Even a pump with controlled sump level can cause trouble—e.g., when transporting the slurry over a longer pipe system or against a somewhat greater head than that for which it was designed. Fluctuations in feed concentration can cause variation of the friction losses in the pipeline, diminishing or increasing periodically the residual pressure that remains available for dissipation in the cyclone itself. Fig. 18a demonstrates an incorrect installation, and the intermediate pump in Fig. 18b solves the problem. The height difference H_2 should be smaller than the pressure loss of the cyclone ΔH_c—if possible, only 50% of it. The safest method, of course, is to use a steady-head tank instead of the intermediate pump (Fig. 18c). This tank's level has to be controlled, which happens automatically if the tank itself is high enough.

For the optimum operation of hydrocyclones, some practical hints may be given in the form of flowsheets. If necessary, the solids content of the feed may be reduced, as shown in Fig. 11. Adding water is simple but generally impractical, as it increases the liquid load on the whole process, especially on any thickeners later in the operation. Recycling the overflow is often helpful. Fig. 19 demonstrates two possibilities. In case A, direct recycling of the overflow to the feed tank can be used where high mass recovery of the cyclone produces a dilute overflow. The recycle factor depends on the original feed concentration c_0, that of the expected overflow c_2, and the demand on maximum feed concentration c_1. The factor n of the recycling flow referred to the original feed can be calculated as follows:

$$n = \frac{c_0 - c_1}{c_1 - c_2} \tag{23}$$

In the example (case A), this results in

$$n = \frac{500 - 305}{305 - 155} = \frac{195}{150} = 1.3$$

Indeed, 26 M equals 1.3 × 20 M (in the flowsheet, M stands for cu m per hr, T for tph, and the numbers preceded and followed by dashes are gpl).

In cases of lower mass recovery, the installation of a thickening cyclone stage in the overflow of the main cyclone produces the required overflow with low solids content. Case B in Fig. 19 is an example. If c_2 is the overflow concentration of the thickening cyclone, Eq. 23 is valid for this case too. We calculate

$$n = \frac{500 - 280}{280 - 87} = \frac{220}{193} = 1.14$$

Here, 23 M recycle equals 1.14 times feed 20 M. In both cases, the regulation is done automatically by the float system shown in Fig. 17, provided that capacity of the cyclone stages has been determined correctly. The secondary cyclone stage in Case B produces a secondary underflow. This may be treated separately or remixed with the overflow, depending on the demand of the flowsheet.

Another possibility of reducing the solids content in the feed to the main cyclone stage is the installation of a scalping cyclone. Fig. 20 demonstrates an example of the refining of kaolin by hydrocyclones. The desired feed

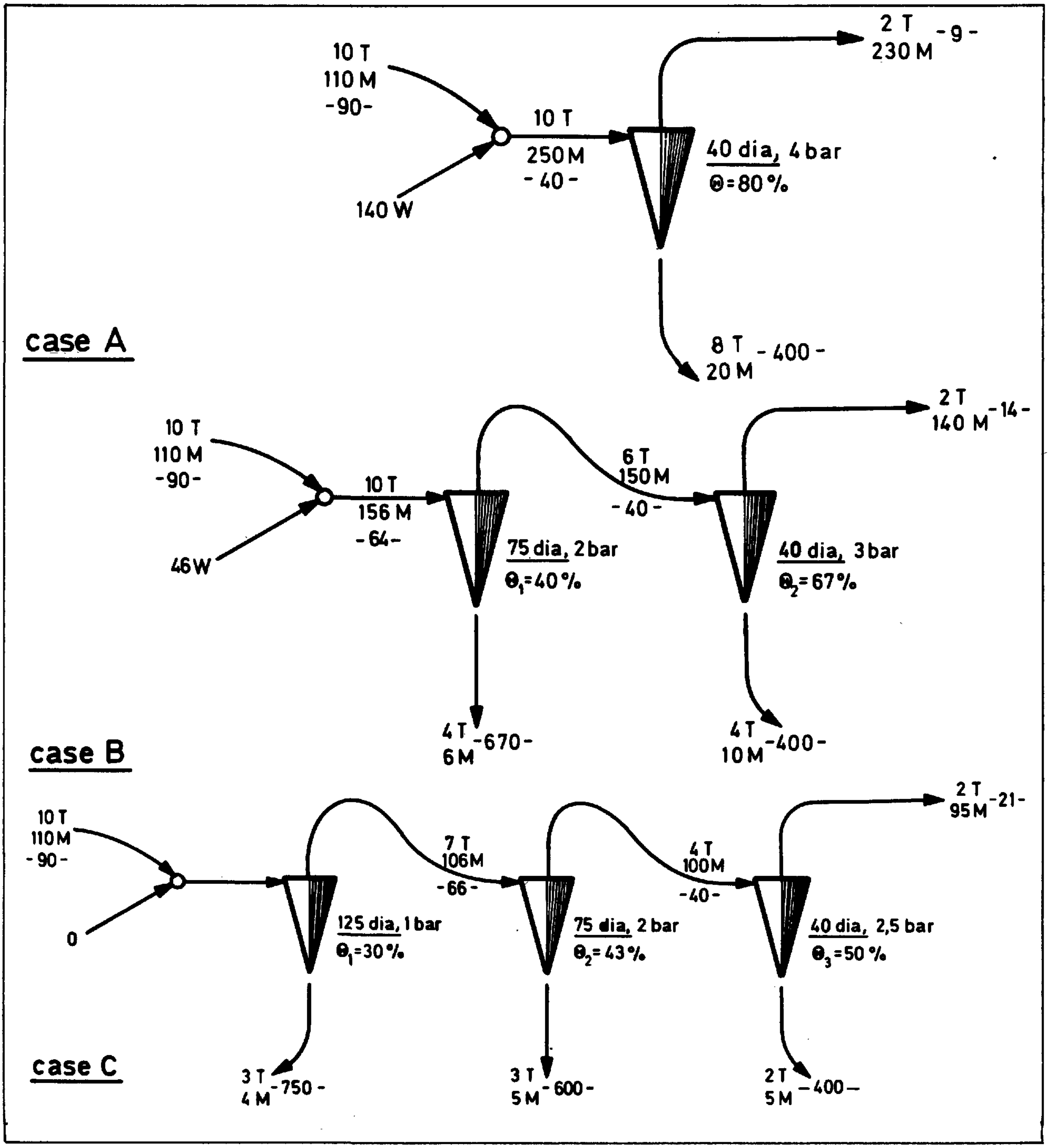


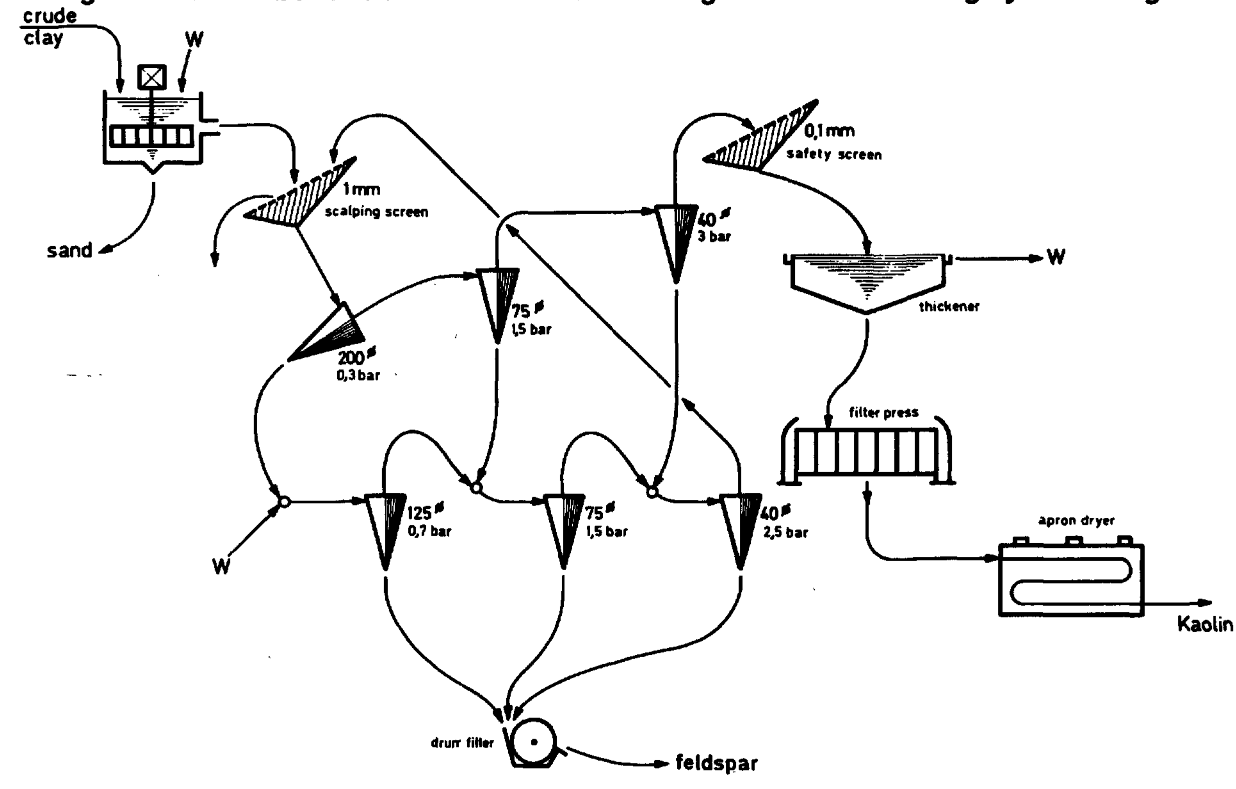
Fig. 20—Scalping-cyclones reduce feed concentration: case A, diluting water is fed into a single cyclone stage; case B, a reduction in diluting water is accomplished with a scalping stage; case C, a three-stage installation avoids dilution.

concentration is 40 gpl. The feed coming from the blunger or log washer may have 90 gpl solids content. In case A, 125% fresh water is added. When one scalping stage is installed (case B), rejecting half of the coarse fraction, the water factor can be reduced to 42%. With two scalping stages (case C), 75% of the coarse fraction will be rejected. The desired feed concentration of the main (third) stage is reached without adding water at all.

A kaolin refining plant is usually operated as in case C. In addition to the reduction of solids content without adding water, there are other reasons for installing scalping stages. One is the reduction of the number of cyclones for the most expensive, main stage (in the example, by a factor of 2.5). Altogether, 26% excess capacity must be treated. But because these cyclones are bigger, this part of the installation is cheaper. Another reason is that in case A, the single cyclone stage is fed with the complete spectrum of particle sizes. The refining effect is poor because the coarse particles hinder the fine separation by stratification. Case C, however, offers optimum conditions for good refining results. Finally, although 26% more capacity is pumped in case C, power consumption is less because the scalping stages are running at lower pressure.

Maintenance costs (wear) also must be considered. The first stage is built using bigger cyclones (125-mm

Fig. 21—Kaolin beneficiation with three refining and three washing cyclone stages



diameter) and is run at lower pressure (1 bar), generating low centrifugal forces (320 g). Therefore, there is little wear. The second stage, with 75-mm-dia cyclones and 2-bar pressure, generates 1,070 g (calculated using Eq. 15). As all coarser particles (plus 50 microns) have already been removed, there is little wear in the second stage, either. The main third stage handles only minus 25-micron particles. Therefore, even at the high acceleration of 2,500 g generated by 40-mm-dia cyclones under 2.5-bar pressure, the wear is not significant. Indeed, this is another big advantage of a three-stage refining plant.

In addition to three refining stages, a modern kaolin refining plant[8,11] incorporates three washing stages, each stage as shown in Fig. 14. The overflows from the first and second washing stages are blended with the underflows from the second and third refining stages, and the overflow of the third washing stage is recycled to the first feed (Fig. 21). The decision on optimum blending of intermediate product streams can be made by using so-called generation like numbers.[18] Ahead of the six-stage cyclone plant, a blunger or stirrer system is installed to disperse the crude clay and to discharge the sands (plus 0.5 mm). The subsequent 1-mm mesh screen eliminates fibers, wood, and leaves that come from the mine quarry. The final overflow passes through the 0.1-mm safety screen (mica screen) and then goes to the thickener. Sludge from the thickener is filtered in automatic filter presses and dried in apron-type dryers. The underflows from the three washing stages, being feldspar, are dewatered on vacuum drum filters.

In kaolin refining plants, hydrocyclones do the main job. In most other mineral treatment plants, they are auxiliary equipment, but, nevertheless, they may be important for achieving optimum results in product quality and yield. □

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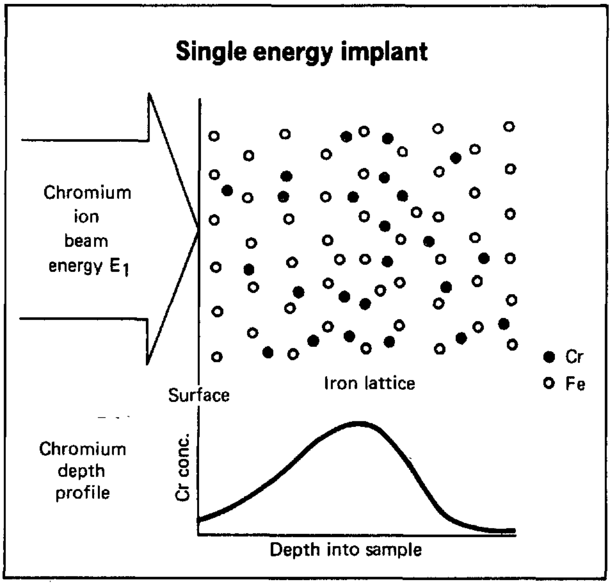


Model of iron lattice after implantation using chromium ions.

lowed by studies of corrosion behavior in gaseous and aqueous environments. It was demonstrated that the surface alloys had corrosion characteristics similar to bulk iron-chromium alloys, while chromium requirements were decreased by many orders of magnitude.

Corrosion problems tackled in geothermal brines

Problems with corrosion are also being tackled in systems for mineral recovery from geothermal brines. In this type of recovery, the tendency has been to use the ordinary, inexpensive structural materials. However, such materials corrode rapidly in contact with brines, which may contain as much as 28% salts, as well as hydrogen sulphide, carbon dioxide, and air. Furthermore, the brines come from the well at temperatures of 350° to 700°F and at pressures up to 450 psi. Another problem is the severe scaling that often occurs in brine processing systems, which may result in clogged piping systems and/or impaired heat transfer. The need to cope with scaling will require detailed knowledge of the chemical composition of the brines, and probably some phase studies.

CVD makes inroads as anti-abrasion technique

Chemical vapor deposition (CVD) of highly pure tungsten metal, pioneered in the mid-1950s by the Bureau of Mines, has not yet gained wide acceptance as a commercial process. However, research has led to extensive use of tungsten CVD in some rather exotic applications—for example, tubing and fixtures for nuclear and aerospace applications, where chemical resistance is critical, and tungsten-rhenium alloys formed as thermocouple wells for nuclear apparatus.

The USBM's Rolla, Mo., research center plans to apply a tungsten facing on steel ring seals for large ball valves used in feeding coal into gasification reactors, in experiments to overcome erosion of the valve seals.

CVD has begun to make its mark in the metals industry, which may lead to extensive commercial production. Some examples:

- A conventional sintered carbide tool insert apparently achieves greatly improved performance and life when the cutting edge has a coating of TiN, TiC, or titanium carbonitride, and toll producers are doing extensive investigations of the possibilities.
- Pyrolytic carbon, a CVD product, is being used in composite materials to meet high temperature, high strength criteria.
- There is widespread interest in CVD of nitrides and carbides of titanium and silicon for corrosion- and abrasion-resistant surfaces (bearings, valves, and tools).
- Pipe and chamber linings produced by CVD of tantalum, tungsten, and other inert metals are reportedly being developed for corrosion resistance.
- Production of Nb_3Sn and Nb_3Ge superconducting ribbon by a continuous CVD process was recently announced. Work on CVD is widespread in solid state electronics, to apply semiconductors, thermionic materials, super-conductors, and tungsten and other metal attachments to miniature electrical devices.
- The USBM research center at Rolla, Mo., will investigate the application of TiN to valve ring surfaces for abrasion resistance in coal-hopper valves. □

Reclaiming and recycling secondary metals

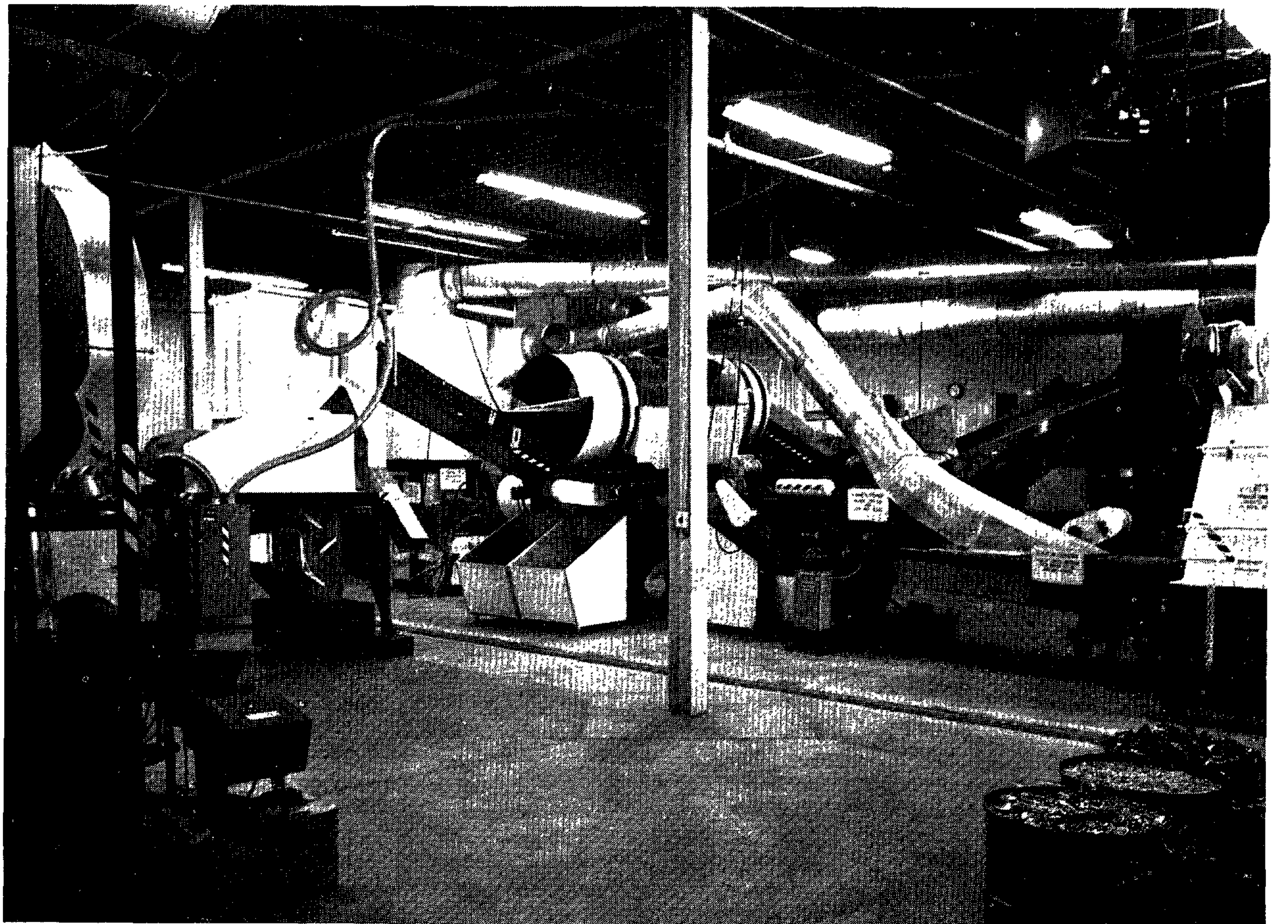
US Bureau of Mines metallurgy research staff

THE PAST TWO DECADES HAVE SEEN STEADY GROWTH in the reclamation of secondary metals from new and used scrap, smelting residues and drosses, and numerous metallic waste solutions. The US industry is now remelting and refining more than 3 million tpy of nonferrous scrap metals. More lead is recovered from scrap than from ores, more than half of US copper and steel requirements are supplied by scrap, and greater than 30% of domestic aluminum is recycled. Improvements in recycling rates will require expansions of processing facilities and innovations in processing technology.

Introduction of new technology during the past 20 years was overshadowed by expansion of plant facilities to handle the increasing volume of secondary metals to be processed. However, it is new technology that has enabled the secondary metals industry to help meet the growing needs of the US. Technologic achievements in the last 20 years include improved cleanup procedures and processes for gaseous and liquid effluents; improved methods for treating and recovering metal values from unusual fumes and byproducts; improved smelting techniques for treating fine scrap and dusts; and development of low-temperature, low-energy hydrometallurgical, pyrometallurgical, and electrometallurgical processes and equipment.

Environmental control influences new technology

The push toward new or improved technology during the 1970s has come largely from the enactment of strict environmental standards for industrial emissions. The impact of these regulations on the nonferrous industry has been great, particularly in the area of smelting. The copper industry is making a major effort to develop and commercialize hydrometallurgical processes to reduce the sulphur dioxide emissions characteristic of conventional smelting processes. This same trend is beginning to surface in the



Metals are reclaimed from urban refuse in a ½-tph pilot plant operated by the USBM in College Park, Md.

lead industry, which reprocesses thousands of tons of lead batteries annually. The US Bureau of Mines recently developed a combination hydrometallurgical and low temperature smelting process for reprocessing scrap lead from batteries without producing sulphur dioxide emissions. The process is being tested commercially by one company.

In the zinc industry, existing or proposed air pollution standards are shifting the emphasis from pyrometallurgical toward electrometallurgical processing of many scrap and waste materials.

The aluminum industry, which generates fluoride and chloride emissions, is spending millions to improve reduction and smelting operations. The Bureau of Mines has developed a hydrometallurgical method aimed at recycling chloride fluxes to reduce disposal problems, and an inert-gas fluxing method that would eliminate chloride fluxes in smelting.

Trend in steel to recover values from flue dusts

One technologic trend in the steel industry is to treat more of plant-generated byproducts such as pickle liquors to recover marketable products such as iron oxide and hydrochloric acid. The industry is also integrating into plant operations the recovery of resources from low-valued wastes—flue dusts, residues, and sludges.

Increasing quantities of flue dusts containing zinc, copper, lead, iron, and other metals are generated at steel mills, brass mills, secondary smelters, galvanizers, and other facilities. At a time when the US is rapidly becoming more dependent on foreign sources for zinc metal, 200,000 to 235,000 tpy are lost in flue dusts at domestic operations. Disposal of these dusts on land is customary, although storage sites are becoming scarce. Furthermore, the untreated wastes cause more dusting problems during transportation and storage, and they can contaminate streams and groundwaters.

In steelmaking, a significant amount of each furnace charge is converted to dust and collected in scrubbers or baghouses. Although these dusts are a potentially rich source of iron, zinc, and lead, very little of them re-enter the industrial cycle. However, the combination of stricter pollution regulations and the shortage of domestic zinc has generated widespread industrial interest in zinc recovery processes. Direct recycling of these dusts to steel furnaces is inadvisable, but a number of other processes have been proposed for separating the zinc and lead from the iron.

The Bureau of Mines studied a sulphuric acid leach, followed by pyrometallurgical methods to recover separate zinc, lead, and iron products. M. F. Munoz has investigated a sulphuric acid leach for separating the metals. Flue dusts that contain 40% or more zinc can be leached in sulphuric acid, and the impure zinc sulphate solution is used as a fertilizer additive. At least one company has converted zinc-bearing steel furnace flue dust into a frit that is subsequently crushed and added to fertilizer for soils deficient in zinc and iron. Mineral beneficiation methods, however, have not been effective.

M. D. Holowaty and the Bureau of Mines have recently studied reduction-volatilization methods for separating the metals. Holowaty suggested a large regional plant to serve several steel companies, which would greatly improve the economics of the process. However, his proposal has not been accepted by the industry.

In Japan, Kawasaki Steel Corp. recently brought on stream a plant for treating 40,000 tpm of flue dust and sludge, recycling the iron to the steelmaking process. In another project, 25 ferrous and nonferrous producers organized Sohtetsu Metal Co. to process 60,000 tpy of steelmaking flue dust.

Utilization of flue dust from brass smelters depends to a large extent on the availability of zinc. When zinc is readily available, metal recovery from dust is economically noncompetitive. The Bureau of Mines has developed both hydrometallurgical and pyrometallurgical processes for recovering zinc, copper, lead, and lead-tin from brass smelter dusts. A substantial fraction of brass and bronze flue dusts is consumed in production of pigments and zinc chemicals and for agricultural purposes.

In new steel technology, the Bureau of Mines has done extensive research on the smelting and chemistry of rebar metal made from incinerated can scrap recovered from municipal waste. A sodium sulphate treatment has been developed to remove or control the copper content in smelting ferrous scrap.

Steady growth for electroslag refining

Developed nearly 40 years ago, the electroslag process is a secondary technique in which a consumable electrode is melted through an appropriate flux. Applications of the process grew slowly but steadily through 1968, then expanded rapidly. Reluctance to accept the process, now used chiefly for specialty steels and superalloys, has hinged on difficulties in evaporative removal of impurities, extensive capital investments already made in proven conventional secondary melting techniques, and general unavailability of reliable cost comparisons.

However, plants employing the electroslag process have increased in number from 20 to 85 since 1968, not including installations in the USSR. This recent, relatively rapid growth in electroslag installations is attributed to their potential for extensive refining reactions, improved ingot surfaces, improved cleanliness, lack of segregation, greater yields, use of inexpensive electrical power, and the relative ease with which ingot shapes can be prepared.

Electroslag melting appears to be demonstrating the viability of producing larger ingots at reasonable costs. While it is considered unlikely that the electroslag process will replace vacuum-arc melting, especially in the US, there is no reason to doubt that the two processes will effectively complement each other in the future. Considerable growth of the electroslag process is anticipated as new secondary melting facilities are constructed and producers become better acquainted with the unique capabilities of the process.

Removing metals from electroplating wastes

Removal of metals and cyanides from electroplating wastes via numerous procedures is summarized in reports of the Environmental Protection Agency (EPA).

One method for destroying cyanides in water, developed at Karlesruhe University in Germany, involves heating the solution to 160° to 200°C, causing the cyanide to decompose into ammonia and a salt of formic acid. Solid cyanide wastes, such as the cyanide-chloride mixtures used in case-hardening of steel, can be treated with steam at 600° to 700°C. The products are carbon monoxide, ammonia, sodium carbonate, and hydrogen.

The Cyanil Co. of Kitchener, Ont., has developed an electrochemical method for destroying cyanides in solution. No chemicals are used, and the process is claimed to be less expensive than conventional methods, such as treatment with chlorine or hypochlorite. As an alternative approach, a cyanide-free zinc plating process has been developed and tested.

In a new Bureau of Mines process for recycling metals and cyanides in electroplating wastes, the cyanides can either be recovered as sodium cyanide for recycling, or burned off. One electroplating plant is already using this process, a second plant has requested permission from state pollution control authorities to start using it, and other plants are considering integrating the process into their waste treatment systems to reduce reagent consumption.

Copper, zinc taken from etching solutions

Recovery of copper and zinc values from spent chromium etching solutions has been given very little attention. Solutions of chromic acid and sulphuric acid are used to etch and brighten brass and zinc parts and to etch copper from printed circuits. The etching of brass dissolves copper and zinc and reduces some of the hexavalent chromium to trivalent chromium. The spent solutions are usually discarded.

In a Bureau of Mines process, hexavalent chromium is reduced to trivalent chromium by a low cost reagent such as waste paper, methyl alcohol, or formaldehyde, and the metal values are recovered by a variety of methods.

Electrochemical machining sludge recovered

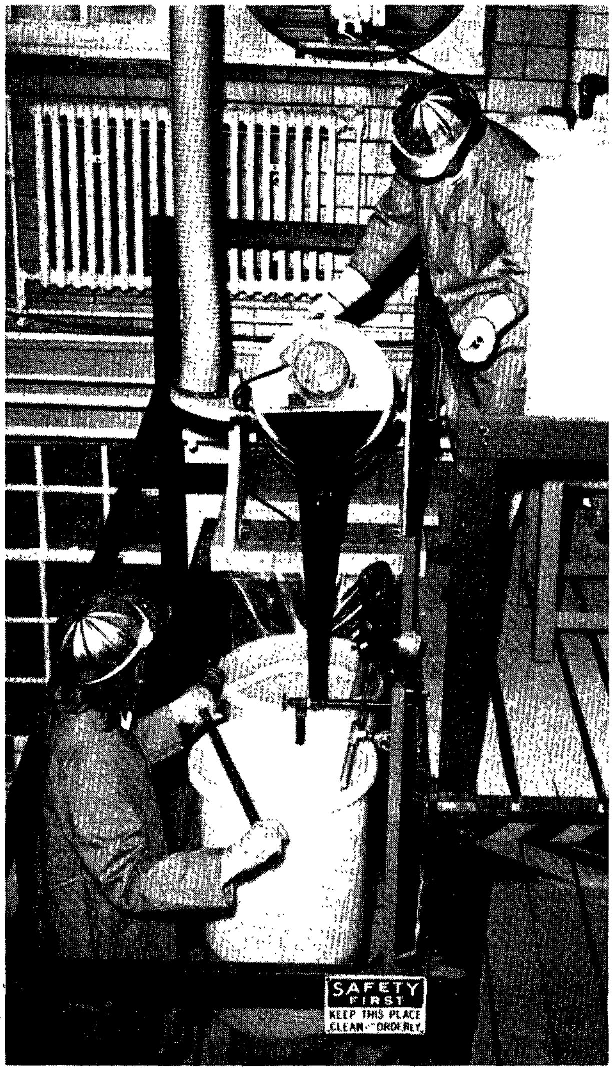
Very little work has been done on recovery of electrochemical machining sludge wastes despite the magnitude of the waste disposal problem. These wastes result from the machining of nickel-base and cobalt-base alloys used in the manufacture of aircraft engines. The wastes consist of very finely divided metal hydroxide particles in an aqueous electrolyte solution, usually containing sodium chloride or sodium nitrate.

Filtration of the sludge to separate the electrolyte solution and the metal hydroxides is an obvious approach, but the filtration rate is much too slow to be practical. The Bureau of Mines has developed a recovery process whereby the sludge is first evaporated to dryness to agglomerate the metal hydroxide particles, and water is then added to redissolve the electrolyte. After this treatment, the filtration rate is 20 to 120 times faster than that of the original sludge. The slurry is filtered to recover the electrolyte solution for recycling to the machining operation, and the metal hydroxides are heated to convert them to oxides, which are subsequently reduced to metal.

Metallurgical wastes recycled from foundries

Improving the efficiency of domestic foundries by recycling waste products and finding domestic substitutes for imports such as chromite and zircon has been studied by the USBM.

One phase of the study, recently concluded, involved development of methods to recover brass and molding sand suitable for recycling from phenolic-bonded brass foundry waste. The material tested—a combination of phenolic-bonded shell mold and residue from deburring operations—was composed of 3% copper as a high-copper brass, 2% phenolic resin, 3% iron oxide, and about 90% molding sand. Brass was recovered by a combination of screening and gravity concentration on a standard concentrating table. The final concentrate recovered 75% of the brass copper at a grade of 45.5% copper. Eighty-three per-



Hydrometallurgical process for treating aluminum oxide slags is among the USBM recycling projects now in the research stage.

cent of the original waste was then reclaimed as sand by burning in a rotary kiln. Foundry evaluation of brass and sand showed both to be suitable for recycling.

Extracting copper from blast furnace slag

The Bureau of Mines has also studied the feasibility of recovering a recyclable copper product from granulated blast furnace slag from Southwire Corp. The slag contained 5-7% total copper and 3-4.5% metallic copper. Bench scale tests showed that tabling of minus 35-mesh material recovered 57% of the total and 83% of the metallic copper in a concentrate that graded 47% copper. Flotation of material ground through 65 mesh with a dixanthogen collector produced a concentrate analyzing 40% copper, with recovery of 82% of total Cu and more than 98% of the metallic copper. Under a cooperative program with Southwire, small-scale continuous testing emphasized only gravity separation techniques. A 40-hr test was made with a circuit that included rod mill grinding and Humphrey's spirals and shaking tables. A coarse (plus 35 mesh) and a fine concentrate analyzed 68% and 72.5% copper, respectively; the composite analysis was 68.6% copper. The composite concentrate recovered 60% of the total and 88% of the metallic copper.

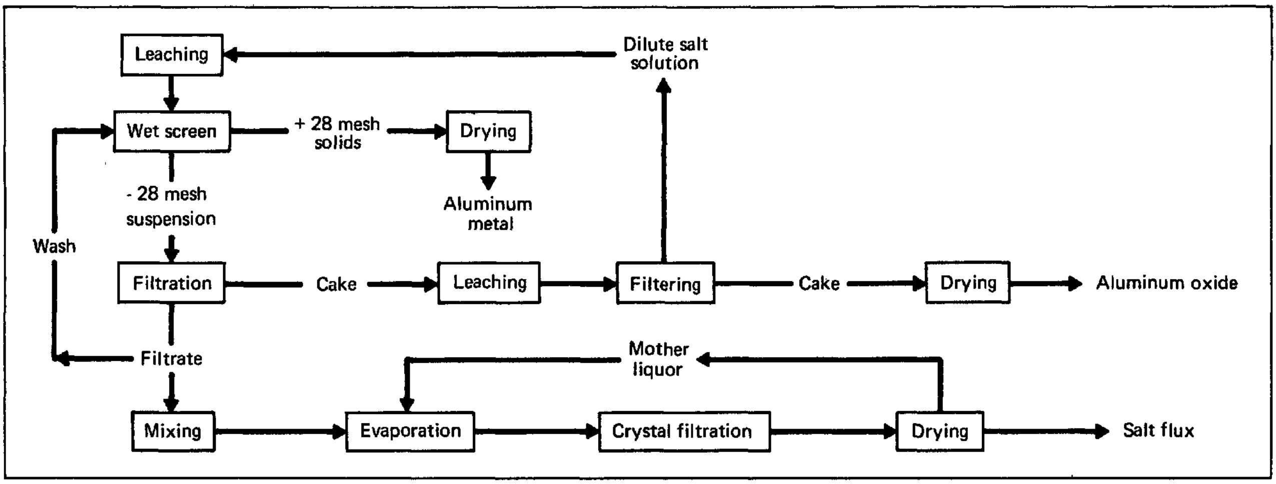
Zircon reclaimed from waste foundry molds

The advent of greatly increased zircon prices has generated interest by the casting industries in reclaiming zircon from waste foundry molds. The Bureau of Mines has been studying the recoverability of zircon flour from spent investment casting molds in cooperation with Sherwood Refractories Inc. The USBM has also been working with Esco Corp. to reclaim zircon and sand from spent molds.

Recycling resources from urban refuse

The Bureau of Mines has been very active in development of continuous mechanical systems for separating materials from mixed refuse, and the agency was the first to demonstrate that the key to solving the problem was the application of minerals processing technology. For several years the USBM's College Park Metallurgy Research Center in Maryland has been operating a ½-tph pilot plant for reclaiming metals and minerals from municipal incinerator residues. The plant is assembled entirely of conventional minerals engineering equipment. It uses continuous screening, crushing, grinding, magnetic separation, gravity concentration, and froth flotation to produce metallic iron

USBM mini-plant flowsheet for recovering products from high-salt aluminum slag



concentrates, high-quality aluminum, clean heavy nonferrous metal composites, pure glass, fine carbonaceous ash, and slag tailings. Ground breaking for a commercial-size plant based on this process was scheduled for early 1975 in Lowell, Mass., by the contractor, Raytheon Service Co. This demonstration project is being funded by EPA, the State of Massachusetts, and the city of Lowell.

More recently, the Bureau of Mines has been operating a 5-tph pilot plant for continuous separation of materials contained in raw refuse. This patented system utilizes multistage processing in a variety of unit operations, including shredding, air classification, magnetic separation, screening, gravity concentration, electrostatic separation, and froth flotation. The system recovers a light-gauge iron fraction that includes tin cans, massive iron, massive heavy nonferrous metals, aluminum, pure glass, and combustibles in a form suitable for use as fuel. All processing is conducted on dry material except for the glass recovery section, which employs jigging and froth flotation.

Bureau of Mines technology is the basis of a commercial-size plant to treat 400 tpd of refuse in Baltimore County, Md., for which ground was broken in August 1974. Teledyne National is the contractor for this project, with funding provided by Baltimore County and Maryland Environmental Services. A 2,000-tpd, $25 million plant to be built in Monroe County, N.Y., by Raytheon Service Co. will also use the patented USBM process.

Even though prices for secondary materials declined markedly during the latter part of 1974 from early record peaks, markets are highly encouraging for products reclaimed from both incinerator residues and raw refuse. Many potential customers have evaluated the products recovered in both of the USBM pilot plants, reporting that these materials are perfectly acceptable.

Two major detinners have processed the ferrous fractions from raw refuse, each recovering 6.5 lb of tin per ton and steel scrap equivalent to a No. 1 bundle containing less than 0.03% tin. Major secondary aluminum smelters consider products from both plants to be of very high quality. Mixed nonferrous metals are chemically similar to automobile radiator scrap and presumably could enter the same recycling channels, although verification will require tests using large tonnages of these products, which are not yet available.

Processing junk autos for metal values

In the scrap processing industry, the shredder has replaced the baler and the shear as the major producer of prepared scrap from junk automobiles. Shredder installations are now being equipped with efficient dust control systems, and the trend is toward increasingly heavy construction for all parts of the shredder auxiliary systems to prevent damage from explosions, which still occur at intervals. Another trend is toward shredding of the entire automobile, including engine, transmission, gas tank (punctured), and seats. Even such items as tires, batteries, and radiators are run through the shredder if they are still in place on a junked car.

Scrap processors are becoming more aware of the potential value of the nonferrous components of junk automobiles. Only a few years ago, most of the nonmagnetic fraction of a shredded car was disposed of in landfill areas after hand-picking to recover some of the copper, zinc, and aluminum.

The Bureau of Mines has developed a system involving air classification and water elutriation for recovering a mixed nonferrous metal concentrate from auto shredder non-magnetic rejects. The system is used by some shredder operators. In addition, several large commercial plants are operating on rejects shipped in from surrounding shredders. Most of these plants use processes similar to those developed by the USBM, with the addition of heavy media or other systems to separate the mixed nonferrous metals.

One steadily mounting problem for shredder operators is disposal of the nonmetallic portion of shredder residues. At present, all of this material is disposed of by landfill. Disposal costs average about $5 per ton and are rising rapidly as hauling and labor costs increase and available landfill areas become scarcer.

In an effort to recover still more of the constituents of junk automobiles, the Bureau of Mines has entered into cooperative agreements with Ford Motor Co. and General Motors Corp. to investigate recovery of plastics from shredder residues. The USBM will develop physical recovery methods, and the auto companies will furnish raw materials and evaluate the recovered plastics for recycling potential. □

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