CHAPTER 12.7

Copper Mineral Processing

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Copper is a metal that exhibits the highest electrical conductivity besides silver. It is regarded as a relatively noble metal. Because of its electrical conductivity, copper is used extensively in the electrical industry for wiring, motors, and cabling. It is also used in plumbing, industrial machinery, and construction materials because of its durability, machinability, corrosion resistance, and ability to be cast with high precision and tolerances.

MINERAL PROCESSING IN COPPER PRODUCTION

This chapter focuses on the use of mineral processing extraction techniques to recover copper minerals from ores. Details of copper extraction by hydrometallurgy and pyrometallurgy are provided elsewhere in the handbook. Overwhelmingly, the mineral processing of copper ores refers to the use of crushing, grinding, and froth flotation to recover copper sulfide minerals from ores to concentrates. A few plants use copper oxide ore flotation, but these are not discussed in this chapter. A typical copper sulfide concentrator flow sheet is shown in Figure 1.

Since large semiautogenous grinding (SAG) mills were popularized in copper concentrators in the 1980s and 1990s, and with the treatment of lower-grade deposits, SAG mills have prevailed and are included in many of the concentrator flow sheets with fewer larger pieces of equipment being employed. In addition, larger gyratory crushers, cone crushers, ball mills, and circular tank flotation machines have also been included as a feature of these flow sheets, allowing copper concentrators to employ fewer pieces of equipment and at lower overall operating costs.

Several of concentrators built before the 1990s often had different approaches for the comminution and flotation circuits. Since then, the complexity and variation in flow-sheet design and configuration has been significantly reduced. Except where unique mineralogical or metallurgical considerations exist, most copper flotation concentrators use a standard approach, like that shown in Figure 2.

Emerging comminution technologies such as high-pressure grinding rolls (HPGRs) and new-generation flotation cells are challenging this norm; however, most deviations from this arrangement are usually due to complications in metallurgy related to ore mineralogy, including

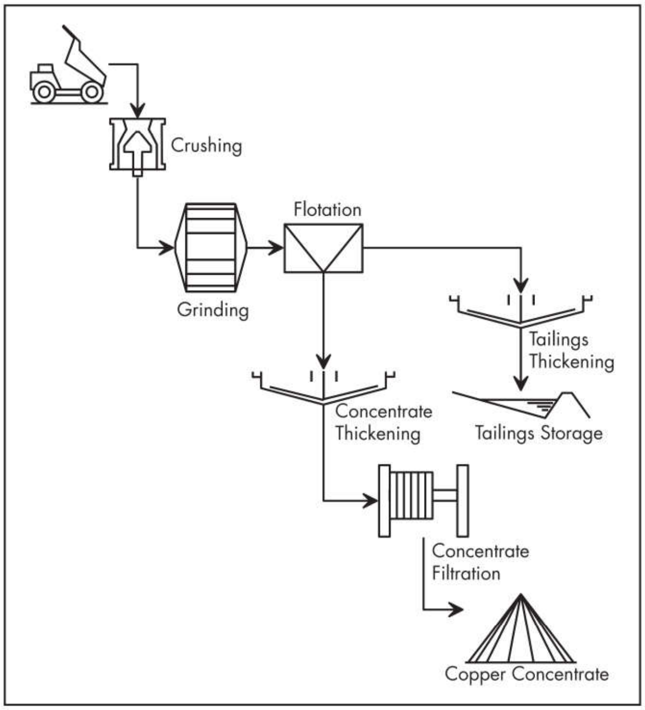

Figure 1 Typical copper sulfide concentrator arrangement

- Oxidation and alterations of copper or gangue minerals;
- Recovery of credit elements such as gold, zinc, or molybdenum;
- Management of penalty species such as arsenic, bismuth, zinc, or talc; and
- Significantly harder ores requiring different comminution approaches (e.g., Bond ball mill work index >20 kW·h/t and low JK drop weight test A×b parameters)

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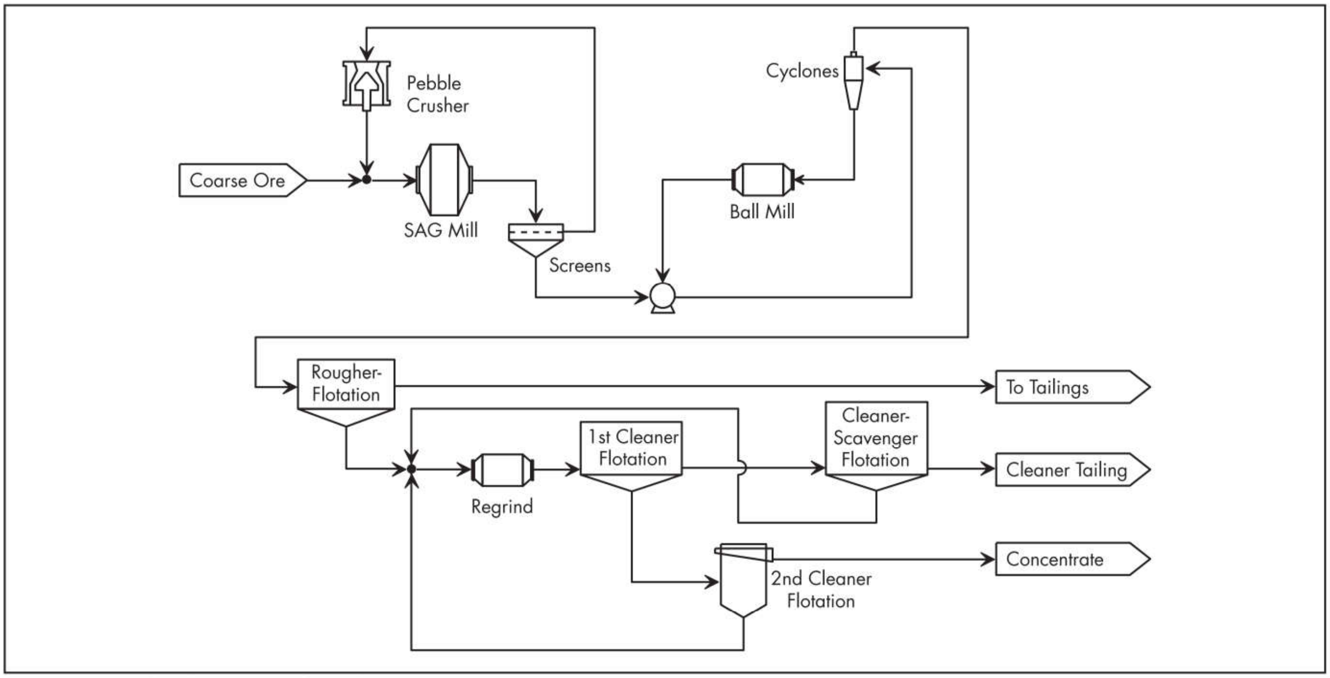


Figure 2 Standard recovery flow sheet

SUSTAINABILITY OF COPPER PRODUCTION

In 2016, 70% of the copper produced in Chile was by flotation to concentrates. The Comisión Chilena del Cobre, or Cochilco, projects that percentage will increase to more than 90% by 2028 (Cochilco 2017). Oxide ores tend to be found near the surface in arid areas and have bright blue and green mineral colors. Consequently, they are easier to find than the deeper sulfide ores. Over the last 30 years, they have been cheaper to exploit via the low-cost heap leaching–solvent extraction and electrowinning process.

Very few new large sulfide deposits have been found since the 1990s. Most of the new copper ore reserves have come from brownfields exploration of existing deposits. Therefore, the deposits with higher grades, amenable infrastructure, social and environmental factors, and high-quality concentrates are already being mined and processed.

Since copper demand is projected to continue to increase, the existing operations have been compensating by increasing throughput, but there has been scant technological innovation to reduce specific energy consumption and water consumption or to improve the metallurgical response of refractory ores.

The Copper Alliance, whose members accounted for about half of the copper in ore mined in 2015, reports that while the mass of copper ore treated has been increasing, the carbon dioxide (CO_2) emissions have also increased from 3.4 to 4.1 (t CO_2/t Cu), a 20% increase, between 2011 and 2015. While worker safety increased, other sustainability measures fell. Water consumption increased, and the total number of workers employed decreased (ICA 2018). Longer-term issues facing the mining industry that could impact copper production in the future include

- Fewer large resource discoveries,
- Lower copper grades,
- Harder ores with complex mineralogy,
- Increasing levels of hazardous elements in copper ores,
- More-stringent environmental regulations,
- Increasing anti-mining sentiment,
- More-stringent tailings disposal regulations,
- Less access to water for mining activities,
- More-stringent quality requirements for copper concentrates, and
- Shortage of experienced skill works in the mining industry.

MINERALOGY AND GEOLOGY OF COPPER ORE DEPOSITS

Copper ores constitute a large and geologically diverse group of ores, including some of the world's largest known ore bodies. Many are dominated by copper as the primary metal of production, others have co-product or by-product gold or molybdenum, while others are part of polymetallic copper-zinc-lead ores.

Mineralogy of Copper Ores

The mineralogy of copper ores is summarized in Table 1. Primary (hypogene) copper ores usually contain only chalcopyrite, and sometimes bornite, as the main copper minerals. These are accompanied by variable amounts of iron sulfide, iron oxide, or copper-iron-arsenic sulfide minerals. Native gold is present in some ores, but gold is often found in solution in the sulfides. Primary ores are often overlain by oxide and supergene zones, which may be of economic importance because of higher enriched copper grades. The mineralogy of these oxide and supergene zones can be complex, both in mineralogical assemblage and in copper mineral textures. Common copper ore minerals are listed in Table 2.

Classification of Ore Types

Given that the first ores to be sent to a concentrator from the mine are near the surface, the early financial success during start-up of a new concentrator often depends on the correct classification of oxide, mixed, and supergene ore zones. These zones have often undergone phyllic and argillic alterations, which can further complicate flotation performance due to viscosity and surface coatings caused by clays. Different techniques can be used to classify drill core and samples into these categories, including the following:

1. **Visual inspection.** Gossan rocks have brown colors due to clays and iron oxides such as goethite and jarosite, which may indicate oxide zones. Care must be taken when logging copper mineralogy visually because the bright blue and green copper oxide minerals on rock surfaces may appear more abundant than what is actually inside the ore. Sulfide minerals may be inside the rock fragments and may be harder to see with the naked eye.

Table 1 Mineralogy of copper deposits

Ore Zone	Mineralogy
Oxide	Cuprite, native copper, native gold, malachite, azurite, goethite, hematite
Supergene	Chalcocite, digenite, covellite, native gold
Hypogene	Pyrite, pyrrhotite, chalcopyrite, bornite, enargite, tetrahedrite, molybdenite, magnetite, hematite, native gold

Table 2 Common copper ore minerals

Name	Specific Gravity, g/cm³	Molecular Formula	% Cu	Cu/S Mass Ratio	Hardness
Chalcocite	5.6–5.8	Cu_2S	79.9	3.96	2.5–3.0
Covellite	4.6	CuS	66.5	1.98	1.5–2.0
Chalcopyrite	4.1–4.3	$CuFeS_2$	34.6	0.99	3.5–4.0
Bornite	4.9–5.1	Cu_5FeS_4	63.3	2.48	3.0–3.3
Digenite	5.6	Cu_9S_5	78.1	3.57	2.5–3.0
Tennantite	4.6–4.7	$Cu_{12}As_4S_{13}$	51.6	1.83	3.5–4.5
Enargite	4.4	Cu_3AsS_4	48.4	1.49	3.0
Tetrahedrite	4.6–5.2	$Cu_{12}Sb_4S_{13}$	45.8	1.83	3.5–4.0
Stannoidite	4.3	$Cu_8Fe_2ZnSn_2S_{12}$	38.3	1.32	4.0
Mawsonite	4.7	$Cu_6Fe_2SnS_8$	43.9	1.49	3.5–4.0
Wittichenite	6.3–6.7	Cu_3BiS_3	38.4	1.98	2.0–3.0
Djurleite	4.2	$Cu_{31}S_{16}$	79.6	3.84	2.3–3.0
Cubanite	4.7	$Cu_2S{\cdot}Fe_4S_5$	23.4	0.66	3.5–4.0
Luzonite	4.4	Cu_3AsS_4	58.8	1.98	3.5

2. **Sequential copper assay.** This is discussed in more detail in the "Sequential Leach Analysis" section that follows.
3. **Geochemical indicators:**
    - Cobalt tends to report to the pyrite in copper sulfide deposits. Where the ore and pyrite has been oxidized, the cobalt will be leached away. Reviewing the cobalt grade down a drill hole can indicate the oxidation profile and grade of pyrite.
    - S/(Fe+Cu) ratio. Copper, iron, and sulfur grades cannot be used alone because copper and gangue mineralogy can overlap, so these grades alone do not indicate copper mineralogy or extent of oxidation. The S/(Fe+Cu) ratio, however, can show contrast between sulfide areas with high values and oxide areas where the sulfur has been depleted by oxidation. Where pyrite is abundant, as compared to copper, a simplified S/Fe ratio can work just as well.

Sequential Leach Analysis

Sequential leach analysis is a geochemical procedure where a small mass of pulverized sample is leached in a series of different, increasingly aggressive lixiviants. After each leach, the solution is assayed for copper. The final residue is digested with strong acids to determine the remainder, or residual, copper. The results can indicate the proportion of different copper minerals in the sample.

A common regime is sulfuric acid–soluble copper (CuAS), sodium cyanide–soluble copper (CuCN), and residual copper (CuR). Other regimes can include weaker acids such as citric or acetic acid to improve contrast between oxide copper minerals and enriched copper minerals, which are partially soluble in sulfuric acid. Because of the different procedures and varying reagents, leach times, temperatures, redox modifiers, and sample preparation, care should be taken when comparing results from different sources.

The proportion of the sequential copper assays can be plotted on a ternary diagram (Figure 3). The exact proportion

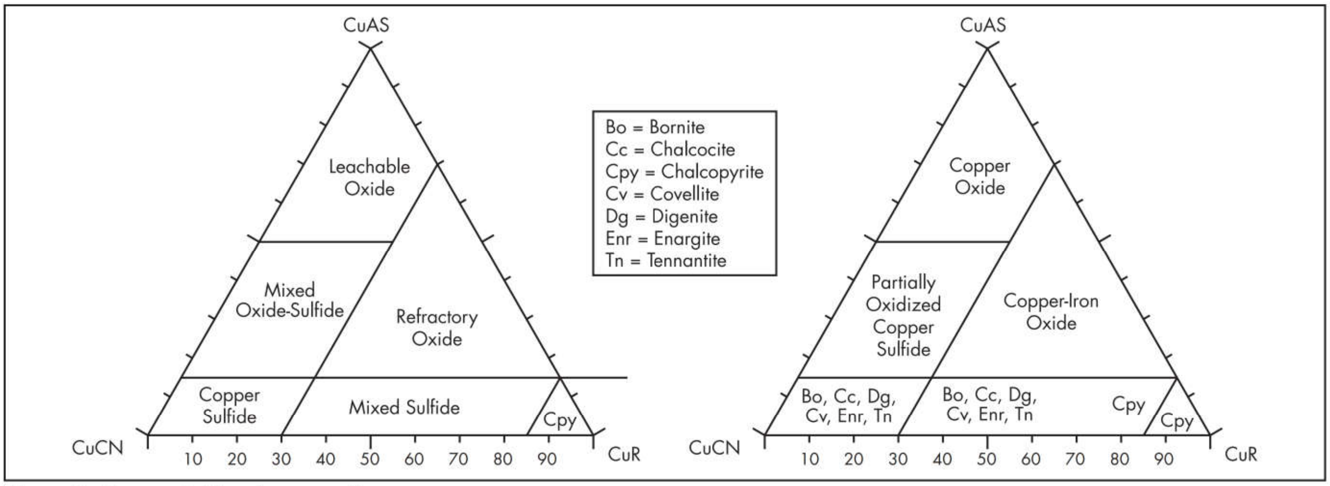


Figure 3 Sequential leach assay diagrams

of each component in each ore type varies by deposit and by sequential leach procedure.

Given the many pitfalls of using sequential leach procedures, the user should pay special attention to the following:

- **Fresh samples.** Old or oxidized samples will show erroneously high acid-soluble values. For fine samples, a low-drying oven temperature should be used to avoid copper sulfide oxidation in the samples.
- **Standard procedure.** The same method should be used for exploration assays and metallurgical sample assays. This is important in the geometallurgical analysis that aligns metallurgical test-work results with the resource block model.
- **Caution with metallurgical products.** Tailing samples will often be ground and dried in a hot oven before assay, partially roasting the sample. Consequently, a sequential assay will show erroneously high acid-soluble values.
- **Oxidative leach solutions.** Enriched copper sulfide minerals such as chalcocite will partially leach in weak sulfuric acid solutions that also contain oxidants such as ferric or chloride ions. Samples that show less than 30% of the copper present as acid soluble could be sulfide samples with no oxides present.
- **Normalize with total copper assays.** The sequential copper assays are less accurate than primary metal assays. Therefore, it is common to normalize the sequential assay by the ratio of the components rather than using the absolute sequential assay grade.

Therefore, the solubilities of single copper minerals in acid and cyanide (Tables 3 and 4) are indicative but not accurate for mine samples.

Chalcopyrite

Chalcopyrite (cpy) is a relatively unreactive copper iron sulfide that is usually associated with pyrite (py). Porphyry hypogene copper mineralogy is almost exclusively chalcopyrite. Chalcopyrite flotation is sensitive to oxidation potential, as it needs to oxidize slightly to be collected by xanthate collectors. High-pyrite deposits (>5% py) can have high oxygen consumption, requiring the use of other collectors, such as dithiophosphate or thionocarbamate (Figure 4).

Bornite

Bornite is a high-grade copper iron sulfide mineral that can be floated with xanthate collectors at lower pH, and with dithiophosphates or thionocarbamates, to form very-high-grade copper concentrates. Small amounts of sodium sulfide (100–200 g/t) can be used to promote bornite flotation, but an excess can lower dissolved oxygen in the slurry, depressing flotation of the other copper sulfides.

Chalcocite, Covellite, Digenite, Enargite, and Tennantite

The copper sulfide minerals—chalcocite, covellite, digenite, enargite, and tennantite—are readily floated using xanthates, dithiophosphates, thionocarbamates, and mercaptobenzothiazole collectors. These minerals can produce high-grade copper concentrates but often have lower copper recoveries due to partial oxidation, clays and weathered gangue minerals, and poor liberation due to complex mineral textures.

Table 3 Solubility of copper minerals in cyanide

		Percent Total Copper Dissolved	
Mineral	**Molecular Formula**	**23°C**	**45°C**
Azurite	$2CuCO_3{\cdot}Cu(OH)_2$	94.5	100.0
Malachite	$CuCO_3{\cdot}Cu(OH)_3$	90.2	100.0
Chalcocite	Cu_2S	90.2	100.0
Copper metal	Cu	90.0	100.0
Cuprite	Cu_2O	85.5	100.0
Bornite	Cu_5FeS_4	70.0	100.0
Enargite	Cu_3AsS_4	65.8	75.1
Tetrahedrite	$4Cu_3S{\cdot}Sb_3S_3$	21.9	43.7
Chrysocolla	$CuSiO_3$	11.8	15.7
Chalcopyrite	$CuFeS_2$	5.6	8.2

Source: Hedley and Tabachnick 1958

Table 4 Solubility of copper minerals in sulfuric acid and cyanide

Mineral Species	Approximate Composition	Approximate Dissolution in Sulfuric Acid Solution	Approximate Dissolution in Sodium Cyanide Solution
Oxides			
Atacamite	$Cu_2Cl(OH)_3$	100	100
Azurite	$2CuCO_3Cu(OH)_2$	100	100
Chrysocolla	$CuSiO_32(H_2O)$	100	45
Cuprite	Cu_2O	70	100
Malachite	$CuCO_3Cu(OH)_2$	100	100
Native copper	Cu	5	100
Tenorite	CuO	100	100
Secondary Sulfides			
Chalcocite	Cu_2S	3	100
Covellite	CuS	5	100
Primary Sulfides			
Bornite	Cu_5FeS_4	2	100
Chalcopyrite	$CuFeS_2$	2	7

Source: Hedley and Tabachnick 1958

Note: Samples are generally finely ground (–150 mesh), and reaction time is generally one hour or less.

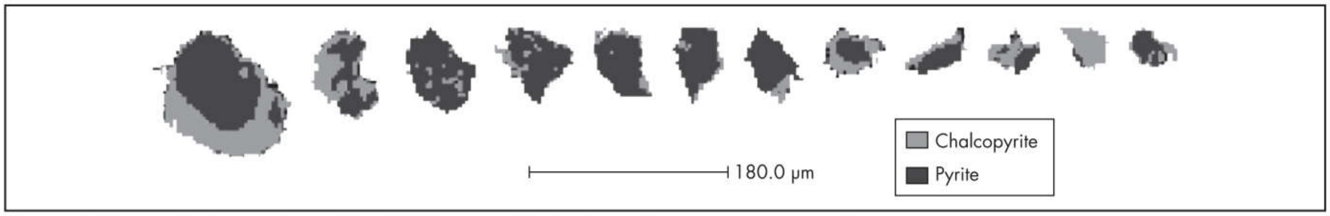


Figure 4 Binary chalcopyrite-pyrite middling particle textures from a copper porphyry ore

Mixed Copper Mineralogy

Supergene copper mineralogy can be complex but generally presents in three textures:

1. Copper sulfides replacing pyrite, either completely or as rims (Figure 5).
2. Copper sulfides replacing chalcopyrite, either completely or as rims (Figure 6).
3. Copper sulfides replacing other copper sulfides.

FLOW SHEETS

Process Selection

Most copper concentrators use the same arrangement for the process:

1. Comminution. A combination of crushing, autogenous grinding mills, SAG mills, and/or HPGRs reduce the ore particle size to P_{80} 1–4 mm, suitable to be fed to ball mills. The ball mills reduce the particle size further to P_{80} 106–250 µm.
2. Classification. Hydrocyclones are, by far, the most popular equipment for classifying the mill discharge. Fine screen decks have been used in smaller copper concentrators in recent years.
3. Rougher flotation. A high recovery froth flotation stage of 15–40 minutes of residence time produces a rougher concentrate in 5%–15% mass pull from the feed. The rougher tailings are usually sent to final tailings storage.
4. Regrind. The regrind circuit can be fed combinations of rougher concentrate, rougher-scavenger concentrate, or cleaner-scavenger concentrate.
5. Cleaner flotation. Two or three stages of cleaning, 5–15 minutes of residence time each, are used to separate the copper minerals from non-copper sulfides and gangue in the rougher concentrate to produce commercial concentrate quality.
6. Cleaner scavenger flotation of 10–20 minutes of residence time is used to capture slow-floating particles from the first cleaner tailings. The cleaner-scavenger tailings usually report to the final tailings.
7. The copper concentrate is thickened and filtered to produce a cake with 8%–10% moisture for shipment to smelters.
8. The combined final tailings are usually thickened to recover water and reagents before being sent to the tailings storage. In some cases, the water is recovered from the tailings storage facility without using a thickener.

This arrangement is shown in the simplified flow sheet in Figure 7.

Despite efforts to develop a standardized design for copper concentrators, new projects have continued to embrace variations of this conceptual arrangement due to factors that include the following:

- Oxidation and alteration of the sulfides or gangue minerals.
- By-products. Economic benefits are realized from recovering secondary products such as molybdenum, zinc, lead, gold, silver, magnetite, uranium, and nickel.
- Penalty elements. Recovery of deleterious components such as arsenic, bismuth, cadmium, uranium, fluorine, talc, antimony, lead, zinc, and iron can result in severe economic penalties for concentrate sales.
- Geography. Plants in difficult terrain may separate the main processes geographically, changing the mass and water balances and the resulting arrangement.
- Tailings storage and water consumption limits. Water recovery from tailings, tailings volume, and geotechnical stability are related. As concerns about tailings stability and scrutiny regarding freshwater consumption increase, tailings deposition strategies are trending toward paste and dry-stacked technologies.
- Preconcentration. Ore sorting, dense media, gravity concentration, and coarse particle flotation have had little industrial success, but as head grades decrease, there is growing interest in preconcentrating the copper sulfide minerals at coarser particle sizes before grinding to the customary rougher flotation particle size.

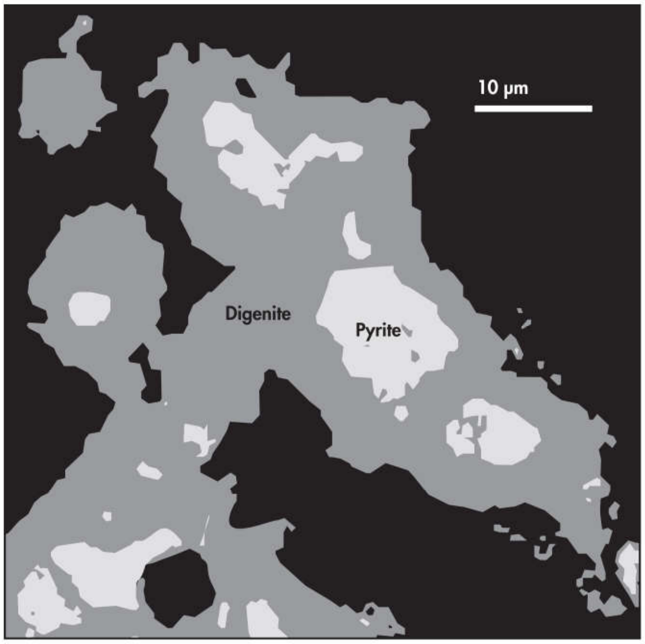


Figure 5 Digenite rimming pyrite from a manto deposit

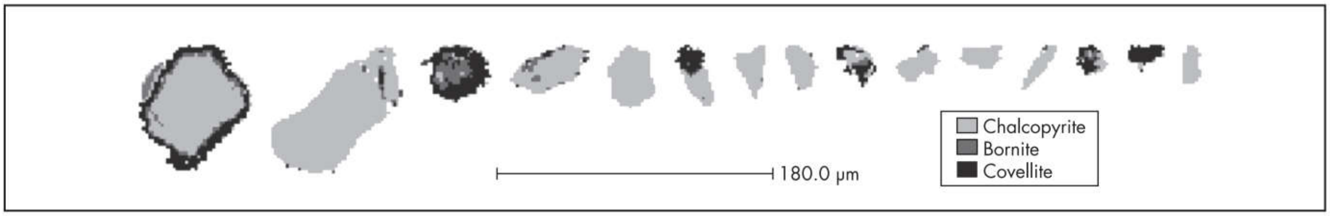


Figure 6 Mixed sulfide textures from a supergene porphyry deposit

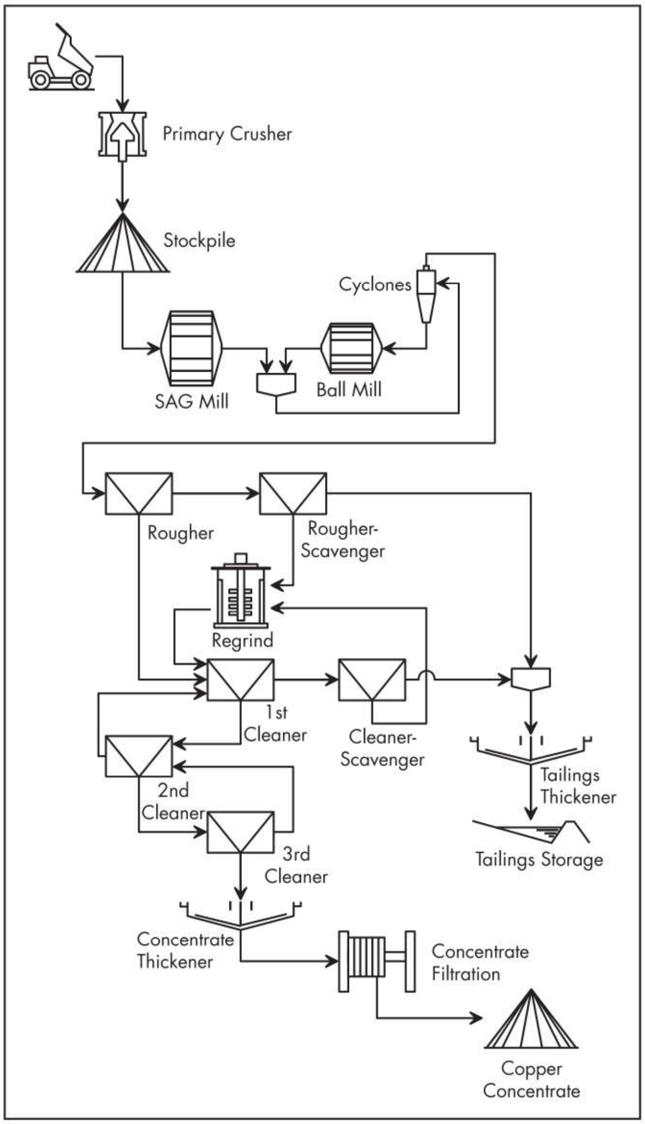


Figure 7 Standard copper concentrator flow sheet

Particle Size Selection

The conventional method of selecting particle size for plant design has been to run batch grinding tests in a laboratory mill, followed by kinetic batch rougher flotation tests. A cumulative grade–recovery curve is developed from the results. A pragmatic economic decision is then made by comparing the marginal copper recovery projected at each grind size versus the marginal capital and operating costs of grinding. This method does not consider the effect that a continuous grinding circuit with cyclone classification will have on the liberation of copper sulfides.

Copper and iron sulfides have higher specific gravity than most nonsulfide gangues. Consequently, a coarser industrial rougher flotation feed size can be considered versus the laboratory tests where cyclones are used for classification. This effect is lost where screens are used as classifiers or where the gangue has a high specific gravity.

Coarse particle recovery processes such as centrifugal concentrators have been used to recover gold and native copper from cyclone underflow streams. Klimpel (1993) reported that recovery optimization efforts for coarse grinds should focus on the coarse particles, whereas fine grinds should focus recovery optimization of the fine particles. This can be seen when reviewing the efforts at the Grasberg mine in Indonesia (P_{80} 210 µm primary, 38 µm regrind) and Prominent Hill mine in South Australia (P_{80} 100 µm, 15 µm regrind). In both cases, mechanical agitation, froth hydrodynamics, and reagent regimes were tested. Different opportunities were identified in each case for copper recovery improvement routes for coarse and fine particles.

Comminution

Copper ore particles are reduced in size to liberate the copper minerals from the gangue so that they can be floated selectively. Comminution is expensive because it consumes a lot of energy; consequently, concentrator designs are configured to preconcentrate the copper at a coarse grind size. The low-grade concentrate is then typically reground to a finer size to achieve liberation from the gangue. Coarse tailings are preferred because they can also reduce tailings storage costs, underground backfill costs, water consumption, and the capital and operating costs associated with crushing and grinding.

In the 1980s, semiautogenous milling replaced three stages of crushing as the preferred approach to comminution for large grinding circuits (Figure 8). This was due to SAG mills having higher capacities, simpler flow sheets, lower operating costs, higher availabilities, reduced maintenance complexities, and the ability to treat wet and sticky ores.

Although most new concentrators continue to favor the SAG mill approach, some existing concentrators are retrofitting the circuit with secondary crushing to increase throughput through the SAG mills. A significant portion of new concentrators are using HPGRs to reduce energy consumption, particularly where the ore is competent or where the energy costs are high. The largest HPGR units have lower capacity than the largest SAG mills; thus, high tonnage is achieved by installing multiple units in parallel (Figure 9).

Flotation

Copper is upgraded from 0.3%–2% Cu in the ore to 22%–45% in a concentrate using the froth flotation process. The flotation circuit design is different in every plant. There is no standard for the flow-sheet arrangement or circuit parameters. There are, however, a few common approaches to pragmatic flow-sheet development.

Reagents

Since all ores are different, most concentrators also have different reagent regimes for copper flotation. Flotation reagents vary depending on factors that include

- Copper mineral assemblage and textures;
- Precious metal (gold, silver) and other valuable base metals (lead, zinc, molybdenum, nickel);
- Iron sulfide (pyrite, marcasite, pyrrhotite) abundance, texture, and associations;
- Nonsulfide gangue interactions (clays, chlorite, talc, carbonates);
- Penalty element mineralogy (arsenic, bismuth, antimony, cadmium); and
- Ore variability.

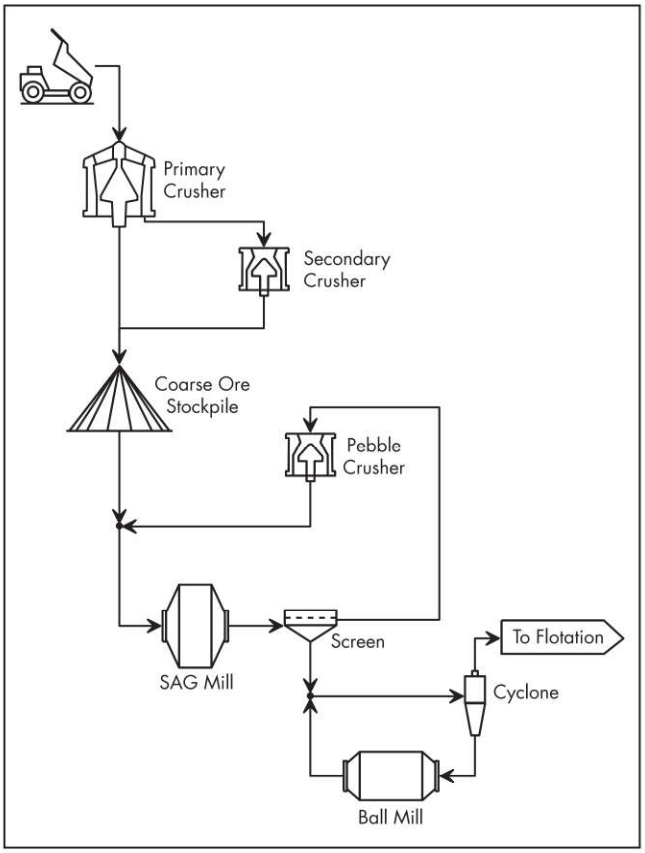


Figure 8 Common semiautogenous milling arrangement

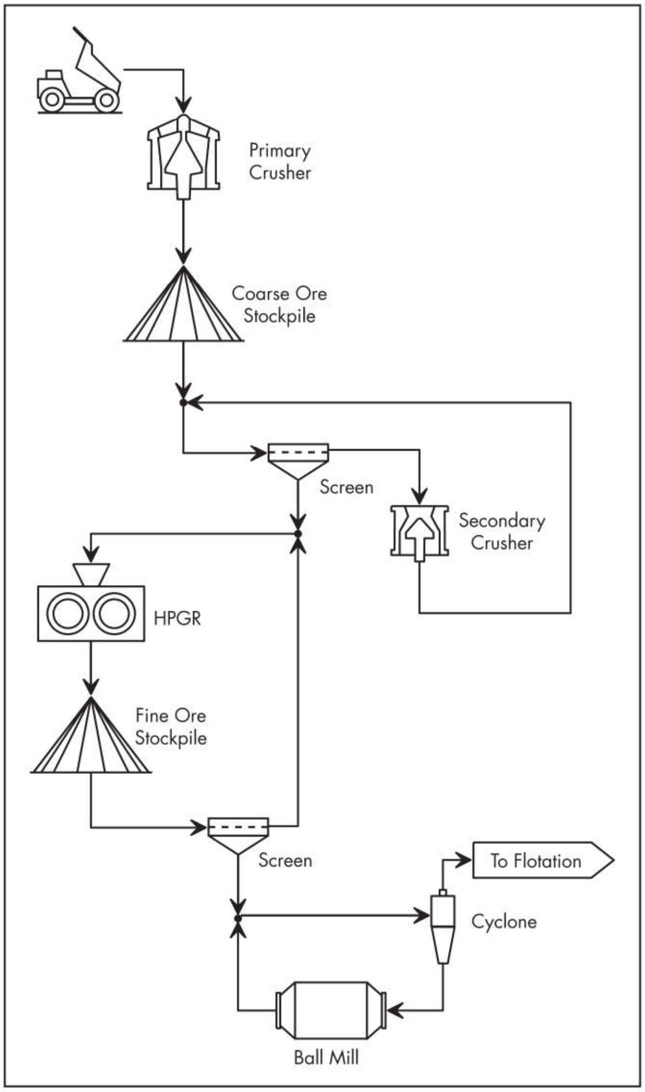


Figure 9 Typical HPGR milling arrangement

The objective is to choose a reagent regime that can consistently recover the valuable minerals into a clean concentrate, at a low cost, within the variations of daily mining feed blends to the concentrator. Table 5 provides a list of reagents commonly used in copper sulfide mineral flotation.

There has been little change in the molecules available for copper flotation in the last 30 years. Reagent manufacturers offer blends of molecules directed at certain ore types, marketing them as number codes rather than by their chemistry. Since there are several reagent suppliers, each with several candidate collectors, a metallurgist may be faced with dozens of different potential reagents to test. Many of the competing reagents have similar chemistry; therefore, testing them is redundant.

It is unusual for a reagent selection to be made on the cost of the reagent alone. Usually the metallurgical performance (selective floatability of the valuable metals) is the key reagent selection criteria. However, reagent schemes can be changed because of other factors, including occupational and health issues.

The optimum reagents for rougher flotation, including collectors, modifiers, and frothers, can be different. Rougher flotation requires strong floatability and low selectivity to achieve high recovery for large, middling particles. Cleaner flotation requires weaker, more-selective floatability to achieve a high-quality concentrate from fine particles. It is costly to change the water or adsorbed reagents between stages, so this practice is limited to separations on final concentrates, such as copper–molybdenum separation, chalcopyrite–enargite separation, and copper–lead separation.

The usual practice is to balance the reagent selection for rougher and cleaner performance. Since rougher flotation is higher tonnage, using lower pH and fewer modifier additions is preferred. It is cheaper to add depressants, pH modifiers, and collectors in the cleaner stage.

Milk of lime is used to increase pH. The lime is prepared in slaking mills and dosed to the grinding circuit and regrind circuits to achieve pH 8–10 in the rougher flotation circuit and pH 9–11 in the cleaners. Lime consumption depends on the ore acidity and the target pH but is generally 0.4–1.5 kg/t lime per metric ton of ore treated. Lime consumption is higher in plants that use seawater, so collectors that allow selectivity at a lower pH are used.

Sulfidization agents such as sodium metabisulfite and sodium sulfite (Na_2S) can be used to sulfidize unstable minerals such as bornite, semi-oxidized minerals, or oxide minerals such as malachite, forming a copper-sulfide coating that can be collected for flotation. Carboxymethylcellulose (CMC)

Table 5 Reagents used in copper sulfide mineral flotation

Reagent Type	Reagent Name	Typical Addition Rate, g/t	Reagent Function
Collector	Xanthate	5–350	Provide the appropriate level of collector for the mineral concerned (e.g., low levels of ethyl xanthate for selective flotation of chalcopyrite, but higher additions of propyl xanthate for bulk flotation)
	Dithiophosphate	10–250	
	Thionocarbamate	10–15	
	Xanthogen formate	2–25	
	Xanthic ester	2–25	
	Mercaptobenzothiazole	25–250	
	Thiocarbanilide	25–75	
Frother	Synthetic alcohols (e.g., methyl isobutyl carbinol, or MIBC)	25–50	Provide the appropriate type of froth (e.g., wet or dry; brittle or persistent; with or without some collecting power)
	Polyglycols (e.g., polypropylene glycol)	5–100	
	Polyglycol ethers	5–25	
	Alkoxy compounds (e.g., triethoxybutane, or TEB)	5–25	
	Pine oil	15–100	
	Eucalyptus oil	25–100	
	Cresylic acid	25–100	
pH Modifier	Quick lime (CaO) and slaked lime ($Ca(OH)_2$)	200–2,400	Provide high pH (>11) for depression of iron sulfides
	Sodium carbonate	500–2,500	Provide pH levels about 8–9
	Sulfuric acid	250–2,500	Provide acid solutions (pH < 6)
	Sulfurous acid and sodium metabisulfite	NA	Provide pH about 6, and depression of galena and sphalerite
Activator	Copper sulfate	100–2,500	Activates depressed sphalerite
	Sodium sulfide and sodium hydrosulfide	250–500	Activates tarnished copper sulfides and some oxide copper minerals
Depressant	Sodium sulfite and SO_2	250–2,000	Depresses sphalerite and galena
	Sodium sulfide and sodium hydrosulfide	200–500	Depresses chalcopyrite in Mo–Cu separation
	Sodium cyanide and Aero cyanide	5–250	Depresses most sulfide minerals, but especially iron sulfides
	Zinc sulfate	50–1,500	Depresses sphalerite
	Sodium dichromate	100–2,500	Galena depressant in Cu–Pb separation
	Sodium silicate	50–1,000	Gangue dispersant
	Nokes reagent (P_2S_5 + NaOH)	5,000	Chalcopyrite depressant in chalcopyrite–molybdenite separation
	Carboxy methyl cellulose (CMC)	5–500	General gangue depressant
	Organic colloids (e.g., glue, gelatin, starch, tannin, and quebracho)	5–500	Gangue depressant for minerals such as talc, mica, and carbonaceous matter
	Dextrin	29–120	Carbonaceous gangue depressant
Other	Fuel oil	250–2,000	Froth modifier
	Nitrogen	NA	Enhances copper and nickel flotation
	Air, oxygen	NA	Oxidizes reductants in the pulp and sulfide mineral surfaces
	Carbon dioxide	NA	Inert carrier gas and pH control

Source: Woodcock et al. 2007
NA = Not available.

and guar gum are used to depress naturally hydrophobic magnesium silicate minerals, such as talc and pyroxene, whereas dextrin and starch are used to depress carbonaceous minerals.

Flow-Sheet Development Test Work

Mineralogy and batch kinetic flotation tests are used to determine the relationship between laboratory rougher grind size. Typically, P_{80} 75, 106, 150, and 212 µm are used for the first trial, and narrower ranges can be tested after that. An economic evaluation is required to determine the optimum grind size for an industrial plant, since the capital and operating costs for achieving the grind must be compared to the value of the metal recovered at each grind size.

Once the grind size has been established, grade–recovery curves for different reagents should be used to select a reagent suite. Finally, collector dose requirement can be determined by stage, adding small amounts of collector in stages, like a titration. The grade and recovery of the first kinetic concentrates will aid in assessing whether the first rougher concentrates can be sent straight to cleaning or if regrinding will be required first. Regrinding and cleaning tests are performed in a similar manner.

Where possible, the flow-sheet and reagent regime should be simple, be similar for all ore types, and have few low middling and recirculating streams. Once a flow-sheet and reagent regime has been established, a locked-cycle flow sheet should be run to check the performance when middling streams reach steady state. Figure 10 shows an example of a selectivity curve comparison where, as more collector is added, the flotation becomes less selective for copper over iron. Figures 11 to 13

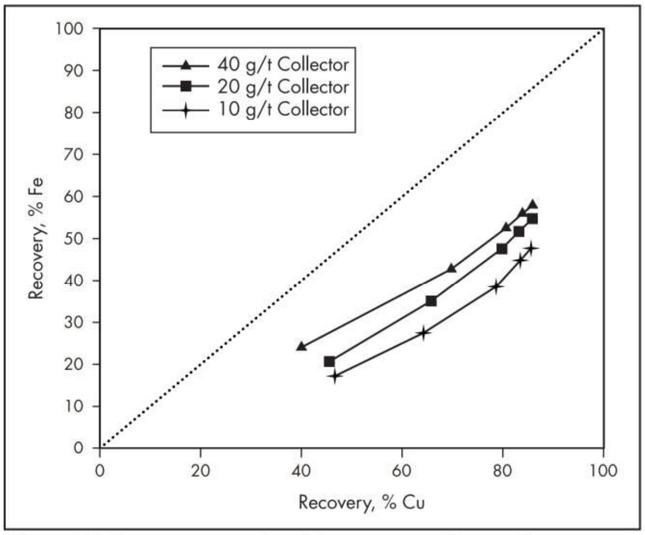


Figure 10 Iron–copper selectivity versus collector dose

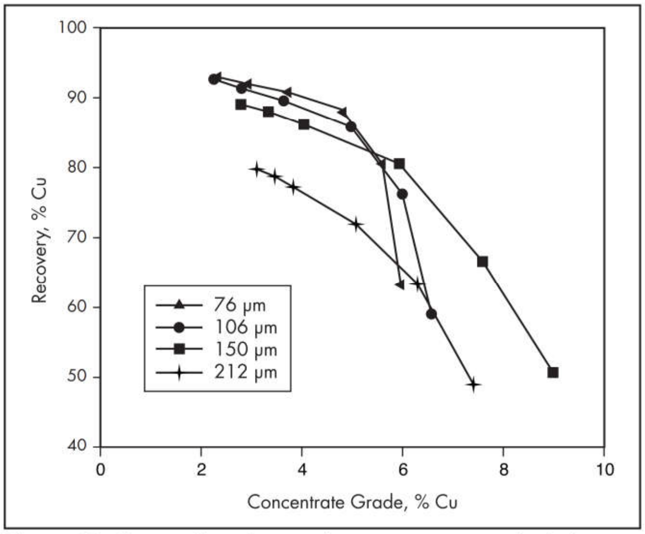


Figure 12 Copper flotation performance versus grind size

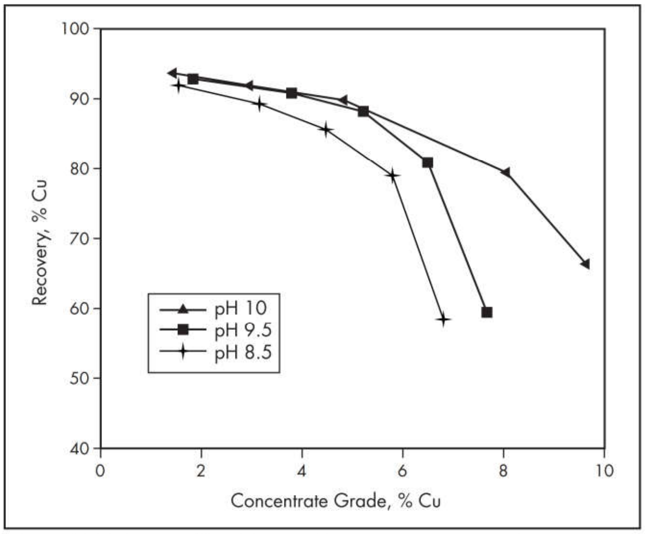


Figure 11 Copper flotation performance versus pH

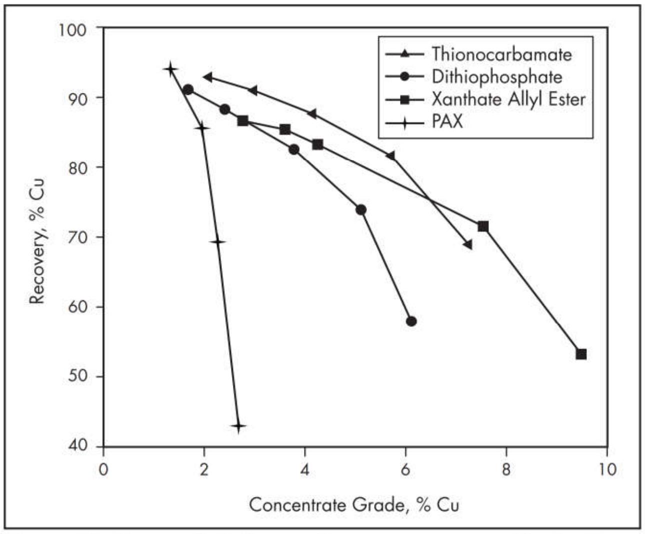


Figure 13 Copper flotation performance versus collector type

show examples of grade–recovery curve comparisons of flotation development test-work results to evaluate pH, grind size, and collector type for copper ores, respectively. The curves that are closer to the top right of each graph indicate better copper floatability and selectivity over gangue.

Grinding Media Effects on Flotation

The forged steel balls in a grinding mill with chalcopyrite and pyrite can create conditions that have a strong impact on copper flotation performance (Greet et al. 2004). Iron from the balls enters the solution because of corrosion. While ball mill ball consumption for low-sulfide, alkaline ores is primarily due to abrasion, in high-pyrite ore grinding, more than 50% of the ball consumption can be attributed to corrosion. All the resulting iron in solution can form iron oxide coatings over the chalcopyrite surfaces, reducing its floatability. The corrosion of the grinding media also has an anodic effect on copper minerals, leaching copper into solution. This copper can partially activate the pyrite, reducing the effect of collectors, crowding the froth, and diluting the concentrate. Operators will typically act to maintain the copper concentrate quality, reducing the copper recovery.

Several approaches can be used to reduce the impact of the grinding media effect on copper flotation performance:

- High pH. Increasing the pH in the grinding mill can passivate steel surfaces and precipitate cations before they affect surface chemistry.
- Autogenous grinding.
- Inert media. High-chrome balls can be used in SAG or ball milling, and high-chrome, sand, or ceramic media can be used in regrind milling.

Conditioning

Large copper concentrators have generally removed conditioning tanks from the circuit. Collectors and lime are added to the SAG mill discharge, leaving the residence time of the ball mill and cyclones for conditioning. Frothers do not require conditioning and are therefore added to the flotation cell feed. Conditioning tanks can act as a buffer to reduce the impact of abrupt changes in flow or grade on flotation performance.

Equipment Selection

Self-aerating and forced-air mechanical machines are used successfully for rougher and rougher-scavenger stages. Either mechanical cells or column flotation cells can be used for cleaner and recleaner stages. Column cells are often favored in ores with high viscosity and clay content, as froth washing is required to reduce gangue entrainment to the final concentrate. Cleaner-scavenger cells are typically mechanical cells to ensure high recovery of fine and coarse middling particles. Typical rougher residence times are between 20 and 30 minutes.

The growing interest in using new-generation flotation machines facilitates high-shear bubble attachment and froth washing in a single unit. These units include the Jameson cell, the staged flotation reactor, and the StackCell, for example. Currently, these units have not replaced rougher cells; however, they have been used in cleaning circuits in place of mechanical and column cleaner cells.

Outotec advises that the range of froth carrying rate should be

- 0.8–0.15 t/m^2/h for rougher cells,
- 0.3–0.8 t/m^2/h for scavenger cells, and
- 1–2 t/m^2/h for cleaner cells.

The froth carrying rate is the dry mass of concentrate that is recovered per hour per square meter of cell froth area (t/h/m^2). If the carrying rate is low, the froth will tend to collapse, returning the concentrate to the flotation cell before it is pushed into the launders, reducing recovery. This can be resolved using froth crowders. If the carrying rate is too high, then there will likely not be enough bubble capacity to recover all of the concentrate required, reducing recovery. Also, the froth will not have enough time to drain adequately, reducing the quality of the concentrate. This can be resolved by adding more flotation cells to get more froth area or by reducing the concentrate load though reagent or flow-sheet modification.

As flotation cells and plant throughputs get larger, the issue of high carrying-rate loads in the cleaner circuits are more common. This is because the volume of flotation cells grows to the cube power compared to the diameter, whereas the froth area only grows to the square power.

Outotec recommends that the lip loading not exceed 1.5 t/h/m. The lip loading is the dry mass of concentrate that is recovered per hour per linear meter of froth overflow to a launder (t/h/m). If the lip loading is too high, then concentric or radial launders can be added to the cells, although that will reduce the froth area for the carrying rate calculation.

Copper Flotation Flow-Sheet Strategies

Basic Copper Flotation

A simple flow sheet has become commonplace in plants that treat low-grade, low-complexity copper ores (Figure 14). Such plants are used to concentrate a high recovery of valuable metals (copper, molybdenum, gold) into a rougher concentrate at a coarse grind size. The primary grind is often between P_{80} 150 and 250 μm. The rougher concentrate is reground to produce a feed to the cleaner circuit of P_{80} 20–56 μm.

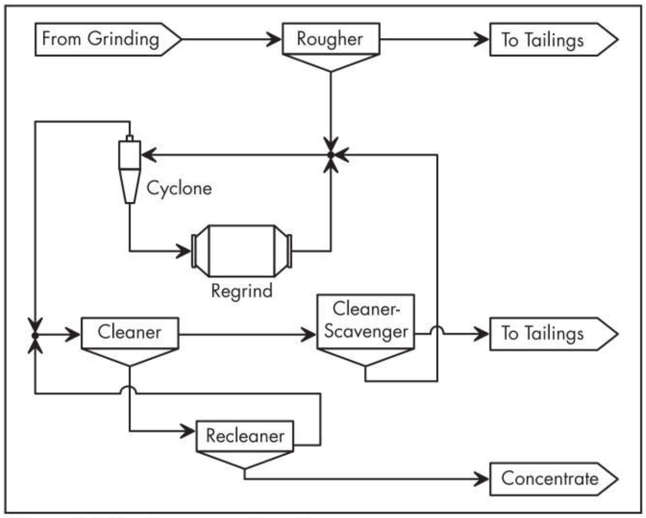


Figure 14 Basic copper flotation arrangement

The first cleaner tailings are scavenged in a high-recovery cleaner-scavenger stage. The tailings from the cleaner-scavenger stage are sent to final tailings rather than returning to the rougher feed. This is to avoid recirculating loads of fine middling and weakly activated pyrite particles between the roughers and cleaners. This recirculating load would destabilize the rougher stage, reducing the residence time. Also, the fine reground particles would degrade the hydrodynamics of the rougher flotation bubbles and froth. Most older plants that used to return the cleaner tailings to the rougher feed have been modified to send the cleaner tailings directly to final tailings.

The first cleaner concentrate can be cleaned further in one or two additional cleaning stages, arranged in countercurrent, where concentrates advance to the next stage while the tailings report to the feed of the previous stage.

Concentrators that use this arrangement include Chuquicamata, Disputada, Escondida, Esperanza, Los Bronces, Los Pelambres, and Ministro Hales (all in Chile); Antapaccay, Constancia, and Las Bambas (all in Peru); Alumbrera (Argentina); Bagdad, Metcalf, and Ray (all in the United States); Boddington (Australia); Sossego (Brazil); Grasberg (Indonesia); Lumwana (Zambia); and Oyu Tolgoi (Mongolia).

Split Rougher Flotation Flow Sheet

The enriched copper minerals, including chalcocite, bornite, and covellite, can cause problems when they are reground: They are brittle and can slime, resulting in recovery losses to the slow-floating –15 μm fraction; and they can inadvertently activate iron sulfides, decreasing the selectivity of the cleaners and reducing the final concentrate grade. Consequently, plants that treat these minerals can often direct the first rougher concentrate to the second cleaner feed, bypassing the regrind and first cleaner stages. Entrained gangue and coarse middling particles will then have to be classified back to the cleaner-scavenger for reject to tailings or to the cleaner-scavenger concentrate for regrinding. The liberated copper particles will report directly to the final concentrate without having been reground. The slower-floating rougher-scavenger concentrate is reground, as per the basic copper flow sheet.

Concentrator design and layout will influence the operator's ability to implement or change a split rougher flow sheet in an existing plant (Figure 15). Bypassing regrind from the first rougher cells requires that these cells be operated at high-grade concentrates. Achieving these high grades requires understanding of the mass pull–grade relationship and how to control it. This can be achieved by hydrodynamics or reagents.

Plants that operate split rougher flow sheets include Batu Hijau (Indonesia) and Salobo (Brazil).

High- and Low-Grade Regrind Circuits

The two Cerro Verde concentrators in Peru use a variation of the split rougher flow sheet. The rougher and rougher-scavenger concentrates are reground separately in high-grade and low-grade regrind mills. This flow sheet has more unit operations than the basic flow sheet, but capital intensity is still low since the two concentrators treat a combined 360,000 t/d (Figure 16). Somewhat similar approaches are used at Antapaccay and Las Bambas (Peru), and Bingham Canyon (United States).

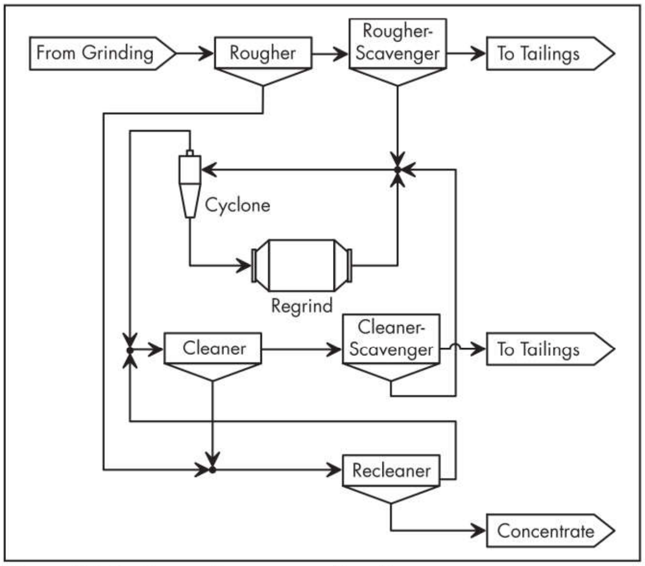


Figure 15 Split rougher copper flotation arrangement

High-Grade Copper Treatment Flow Sheet

Plants that treat high-grade copper ores may find that the cleaner circuit cells' carrying capacity or concentrate lip loading may be high, lowering cleaner circuit recovery. To reduce the burden of the fast floating, liberated copper particles from the cleaner circuit, a cleaner-scalping stage can be used. A high-capacity, high-shear flotation machine with froth washing capabilities is ideal for this stage (Figure 17). The cleaner-scalper concentrate reports to the final concentrate, while the tailings report to the normal cleaning circuit, but with reduced copper tonnage.

Plants that use this arrangement include Carmen de Andacollo (Chile), Phu Kham (Laos), Prominent Hill, and New Afton (Canada). Carmen de Andacollo scalps final concentrate directly from the rougher concentrate using mechanical flotation cells before regrind. Phu Kham and Prominent Hill scalp final concentrate from the regrind discharge using Jameson cells. New Afton scalps final concentrate from the regrind discharge using a bank of staged flotation reactors.

Sand-Slime Circuits

Split sand-slime circuits can be used where clays in the slimes are found to hinder the collection and recovery of the coarse particles. This option is expensive because it requires two rougher flotation circuits to be built and run in parallel but has the advantage of being able to focus the reagent and froth recovery arrangements for coarse and fine particles separately (Figure 18).

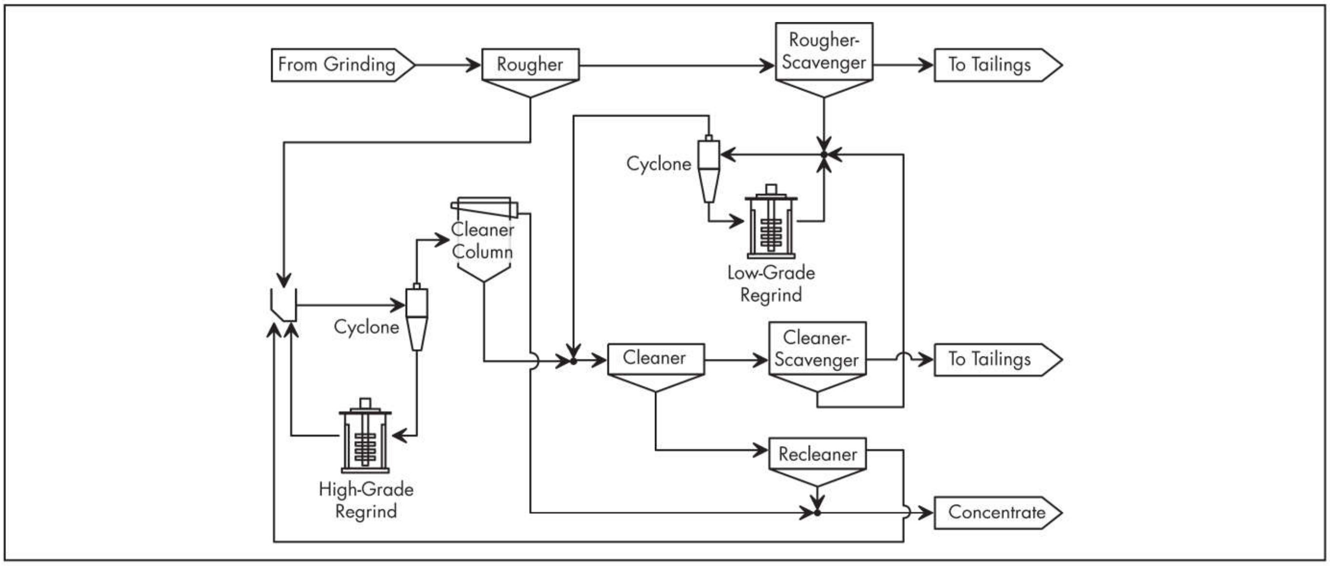


Figure 16 High- and low-grade regrind copper flotation arrangement

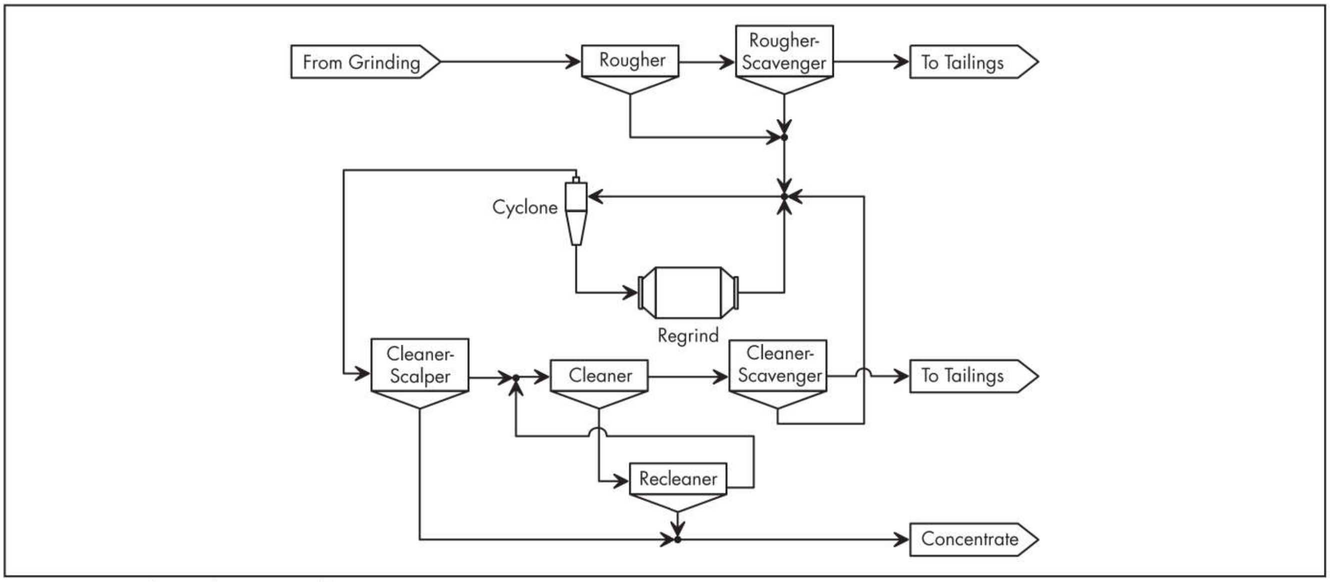


Figure 17 High-grade copper flotation arrangement

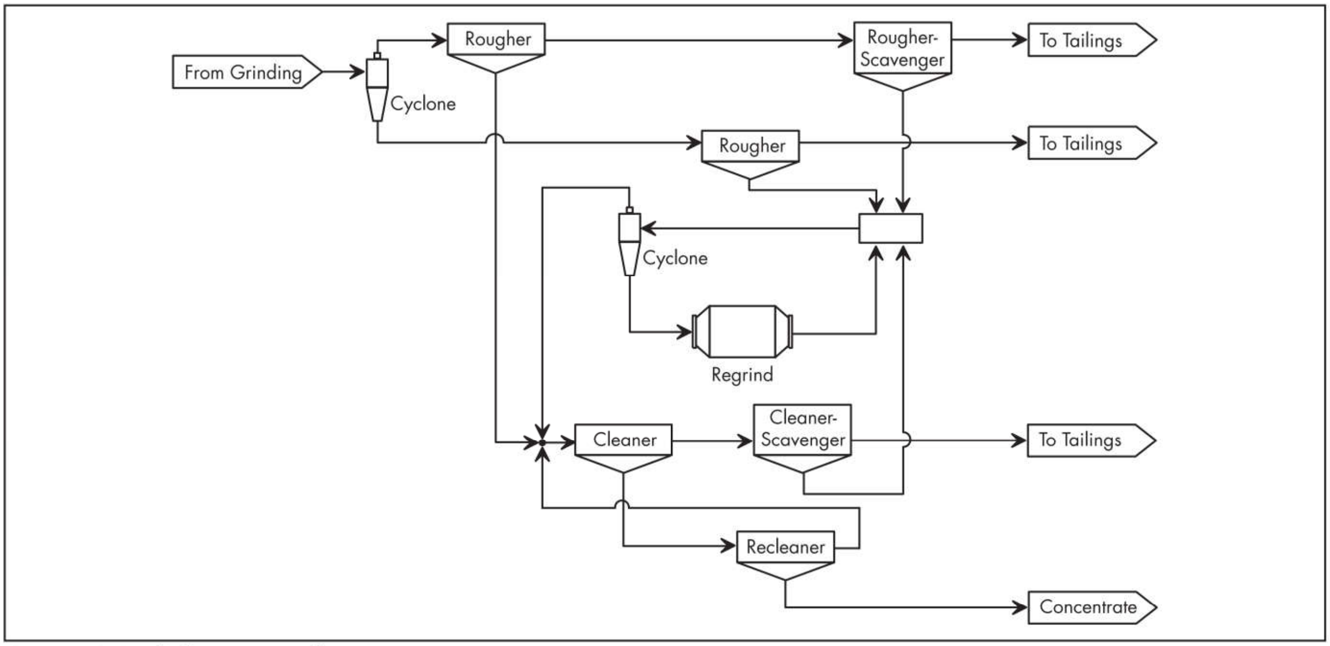


Figure 18 Sand-slime copper flotation arrangement

Cuajone in Peru uses this arrangement. Collahuasi in Chile has a similar approach, where the rougher tailings are cycloned to recover coarse particles to the underflow, which are scavenged by flotation, then reground for cleaning.

Removing Metallic Copper in the Milling Circuit

Where present, it can be favorable to remove metallic copper or gold particles from the circulating loads in a grinding circuit. The density of copper is 8.95, and gold is 19.3; therefore, particles of these metals will be ground to a very fine size before reporting to the cyclone overflow, since the cyclone classifies particles by density and by size. This approach is useful for other heavy minerals, such as galena, for example, as it has a density of 7.2–7.6 and is fragile, so will slime in a closed grinding circuit.

Smaller gold operations often use gravity bowl concentrators, but these devices are less efficient for sulfide ores, have limited capacity, and add water dilution to the circuit. Several larger concentrators treating copper-gold ores use a combination of flash flotation and gravity separation to enhance gold recovery (Figure 19). A portion of the primary cyclone underflow stream is passed through the flash flotation cell to recover liberated, fast-floating gold particles. This prevents them from returning to the ball mill, where they will be overground with the potential of not floating them in the downstream rougher circuit.

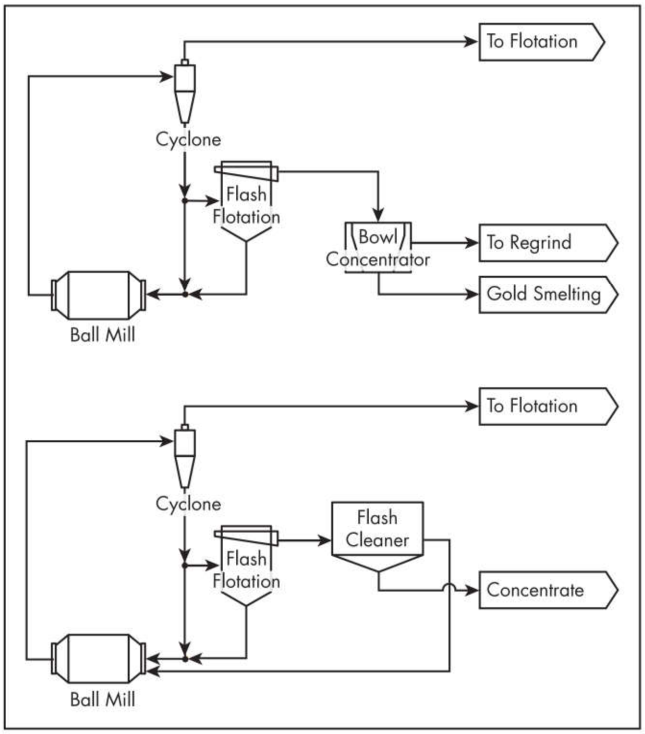


Figure 19 Grinding circuit scalping alternative arrangements

A disadvantage of removing metals in the grinding circuit is that the primary cyclone overflow no longer represents the plant feed for metallurgical accounting purposes. Care needs to be taken in frother selection using a flash cell in the grinding circuit. The flash circuit in grinding normally requires strong frothers to work properly because of the coarse grind. Successful implementation requires that the strong frother for flash flotation be compatible with the subsequent rougher circuit.

Tintaya (Peru) used gravity concentrators in place of flash flotation. Boddington; Cadia and Telfer (both in Australia); and New Afton use flash flotation, but the flash flotation concentrates are handled in different ways, depending on the proportion of copper sulfides, gold, and pyrite, and their associations. Cadia does not have a cyanidation circuit, so the gold metal is recovered from the flash flotation concentrate via gravity concentration and is then upgraded and smelted to doré. Telfer, on the other hand, has both gravity concentration for gold and flash flotation for copper and copper sulfides in the grinding circuit.

Copper Concentrate Treatment to Recover By-Products

Copper can often be present in ores with other metals that are co-collected to a bulk concentrate for subsequent separation. In Chile, this arrangement is known as collective flotation (bulk copper-molybdenum) and selective flotation (copper–molybdenum separation). Other metals can include lead, bismuth, molybdenum, nickel, and arsenic. A generalized flow sheet for this arrangement is shown in Figure 20. Depressants for copper separation flotation are listed in Table 6.

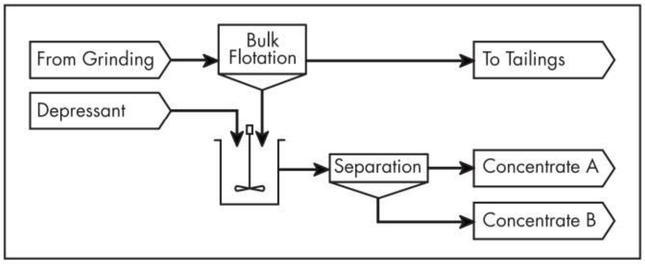


Figure 20 Copper concentrate separation arrangement

Table 6 Depressants for copper separation flotation

Bulk Concentrate	Concentrate A	Concentrate B	Depressant
Cu-Mo	Copper	Molybdenum	NaHS
Cu-Bi	Copper	Bismuth	Cyanide
Cu-Ni	Nickel	Copper	Oxidant
Cu-As	Copper	Arsenic	Reductant
Cu-Pb	Lead	Copper	Dichromate or sodium metabisulfite
Cu-Pb	Copper	Lead	Cyanide

Ore-Specific Flow-Sheet Strategies

Copper-Gold Ores

Copper and gold are often found together in the same deposits, though in different proportions and mineral assemblages. Gold that reports to a commercial copper concentrate is readily recovered by the smelter, and so has favorable payment terms. Cyanide-soluble copper that reports to a gold cyanidation plant will increase cyanide consumption and augment the cost of gold recovery from the leach solution.

Gold that is associated with pyrite can be high or low grade. In South America, the pyrite associated with porphyry copper ores has low gold grades. The copper-gold deposits in Southeast Asia and Australia tend to have a high gold grade in the pyrite, which can be recovered as a separate concentrate for downstream treatment to recover the gold.

If the pyrite gold grade is low, then it cannot be co-floated with the copper, as the decrease in copper concentrate grade will incur more additional transport and treatment charges than the gold will attract in credits. If the gold-to-copper ratio is high, then a bulk gold-pyrite-copper concentrate may attract favorable commercial terms at copper smelters.

Gold that in a low-grade pyrite concentrate or in copper flotation tailings may be recovered by cyanidation, if amenable. If the gold is associated with pyrite or arsenopyrite and is refractory to cyanidation, then oxidation would be required before it can be leached. Where cyanide-soluble copper is present in the feed to a gold cyanidation plant, copper flotation may be primarily used to reduce cyanide consumption. Liberated native gold and electrum can be recovered from the grinding circuit using flash flotation or gravity concentrators.

Copper-Lead-Zinc Ores

Polymetallic base metal mines often contain copper, lead, zinc, and other minor metals such as silver, gold, arsenic, and antimony. These ores are typically heterogeneous, and so a

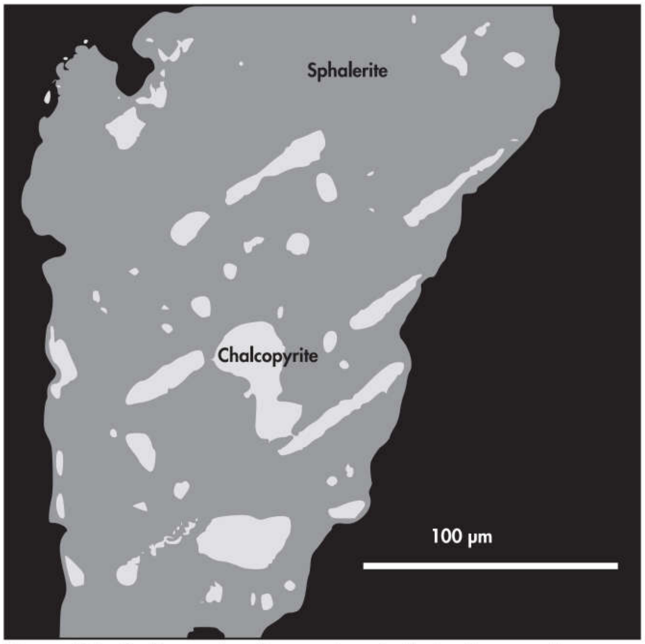


Figure 21 Chalcopyrite disease in sphalerite from a skarn deposit

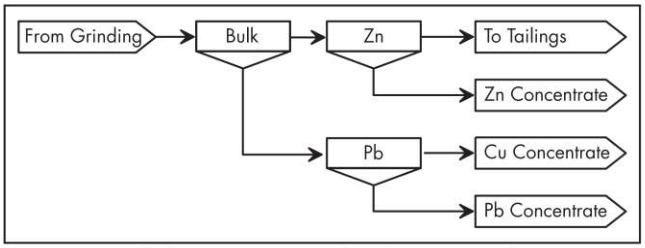


Figure 22 Bulk flotation approach to base metal flotation

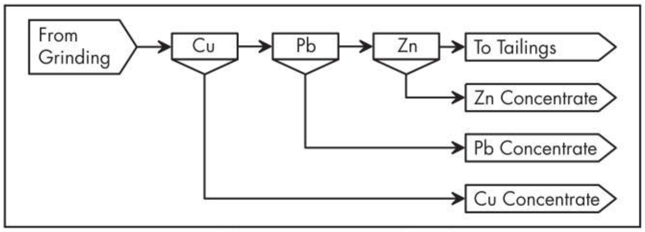


Figure 23 Sequential flotation approach to base metal flotation

treatment plant design can follow several approaches, including campaign ore and mining blends:

- **Campaign ore.** Antamina (Peru) is one of the world's largest base metal mines and runs one- to three-week campaigns of eight ore types based on relative copper, zinc, bornite, and bismuth grades. This is done to facilitate high recoveries and concentrate grades for copper, lead, zinc, and molybdenum concentrates. Antamina can achieve this through detailed geometallurgical test work and careful open pit mine planning.
- **Mining blends.** Most mixed base metal concentrators receive the ore from underground mines. Since underground mining is expensive compared to open pit mining and concentrator operating costs, the underground production rate is the bottleneck and there is little opportunity for optimizing the concentrator feed for metallurgical response. Therefore, the concentrator flow sheets and reagent regimes are designed to be flexible rather than optimized. Multiple fine ore bins can be used to blend ore types, but inevitably the concentrator feed varies, affecting the recovery and product quality.

Copper and zinc are commonly found together in volcanogenic massive sulfide, skarn, stratabound, and massive sulfide deposits. It is usual to find chalcopyrite rather than enriched copper sulfide minerals associated with sphalerite.

Chalcopyrite disease in sphalerite is a particularly difficult texture to treat by flotation (Figure 21). The chalcopyrite inclusions in the sphalerite are too fine to liberate by grinding, and since a portion of the particle surface is chalcopyrite, the particle will be impossible to selectively float from chalcopyrite.

When working with deposits that have copper sulfide minerals and sphalerite in different domains, combining the two in grinding may result in activation of the sphalerite by the copper sulfides. For example, bornite from a porphyry domain mixed with sphalerite from a neighboring skarn domain could cause an increase in zinc displacement to the copper concentrate, from 5% to 90%, that cannot be controlled by depressants. This will reduce zinc recovery and reduce the quality of the copper concentrate.

The two basic approaches to copper-zinc and copper-lead-zinc flow-sheet design are bulk and sequential. The bulk flotation process is, by far, the most popular, because it decouples the difficult copper–lead separation flotation stage from the simple zinc flotation stage (Figure 22). This reduces reagent costs and operating complexity.

One of the issues with the bulk flotation process is that once copper in depressed, it is difficult to activate again. This means that any impurities in the bulk concentrate can be concentrated in the copper concentrate, reducing its quality. Therefore, for some deposits it is advantageous to selectively float a copper concentrate first, especially where gold or silver recovery to the copper concentrate can be improved. This is done using the sequential flotation arrangement (Figure 23). Copper is collected by using reagents that have a high selectivity of copper over galena, sphalerite, and iron sulfides. Lead is next collected using zinc cyanide complex to depress the zinc, and weak xanthate, thionocarbamate, or Aerophine to float the galena. Finally, the zinc is recovered by activating the sphalerite with copper sulfate and collecting with a thionocarbamate at pH 9–10 or a xanthate collector at pH 10–12.

As long as the copper mineralogy is chalcopyrite and no cyanide-soluble copper minerals are present, then cyanide can be used to depress the copper, allowing for the relatively simple separation of copper and lead by floating the galena. If cyanide-soluble copper minerals and/or precious metals are present, or if the ore is not otherwise amenable to copper depression by cyanide, then lead depression can be used. In the past, lead depression was achieved by adding dichromate reagent. Unfortunately, this effective reagent poses risks to health and the environment; consequently, it is often no longer approved. In such cases, a sulfoxy species such as

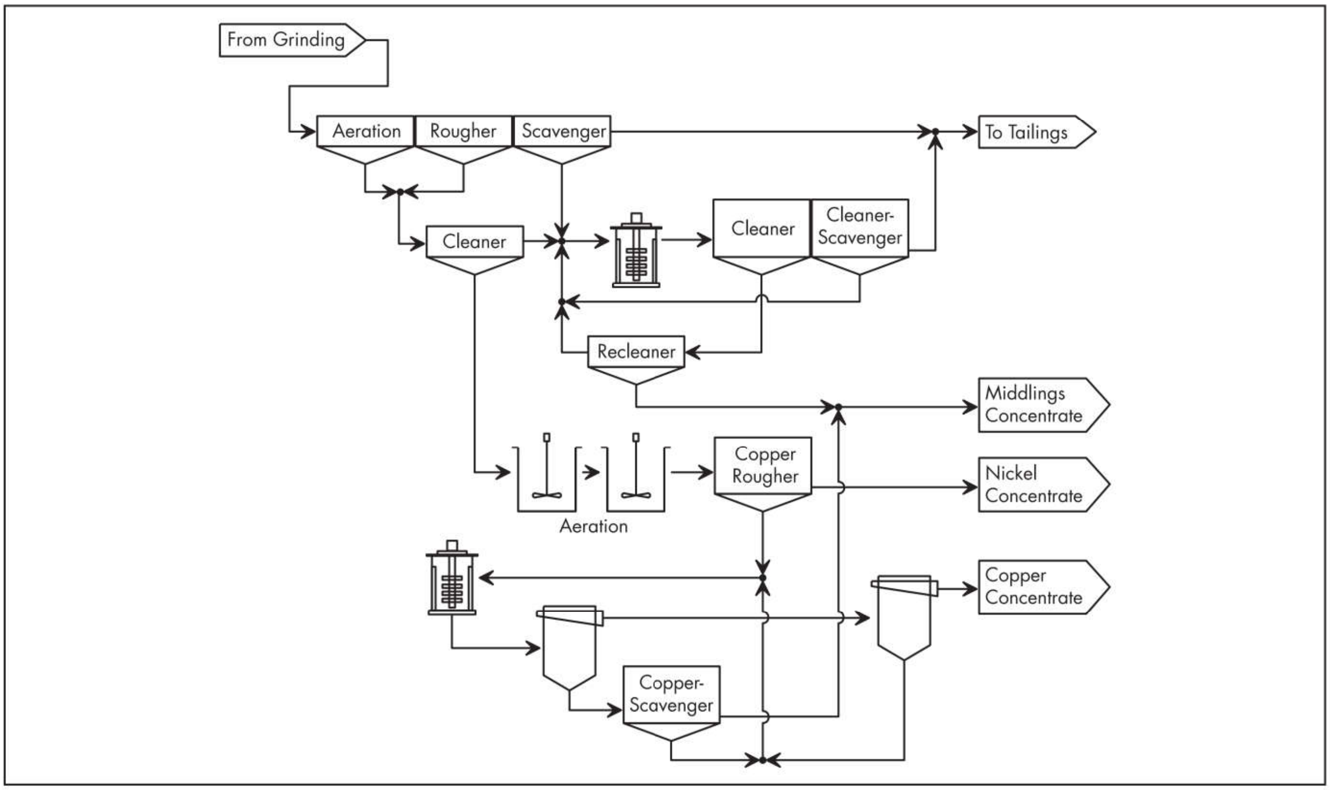


Figure 24 Voisey's Bay, Canada, flow sheet

sulfur dioxide (SO_2) gas or sodium metabisulfite can be used. Variations on these processes can include the following:

- **Carbon pre-float.** Organic carbon can rob reagents from flotation, flatten froth beds, and dilute concentrates. A pre-flotation stage to remove the carbon can be used, either without reagent addition or with a small amount of fuel oil added to grinding.
- **Talc pre-float.** Zinc ores can contain naturally hydrophobic talc that can slime in the regrind mill and dilute the concentrate grade. Pre-float of talc without reagents can help at high grades, but at lower grades, CMC addition can aid in depressing the talc.
- **Scalping lead from grinding.** Because of its high specific gravity, galena tends to recirculate in the cyclone underflow, eventually sliming, reducing lead recovery, and possibly inadvertently activating the sphalerite to float in the bulk concentrate. The use of flash flotation in the grinding circuit can be used to recover the galena when it is still coarse.

Clearly, several of the preceding processes are incompatible. The variety of flow sheets and reagent suites in copper-lead-zinc concentrators is much greater than for other kinds of copper concentrators.

Copper-Nickel Ores

Copper-nickel ores typically contain copper as chalcopyrite and nickel as pentlandite. These minerals are usually accompanied by pyrrhotite and talc, which impede the production of high-quality concentrates. The focus of the flow sheet and reagent additions will depend on the ratio of copper to nickel.

In Australia, the copper head grades are low, and the separation is made as a cleaning process for a nickel concentrate, whereas in Canadian operations, the separation results in two commercial concentrates (Figure 24). Chalcopyrite, pentlandite, and some pyrrhotite are recovered to a bulk concentrate using a strong xanthate collector. The bulk concentrate is typically reground to liberate the chalcopyrite from the pentlandite. The copper and nickel in the bulk concentrate can be separated by two regimes:

1. The bulk concentrate is oxidized by aeration in the presence of lime for 30 minutes to depress the pentlandite.
2. The pentlandite is depressed by raising pH higher than 10 using lime and by adding cyanide, dextrin, or starch.

The chalcopyrite and pyrrhotite are selectively floated, producing a nickel concentrate in the flotation tailings. The chalcopyrite-pyrrhotite concentrate is then upgraded using SO_2, Na_2S, or triethylenetetramine to depress the pyrrhotite. Talc can be dispersed in the rougher flotation using soda ash or depressed using CMC. Although modern reagent regimes are more efficient at rejecting pyrrhotite, some older operations continue to remove it from final concentrates using magnetic separation.

Copper Ores High in Arsenic

Arsenic mineralogy can be difficult to identify and map in a copper deposit. The recovery of arsenic can be affected by the arsenic mineralogy, mineral textures, and reagent regime. Enargite and tennantite are copper sulfide minerals and, consequently, are extremely difficult to separate from other copper minerals by flotation. Limited success has been reported by

Chuquicamata using an oxidation process and by Northparkes (Australia) using a reductive process. In any case, separating the enargite/tennantite from the other copper minerals also results in a corresponding separation of copper. The Ministro Hales concentrator at Chuquicamata treats high-enargite ore through a conventional flotation concentrator, and the resulting high-arsenic copper concentrate is treated by roasting.

Orpiment and realgar are uncommon in copper sulfide ores but are readily separable by flotation. Arsenopyrite can be separated from copper sulfides by using the same strategies as for selective flotation against iron sulfides, such as using a selective collector, high pH, or cyanide. Tetrahedrite can also contain arsenic, copper, and precious minerals. The amount of arsenic in the tetrahedrite is usually small compared to the precious metal credits and therefore is not of concern. Tetrahedrite has strong floatability under normal copper flotation conditions.

Currently 0.5% As is the commercial limit for copper concentrates, and many copper head grades are 0.5% Cu or less. Assuming that copper and arsenic recoveries are similar, and with a copper upgrade ratio of 60:1, an arsenic head grade of more than 0.008% or 80 g/t As may result in a high-arsenic copper concentrate. If the arsenic can be depressed, then the acceptable head grade of arsenic can be higher. Therefore, mine planners should calculate the copper concentrate arsenic grade using the copper and arsenic recoveries, and copper and arsenic head grades.

Geometallurgical Planning

Plant operators should be well informed of the expected metallurgical performance from the geometallurgical model. If an ore type enters production that fouls the concentrate grade, the operators may act to maintain the concentrate grade by cutting recovery, unnecessarily. Conversely, if a lower recovery is expected, the operators should know to avoid over-adding reagents that may result in a lower-than-necessary concentrate grade. Geometallurgical planning can be used to avoid unexpected excursions in rock hardness, pulp viscosity, and penalty element grades that could affect the throughput, metallurgy, and profitability.

The performance of copper flotation can be predicted by bench-scale flotation tests. Kinetics flotation tests are used to determine the rougher recovery, whereas kinetics and locked-cycle tests are used to predict the cleaning circuit losses and concentrate quality for each ore type.

The metallurgical performance of mining blends can often be calculated from the sum of the parts. That is, the fine metal recovery and concentrate grade are calculated separately for each ore type in the blend, then the concentrates are combined to total the fine metal recovery and a weighted average concentrate grade for the blend. This does not apply where there are toxic effects between ore types. Examples of such toxic effects could include

- Clays that reduce floatability for the entire blend,
- Pyrite or sphalerite in one ore type that is activated by enriched copper sulfides in another, and
- High-pyrite ore types that may reduce the ability to recover slow-floating components such as gold or molybdenum from another ore type.

On the other hand, if one ore type contains high penalty element grades, such as arsenic or bismuth, then blending it with other cleaner ore types can result in a cleaner final concentrate that does not attract any penalty.

Thickening and Filtration

Concentrate is thickened and filtered before being fed to a smelter. Conventional or high rate concentrate thickeners are used to thicken flotation concentrate to 60%–68% solids prior to filtration. Entrained air can cause froth to form on the thickener surface. Vertical and horizontal plate pressure filters are most commonly used for filtration, achieving 8%–10% moisture in the filter cake.

Usually the smelters are distant from the concentrator, so the concentrate is transported there by any combination of rail, truck, pipeline, or ship. Where pipelines are used, some unit processes can be relocated from the concentrator site to the pipeline discharge site. For example, Los Bronces, Collahuasi, and Esperanza have the copper-molybdenum separation plants, concentrate thickening, and concentrate filtration at sea level where the concentrate pipelines discharge at the coast.

Antamina, Escondida, Grasberg, Los Pelambres, Ridgeway (Australia), and Bingham Canyon transport their concentrates via pipeline, before thickening and filtration. Thermal drying of copper concentrates was common in older plants that used disk vacuum filters but is uncommon in newer plants that use pressure filters.

Seawater Washing

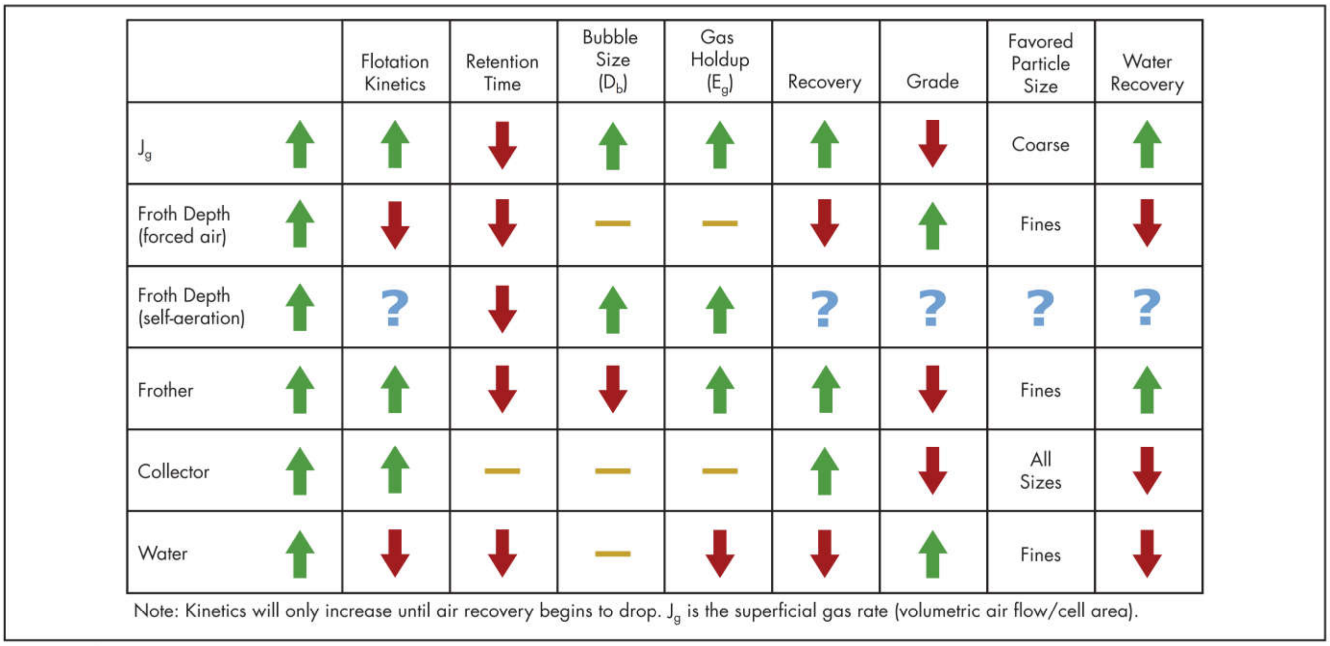
When seawater or saline water has been used for flotation, the final concentrate needs to be washed with clean water to avoid smelter penalties for chloride levels. Batu Hijau uses three stages of countercurrent decantation to wash the concentrate. Fresh water captured from rain is used. Esperanza uses a washing stage in the horizontal plate pressure filtration of final copper concentrates. Reverse osmosis water consumption per metric ton of final concentrate is 0.25 m^3/t.

Emerging Processes

The energy and grinding steel used in comminution is a major contributor to the overall operating costs (20%–30% of operating expenses) and carbon footprints of the mining operation (15–30 kg C/t ROM [run-of-mine]). Furthermore, storage of finer tailings typically consumes more water than that of coarser tailings particles. Therefore, companies are developing new technologies to recover copper at coarser sizes.

- **Ore sorting.** Advanced sensors characterize individual particles and classify them.
- **Crossbelt analysis.** Crossbelt analysis by neutron activation determines if the ore is above the economic cutoff grade. Low-grade ore is rejected before entering the mill.
- **Coarse particle flotation.** Where liberation is achieved at coarse particle sizes, a low-quality concentrate can be recovered, decreasing the energy required to reduce all of the ore to the same size. The coarse concentrate is then reground and treated in a traditional flotation circuit. Cadia has announced that it plans on using the Eriez HydroFloat equipment for this process.
- **Rejection of pebbles.** Where pebbles are low grade, they can be rejected directly or rejected by ore sorters on the recycling stream.

Another tactic is to spend less specific energy to achieve the same final product size. Strategies include the following:



	Flotation Kinetics	Retention Time	Bubble Size (D_b)	Gas Holdup (E_g)	Recovery	Grade	Favored Particle Size	Water Recovery
J_g ↑	↑	↓	↑	↑	↑	↓	Coarse	↑
Froth Depth (forced air) ↑	↓	↓	—	—	↓	↑	Fines	↓
Froth Depth (self-aeration) ↑	?	↓	↑	↑	?	?	?	?
Frother ↑	↑	↓	↓	↑	↑	↓	Fines	↑
Collector ↑	↑	—	—	—	↑	↓	All Sizes	↓
Water ↑	↓	↓	—	↓	↓	↑	Fines	↓

Note: Kinetics will only increase until air recovery begins to drop. J_g is the superficial gas rate (volumetric air flow/cell area).

Courtesy of Flottec

Figure 25 Flotation operating strategy summary

- Intensive blasting. Mine operations use more intensive drilling and blasting of the ROM, creating increased fracturing, reducing the mill's feed particle size and grinding work index.
- Secondary crushing
- HPGRs
- Removal of tramp grinding ball steel using magnets on the mill discharge to reduce the amount of steel in the recirculating load.

PLANT OPERATING STRATEGIES

Although it is the honor-bound duty of every metallurgist to maximize recovery, investors in a mining operation are more interested in the return on capital. Most copper mines operate with the primary comminution equipment as the throughput bottleneck, and so these units are force fed, regardless of plant design criteria or optimum metallurgical recovery. Invariably a coarser primary grind results, giving a slightly lower copper recovery, but the higher tonnage disproportionately increases revenue. For example, an expansion of 10% tonnage may increase the primary grind P_{80} by 25 µm and decrease the flotation rougher residence time by 10%. Combined, they may decrease the rougher recovery by 2% and the final recovery by 1.5%, and may increase the cash flow of the operation by 8.5%.

In the short term, the other areas have to deal with the grade supplied by the mine and the tonnage and grind determined by the grinding area. The regrind and cleaner areas are often operated on a minimum concentrate specification basis. The cleaner flotation pH, reagent addition, and mass pull are configured to achieve high recoveries, only as long as the concentrate quality is met. In the medium and long term, plant modifications can be engineered to improve circuit performance, but budget priority is often given to increasing throughput before metallurgical recovery. As a result, it is important that plant metallurgists ensure that the flotation flow sheet, reagents, and hydrodynamics are optimized for selectivity and kinetics (Figure 25).

Instrumentation

Central control rooms fitted with distributed control system or supervisory control and data acquisition system interfaces are where the plant operators can monitor plant performance. Common instrumentation and control systems can include the following:

- Status, speed, or power load of mechanical equipment such as mills, pumps, and feeders.
- Monitor froth height, froth velocity, and air addition rates for flotation cells.
- Monitoring of major streams for flow rates, tonnages, and slurry densities.
- Monitoring pH and redox potential to control reagent addition.
- In-stream X-ray fluorescence (XRF) analysis of most major streams to chemical analyses (copper, zinc, lead, iron).
- Some plants have particle size indicators on the flotation feed stream.
- Closed-circuit television cameras on conveyor transfer points, sumps, crusher chambers, and other areas.
- Tailings and concentrate pipeline systems, including leak or blockage detection.
- Expert systems are commonplace for optimizing throughput of stable comminution circuits. They are also useful in maintaining froth heights by controlling the cascading surges in flow rates between cells by manipulating the intercell valves. Expert systems have not yet taken holistic operating control of flotation concentrators.

Concentrate Panning

Operators of smaller and older concentrators still practice the art of panning the concentrates to assess the separation efficiency and contaminant content to make process control decisions. Operators at larger and modern plants rely on in-stream XRF analysis of the concentrate streams and do not pan the concentrates.

Froth Cameras

Froth cameras are used extensively to monitor and control the froth removal rate (linear speed of overflowing froth) from the flotation cells. Control strategies vary, but most use a combination of airflow rate and froth height control to maintain a target froth removal velocity. Although the cameras can also monitor bubble size, bubble collapse rate, froth color, and texture, there has yet to be a general implementation of a control loop to use these parameters industrially.

Rougher Mass Pull

Capacities for the regrind and cleaner circuits are usually designed based on locked-cycle test results and scale-up from laboratory test work. Changes in tonnage, ore types, grind size, and liberation profiles mean that the duty of the industrial circuit can be quite different to design. In some cases, extra regrind capacity is required, but more often, regrind requirements are reduced, possibly because of a preferential overgrinding of the dense sulfide minerals due to overclassification in the cyclones. Generally, as long as the regrind and cleaning circuit retention times can be maintained, the rougher mass pull should be maximized to capitalize on metal recovery.

Control of froth height, frother addition, and air addition should be maintained to ensure constant removal of froth from all cells. Self-induced aeration cells can form a wave on the cell surface that can lead to unmineralized slurry overflowing into the concentrate, which should be avoided. Since the flotation feed grades of copper and iron sulfides vary, operating the rougher flotation to a target concentrate grade will lead to a loss in copper recovery.

Cleaner-Scavenger Recirculating Load

Since the cleaner flotation stages are typically run in closed circuit between high-recovery scavenger cells and high-grade concentrate recleaner cells, a recirculating load will form that must be managed. It is critical that the floatability copper minerals in the scavenger concentrate be increased to allow it to be recovered in the cleaner stages. The floatability of these particles can be increased by either

- Regrinding coarse middlings to improve the liberation of the copper minerals, or
- Adding additional collectors to increase the hydrophobicity of the particles.

Regrinding will not improve the floatability of the fine copper losses, which are often a substantial portion of the copper losses. Adding strong collectors that are less selective between copper and gangue minerals, such as pyrite, will increase the floatability of the copper and gangue, ultimately reducing the grade of the final concentrate.

The use of column cells for the final stages of concentrate cleaning are popular because a deep froth bed and froth wash can reduce entrained gangue minerals from the froth, decreasing the insoluble gangue grade of the final concentrate. However, traditional column cells can have trouble recovering fine particles because of the low shear mixing of the bubbles and slurry. Also, heavy particles such as native gold and copper, and particles of coarse copper sulfides, can have trouble maintaining bubble attachment up the column and through the froth.

CONCENTRATE MARKETING

Some of the older copper producers have integrated processes that produce copper sulfide concentrates from their own mines and process it into copper using their own pyrometallurgical (smelting or roasting) or hydrometallurgical processing (leaching) to produce London Metal Exchange–quality cathodes for sale to copper users. Most new copper concentrators focus on producing commercial-grade copper concentrates, which are then sold to custom smelters. These smelters pay for the concentrate based on the agreed copper price, copper content, copper grade, penalty element grades, credit element content, treatment, and refining fees. The net smelter return (NSR) is the amount paid by to the concentrate producer and is often used by mine planners in determining the value of a block of ore, expressed as NSR $/t.

Most commercial copper concentrates have between 22% and 32% Cu, but the penalty and credit element contents of the concentrates can alter the value significantly. The commercial terms are usually expressed in U.S. dollars per dry metric ton (US$/dmt).

Smelters and concentrate producers typically enter into medium- to long-term take-off contracts, where all or a portion of concentrate produced will be purchased by the smelter using previously agreed terms. A concentrate specification will be agreed to and deviation from that specification can result in penalties, renegotiation, or refusal to purchase the concentrate.

Downstream costs for concentrate producers can include the following:

- Transport to port
- Port fees
- Shipping fees
- Insurance
- Marketing fees

The transport and port fees are imposed on the wet concentrate mass, or wet metric tons (wmt). Insurance and marketing fees are often estimated as a percentage of the concentrate value.

The commercial terms with the custom smelter usually include the following:

- Payable metal
- Treatment charge ($/dmt)
- Refining charge ($/lb Cu)
- Precious metal credits (gold, silver)
- Penalty element deductions

The payable copper content is often the lower amount between 1% deduction in grade or 96.5% of the metal contained. For example, 1 t at 30% Cu:

$$1\ \text{t} \times (30 - 1)\% = 0.290\ \text{t Cu}$$

$$1\ \text{t} \times 96.5\% \times 30\% = 0.2895\ \text{t Cu}$$

Therefore, the payable copper is 0.2895 t Cu, because it is the lower amount.

Table 7 Concentrate terms

Species	Penalties, in US$
As	$2–6/dmt concentrate for each 0.1% above 0.1% $10–20/dmt concentrate for each 0.1% above 0.5%
Bi	$2–8/dmt for each 0.01% above 0.02% Bi
Cd	$2–6/dmt for each 0.01% above 0.02% Cd
Cl	$0.5–1.0/dmt concentrate for each 0.01% above 0.05% Cl
F	$1/dmt concentrate for each 0.01% above 0.03% F
Hg	$2–4/dmt concentrate for each 1.0 g/t above 10 g/t Hg
Pb + Zn	$2.5–5.0/dmt for each 1.0% above 6% (Zn + Pb)
Sb	$2–6/dmt for each 0.1% above 0.1% Sb

The payable gold content is often the lower amount between 1 g/dmt deduction in grade or 96% of the metal contained. In this example, if a concentrate contains <1 g/t Au, then no gold credits are paid. The payable silver content is often the lower amount between a 30-g/dmt deduction in grade or 90% of the metal contained. In this example, if a concentrate contains <30 g/t Ag, then no silver credits are paid.

Treatment charges are designed to pay for the smelting process and are typically between $60 and $120/dmt of concentrate. Refining charges are designed to pay for the electrorefining of the copper metal and are often dependent on the prevailing electrical energy cost, which is typically between $0.06 and $0.12/lb of payable copper. Refining charges can also apply to gold and silver credits too, typically between $5 and $10/oz for gold and $0.5 and $1.0/oz for silver.

Penalties are charged by smelters for undesired concentrate components that cause operational or environmental issues for the smelters. These penalties are usually expressed as a charge per unit grade above a threshold. There are upper limits for most of the species, beyond which sale of the concentrate can be difficult or prohibitively expensive (Table 7).

China has many of the world's custom smelters and has mandated that copper concentrates with more than 0.5% As cannot be imported into China. Few smelters will purchase copper concentrates with over 0.5% As. Producers have the following options:

- Sell the concentrate at a low price to a trader who will mix it with low arsenic concentrate stocks.
- Treat the concentrate using roasting or hydrometallurgical processes, then deal with the disposal issues of the arsenic by-products.

As the quality of concentrates continues to decline, while environmental standards increase, it is possible that it will become harder for concentrate producers to market their concentrates.

Moisture

Copper concentrate marketing can be affected by the moisture content of the concentrate to be shipped. If the moisture content is too low, potential dusting can lead to fire, safety, and environmental risks. If the moisture content is too high, it will increase transport charges and it may pass the transport moisture limit (TML). Above the TML, the concentrate may fluidize in a moving truck or ship, causing the vessel to become unstable and topple. Concentrate over this limit can be subject to costly drying before it can be shipped, slowing the sale of the concentrate.

Other Provisions

Other cost provisions include marketing (the personnel, legal, commercial, and logistical costs of concentrate sales), insurance, royalties, taxes, and losses (theft, spillage, dust).

WATER

Process water management for copper flotation concentrators can be complicated by reagents added, acid mine drainage, raw water quality, and soluble salts in the ore itself. Contaminated water must be either treated or excluded from the recycled process water regime. The use of seawater over potable water is becoming favored in arid areas to avoid conflict with competing water uses (Schumann et al. 2003).

Raw water sources include the following:

- **Fresh water from wells or water courses.** Where fresh water is plentiful and does not have competing uses for agriculture or population centers, fresh water is preferred for copper flotation because it is low cost, does not interfere with reagents, and does not pose issues for safety or maintenance. When fresh water is only available seasonally, storage in reservoirs can be required to ensure yearlong production.
- **Seawater.** Chile has established legislation requiring new copper concentrators to only use water from the ocean. In other areas, seawater can also be chosen where fresh water is not available or socially acceptable. Seawater has its own set of hydrodynamic character. For example, seawater or water with high ionic strength generates small bubbles, even without frother. Seawater can buffer lime so that pH <10 is difficult to achieve. Seawater can collapse the double layer, causing agglomeration problems. There are two approaches to the use of seawater:
  1. **Desalinated seawater via reverse osmosis (RO).** This approach achieves the previously mentioned benefits of fresh water, but with the added cost of the RO process, power transmission to the RO plant, and transport of the fresh water from the coast to the concentrator. In Chile, this can account for more than 20% of project capital cost for mines at more than 3,000 m above sea level. Desalinated water is used at Cerro Lindo in Peru, and Candelaria and Escondida, both in Chile.
  2. **Use of seawater directly in flotation.** Since lime is used to raise the pH in flotation, the salinity of process water is significantly lower than seawater. Several major operations have successfully used seawater for the flotation of copper and other base metals. Seawater is used at Texada (Canada), Batu Hijau, and Esperanza. The disadvantages include increased lime consumption, increased maintenance costs due to corrosion, increased complexity of molybdenum flotation, and the environmental risks associated with transporting the water and storing saline tailings in perpetuity. The higher specific gravity of seawater can increase transport costs to high altitudes, justifying the cost of desalination, and lower thickening rates. The concentrates also require washing to reduce the halide concentration before smelting.
- **Sewerage water.** Sewerage water can have a low social cost but can present operating difficulties and reduce metallurgical performance, given the presence of salinity,

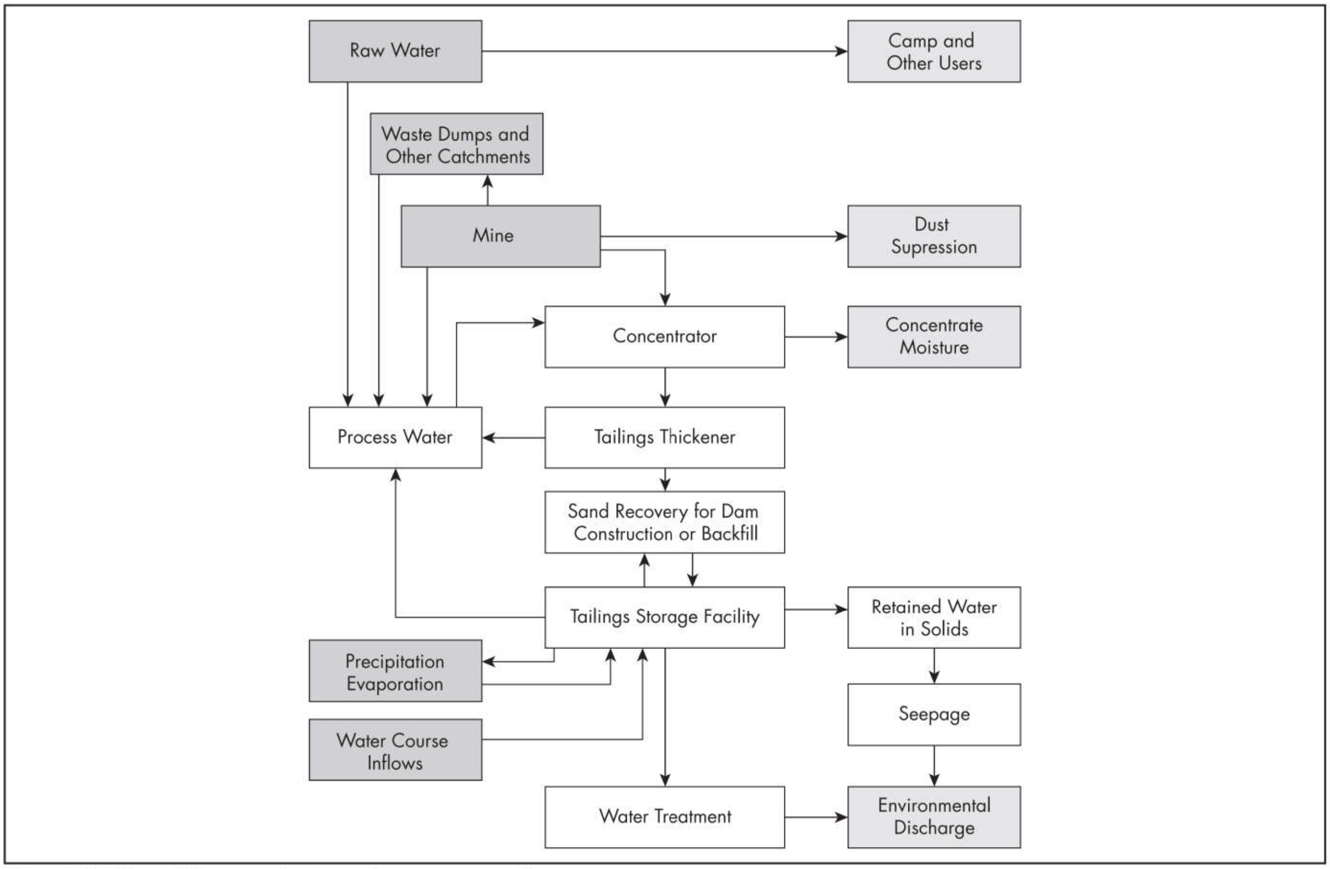


Figure 26 Site-wide water balance for a copper mine

detergents, chemical and biological oxygen demand, and colloids. Sewerage water is used at Cerro Verde and Cadia.

Further consideration should be given to other factors that can affect process water quality that may be missed during project development:

- **Acid generation from the mine, waste rock stockpiles, or tailings facility.** Iron sulfides can oxidize when exposed to air, adding ferric, sulfate, and other ions to water, potentially increasing lime consumption and affecting flotation recovery and selectivity.
- **Accumulation of reagents.** Water-soluble frothers, collectors, depressants, or their decomposition products may inhibit the easy recycling of water between flotation processes. Water from copper-molybdenum separation plants and copper-lead separation plants, for example, can be problematic in copper flotation.
- **Increased concentration of dissolved salts due to evaporation.** Evaporation in the tailings facility can concentrate the dissolved salts in the process water.
- **Ores in arid climates.** Ores in arid climates may have soluble components, such as copper, iron, and acid salts that may impact the metallurgical response.
- **Harsh waters.** Harsh waters can also affect reagent solution preparation as well. Many flocculants and organic depressants will not dissolve well in harsh waters, so fresh water is needed for preparation.
- **Buildup of ions.** The buildup of specific ions in the water can affect flotation. This is sometimes evident between winter and summer when the temperature differences can affect the solubility of various materials.

The water balance within the concentrator is usually of little consequence to the overall water consumption of the operation. Water consumption is typically dictated by the tailings disposal and storage arrangement (Figure 26). The balance between evaporation and precipitation in the mine, waste rock storage, and plant and tailings catchment areas will determine if the operation will have a positive or negative water balance. A negative balance will require raw water to be supplied. A positive water balance will require water to be treated to meet regulations and subsequently be discharged into the environment.

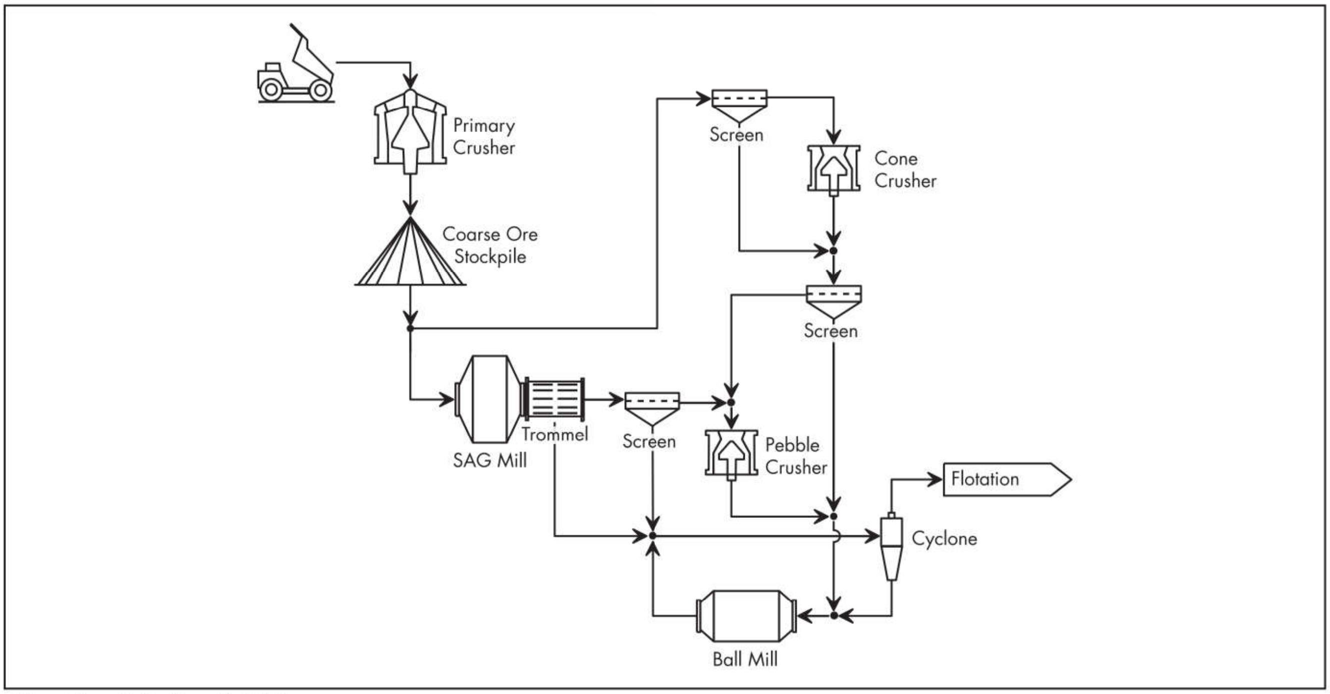
Where water is scarce, a reduction in the raw water requirement can be achieved by reducing water lost from tailings to evaporation and to encapsulation in the deposited tailings. The approximate net water consumptions per metric ton of ore treated are as follows:

- Conventionally subarial deposited tailings, >0.75 m^3/t
- Thickened tailings, 0.5–0.75 m^3/t
- Paste-thickened tailings, 0.25–0.5 m^3/t
- Filtered tailings, 0.15–0.25 m^3/t

LARGE-SCALE COPPER CONCENTRATORS

Antamina Copper-Zinc Mine, Peru

Antamina is a copper-zinc mine located in the Department of Ancash in the high Andes of Peru. The original concentrator was designed for 70,000–100,000 t/d, depending on ore types. A simple flow sheet for the Antamina mine is shown in Figure 27, and the major process equipment are listed in Table 8.



Adapted from Rybinski et al. 2011

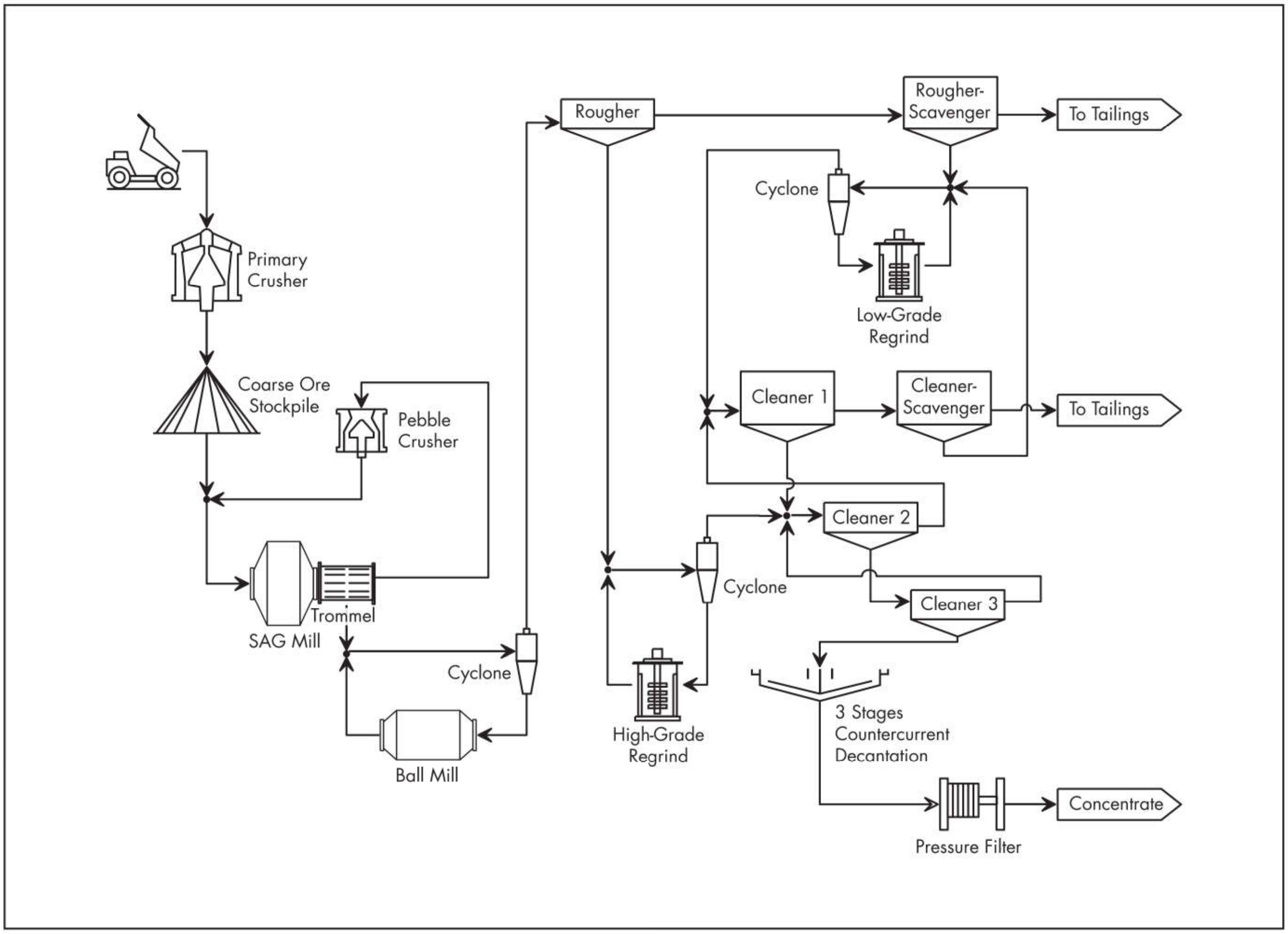
Figure 27 Simplified Antamina comminution flow sheet

Table 8 Major process equipment, Antamina

Equipment	Quantity	Equipment Description
Primary crusher	1	1.52 m × 2.26 m gyratory crusher, 750 kW
SAG mill	2	38 ft diameter × 19 ft effective grinding length, 20.1 MW
Ball mill	4	24 ft diameter × 36 ft, 11.2 MW
Pebble crushing	2	Cone crushers, 800 hp
Pebble crushing screen	2	10 ft × 24 ft double-deck screens
Cyclone	4 clusters	660-mm cyclones, 14 per cluster
Flotation cell, rougher, copper	32	Four rows of 8 cells of 130-m^3 forced-air machines
Flotation cell, cleaner-scavenger	5	One row of 5 cells of 130-m^3 forced-air machines
First cleaner cell, copper	6	4.3 m diameter × 14 m high column cells
Second cleaner cell, copper	4	4.3 m diameter × 14 m high column cells
Regrind mill, copper	2	VTM-1000, 1,000 hp
Flotation cell rougher, zinc	32	Four rows of 8 cells of 130-m^3 forced-air machines
Flotation cell, cleaner-scavenger	5	One row of 5 cells of 130-m^3 forced-air machines
First cleaner cell, zinc	7	4.3 m diameter × 14 m high column cells
Second cleaner cell, zinc	3	4.3 m diameter × 14 m high column cells
Regrind mill, zinc	2	VTM-1000, 1,000 hp
Concentrate thickener, copper	1	35 m diameter
Concentrate thickener, zinc	1	35 m diameter
Concentrate thickener, bulk	1	25 m high rate
Concentrate thickener, molybdenum	2	6 m diameter
Concentrate filter	4	Pressure filters PF144

Batu Hijau Copper Mine, Indonesia

Batu Hijua is a copper-gold mine located on the Indonesian island of Sumbawa in Southeast Asia. The original concentrator was designed for 80,000 t/d and uses seawater. A simple flow sheet for the Batu Hijau mine is shown in Figure 28, and the major process equipment are listed in Table 9.



Adapted from DeMull et al. 2001

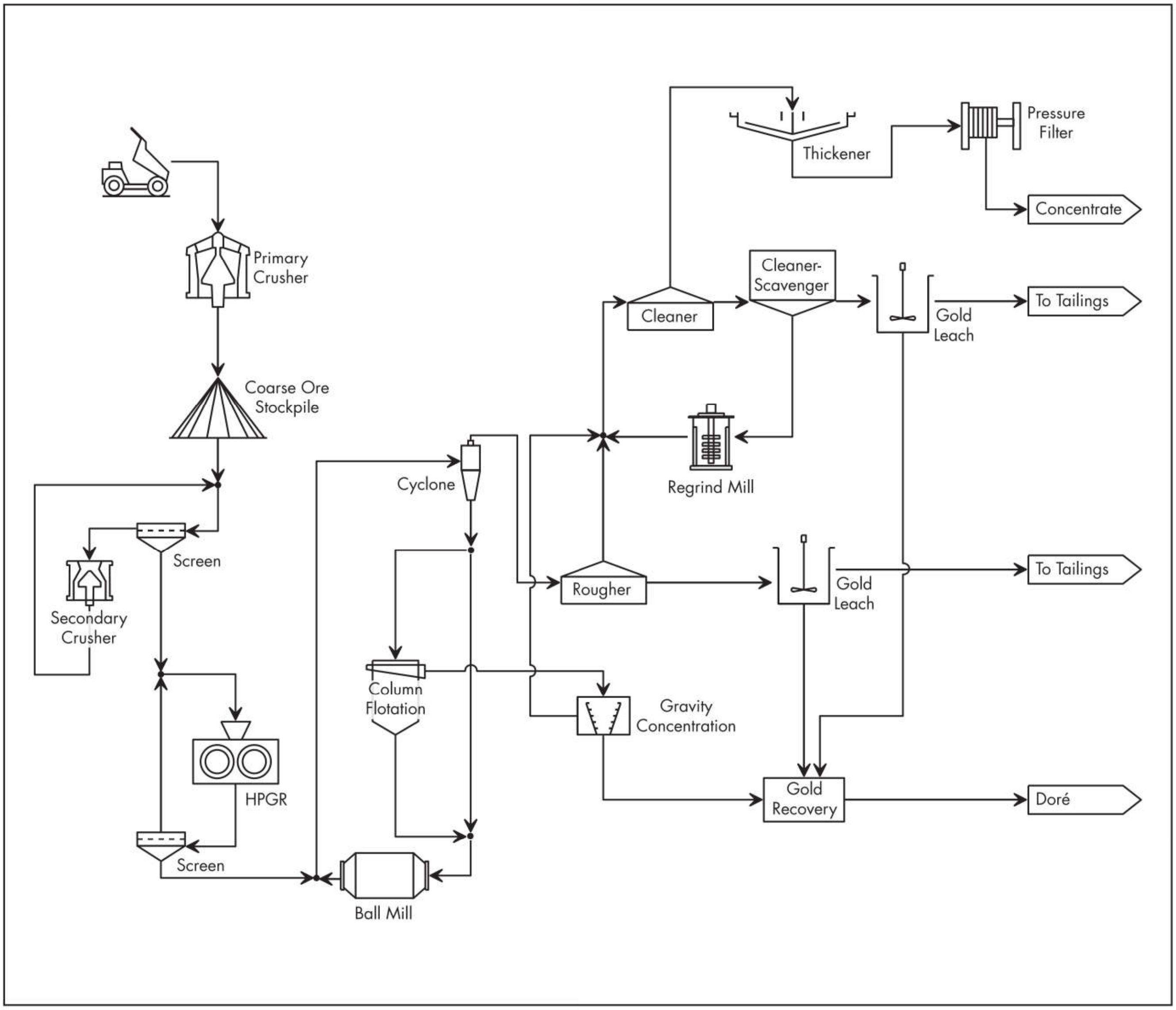
Figure 28 Simplified Batu Hijau flow sheet

Table 9 Major process equipment, Batu Hijau

Equipment	Quantity	Equipment Description
Primary crusher	2	60 in. x 89 in. gyratory, 750 kW
Pebble crusher	2	1,000-hp cone crushers
SAG mill	2	36 ft × 19 ft, 13,423 kW
Ball mill	4	20 ft × 33.5 ft, 6,711 kW
Cyclone	4 clusters	33-in.-diameter cyclones
Flotation cell	50	127-m^3 self-aspirating cells
First cleaner	4	42.5 m^3
Second cleaner	10	14.2 m^3
Third cleaner	4	14.2 m^3
Cleaner-scavenger cell	4	42.5 m^3
Regrind	2	VTM-1250
Polishing mill	1	500 hp
Thickener	3	25 m diameter
Filter press	2	Pressure filters

Boddington Copper Gold Mine, Western Australia

Boddington is a copper-gold mine northwest of Boddington in Western Australia. The concentrator was designed for approximately 100,000 t/d. In 2017, the plant produced 787,000 oz Au and 80 million lb Cu. The comminution circuit incorporates three-stage crushing with HPGRs and ball mills. A simple flow sheet for the Boddington mine is shown in Figure 29, and the major process equipment are listed in Table 10.



Adapted from Dunne et al. 2007

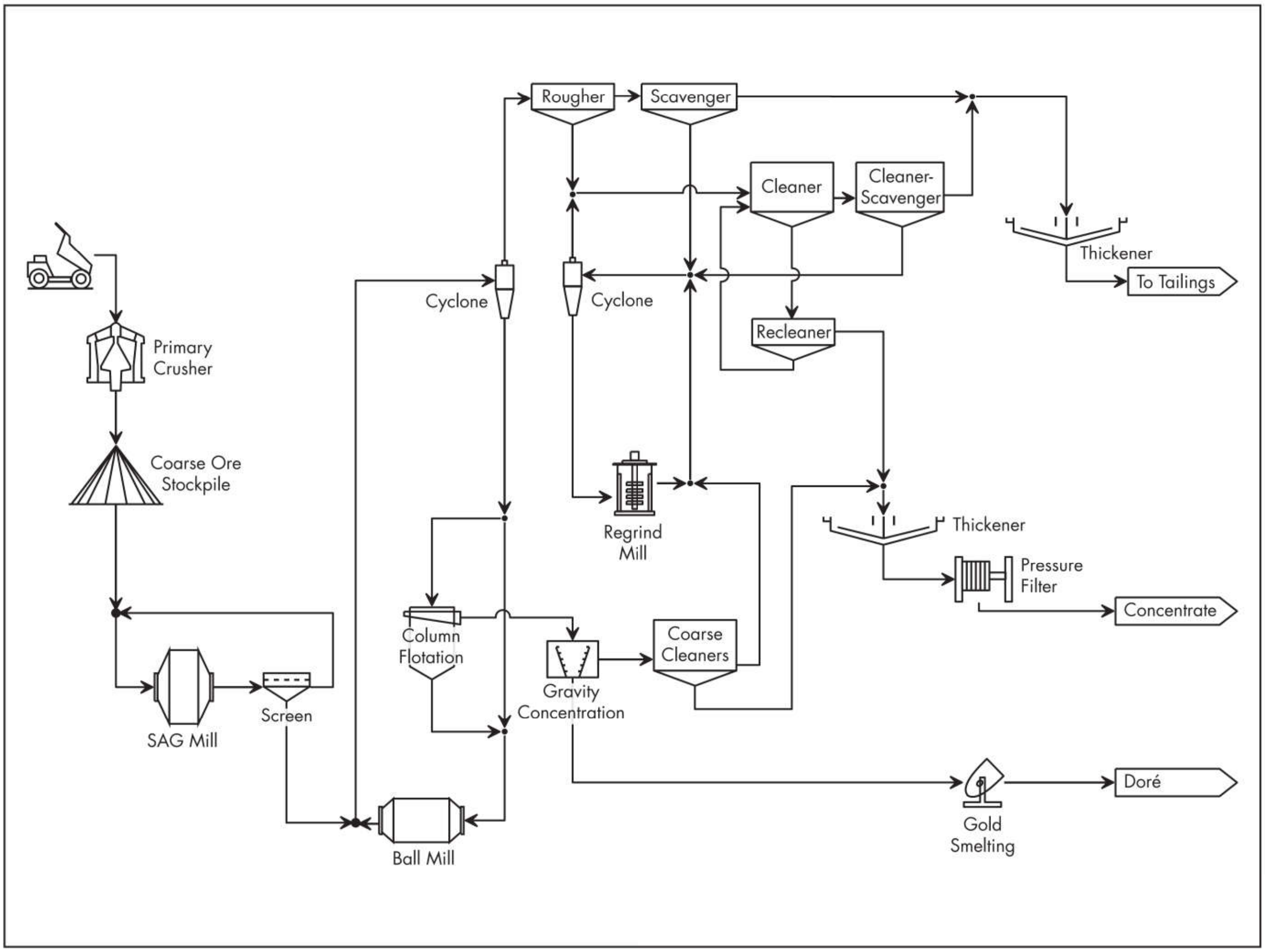
Figure 29 Simplified Boddington flow sheet

Table 10 Major process equipment, Boddington

Equipment	Quantity	Equipment Description
Primary crusher	2	60 in. × 113 in. gyratory
Coarse ore stockpile	1	40,000 t live
Banana screen	4	3.6 m × 7.3 m single-deck
Secondary crusher	6	1,000 hp cone crusher, 746 kW
HPGR	4	2.4 m diameter × 1.65 m 2 × 2.8 MW drives
Wet screen	8	3.66 m × 7.93 m (aperture of 9–10 mm)
Ball mill	4	7.9 m × 13.4 m, 16 MW
Primary cyclone	26	12 cyclones per cluster

Cadia Copper-Gold Mine, Australia

Cadia is a copper-gold mine located close to Orange, New South Wales, in Australia. The original concentrator was designed for ~47,000 t/d. The flash flotation cells in the comminution circuit treat ~50% of the circulating load, and the gold in the flotation concentrate is recovered by a centrifugal gravity separator. A flow sheet for the Cadia mine is shown in Figure 30, and the major process equipment are listed in Table 11. Cadia had the world's first 12.2-m SAG mill, which has been the basis for important SAG milling models.



Adapted from Cesnik 2009

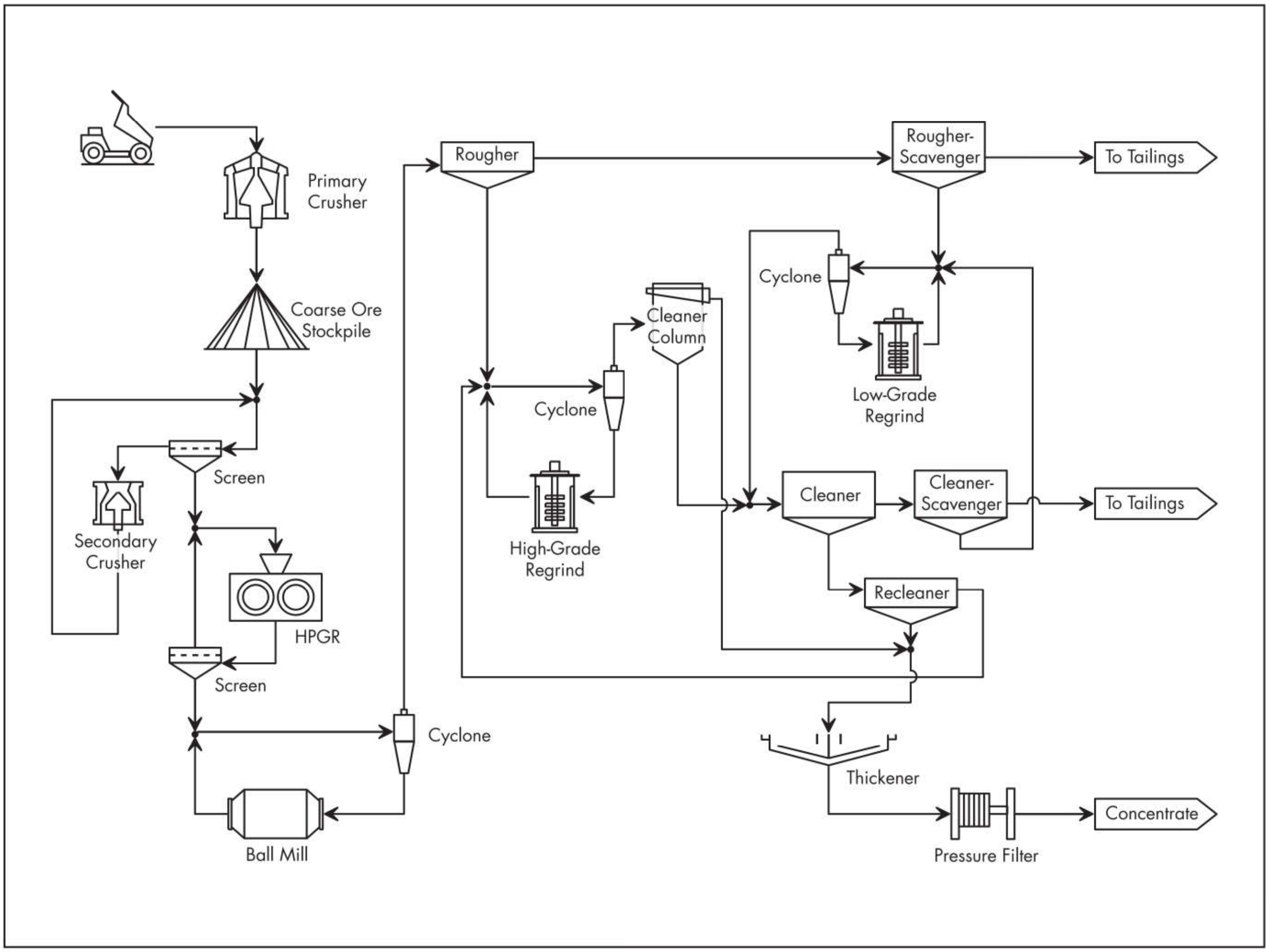
Figure 30 Simplified Cadia flow sheet

Table 11 Major process equipment, Cadia

Equipment	Quantity	Equipment Description
Primary crusher	1	60 in. × 110 in. gyratory
SAG mill	1	12.2 m × 6.1 m effective grinding length, 20 MW
Ball mill	2	6.7 m × 10.97 m; 8,760 kW
Rougher flotation	14	Two rows of seven 150-m^3 forced-air machines
Regrind mill	1	Vertimill, VTM400
Flash flotation cell	1	SK1200 flash float
Cleaner flotation	6	30-m^3 forced-air machines
Scavenger-cleaner flotation	4	30-m^3 forced-air machines
Recleaner flotation	4	8.5-m^3 forced-air machines
Tailings thickener	1	53 m diameter
Concentrate thickener	1	12 m diameter
Concentrate filter	1	Plate-and-frame, VPA1540/24 press

Cerro Verde Copper Mine, Peru

Cerro Verde is a copper-molybdenum mine located close to Arequipa southwest of Peru. The original concentrator was designed for 108,000 t/d, and a second plant was built to treat an additional 240,000 t/d. The comminution circuit consists of three-stage crushing with HPGRs and shell-mounted ball mills. A simple flow sheet for the Cerro Verde mine is shown in Figure 31, and the major process equipment are listed in Table 12.



Adapted from Vanderbeek et al. 2017

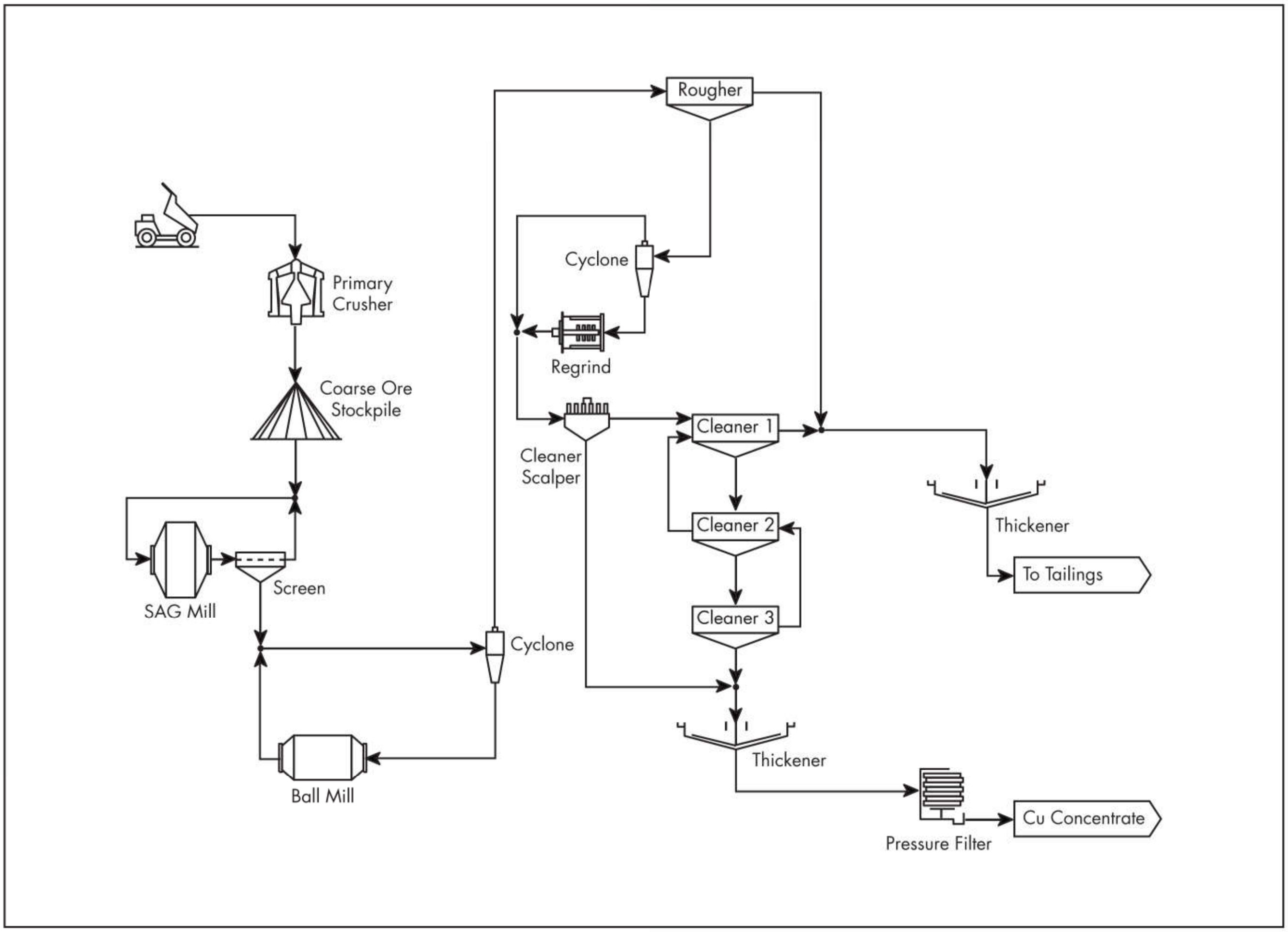
Figure 31 Simplified Cerro Verde Concentrator II flow sheet

Table 12 Major process equipment, Cerro Verde Concentrator II

Equipment	Quantity	Equipment Description
Primary crusher	2	1.5 m × 2.9 m gyratory crushers, 746 kW
Secondary screen	8	3.6 × 7.9 m double-deck banana screens
Secondary crusher	8	Cone crushers, 1,250 hp each
Tertiary crushing, HPGR	8	2.4 m diameter × 1.65 m wide, 2.5-MW twin drives
HPGR screen	12	3.66 × 8.5 m double-deck wet screens
Ball mill	6	8.2 m × 14.6 m, shell-supported 22-MW gearless drives
Primary cyclone	6	Sixteen 33-in.-diameter cyclones
Rougher flotation	6 rows	Nine 257-m^3 self-aspirating mechanical cells per row
Cleaner-scavenger cell	12	Two rows of six 257-m^3 cells
Final column cleaner	6	4.88 m diameter × 12 m high
Concentrate thickener	1/1/1	35 m diameter for copper and 45 m diameter for bulk concentrate, plus 35 m diameter for clarifier
Tailings thickener	4	80 m diameter

Prominent Hill Copper Mine, South Australia

Prominent Hill is a copper-gold mine located in Australia. It processes approximately 8 million t/yr of copper ore and has Jameson flotation cells in the rougher circuit. A simple flow sheet for the Prominent Hill mine is shown in Figure 32, and the major process equipment are listed in Table 13.



Adapted from Woodward et al. 2013

Figure 32 Simplified Prominent Hill flow sheet

Table 13 Major process equipment, Prominent Hill

Equipment	Quantity	Equipment Description
Primary crusher	1	60 in. × 89 in. gyratory, 600 kW
Coarse ore stockpile	1	30,000 t live, 130,000 t total
SAG mill	1	10.36 m diameter × 5.18 m effective grinding length, 12 MW
SAG discharge screen	12	3.66 m × 8.5 m double deck wet
Ball mill	1	7.3 m diameter × 10.5 m, 12 MW
Primary cyclone	1 cluster	Cluster of 16 × 840 mm cyclones
Rougher flotation	6	150-m^3 forced-air tank cells
Rougher concentrate regrind mill	1	M10,000 IsaMill
Concentrate scalper	1	J5400/18 Jameson cells
First cleaner flotation	8	50-m^3 forced-air tank cells
Second cleaner flotation	6	20-m^3 forced-air tank cells
Third cleaner flotation	4	20-m^3 forced-air tank cells
Tailings thickener	1	45 m high rate
Concentrate thickener	1	23 m high rate
Concentrate thickener	1	Horizontal plate pressure filter, 156 m^2

Other Concentrators

Other large copper concentrators include the following (including sources where additional information can be obtained):

- Alumbrera copper mine, Argentina (Morrell and Munro 2010)
- Candelaria copper mine, Chile (Marsden and Ogonowski 1999)
- Constancia copper mine, Peru (Klohn et al. 2016)
- Escondida copper mine, Chile (Bergholz and Schreder 2004)
- Grasberg copper-gold mine, Indonesia (Veloo and Coleman 1996)
- Lumwana copper mine, Zambia (Araya et al. 2014)
- Phu Kham copper mine, Laos (Hoyle et al. 2013)
- Salobo copper mine, Brazil (Godoy et al. 2010)
- Sossego copper mine, Brazil (Mendonça et al. 2015)

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