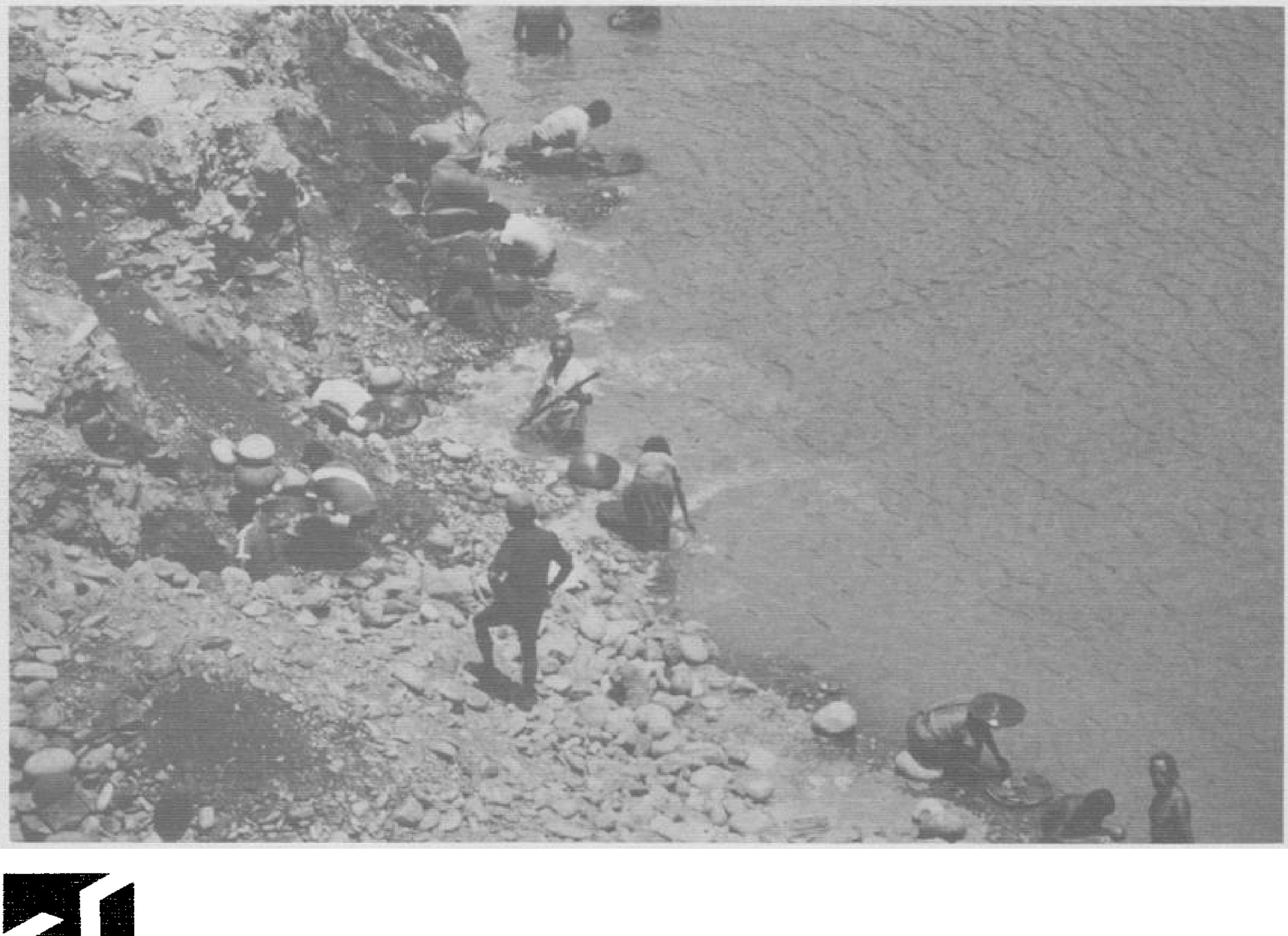
Michael Priester, Thomas Hentschel

Small-Scale Gold-Mining

Aus der Arbeit von gtz



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Small-Scale Gold-Mining

Processing Techniques in Developing Countries

A Publication of Deutsches Zentrum für Entwicklungstechnologien – GATE
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Introduction

In small-scale gold mining, the raw ore containing only about 1 - 100 g Au per ton has to be concentrated into marketable products containing at least about 90 % Au. The requisite processes are diverse, complicated, and often involve the use of toxic reagents (cyanides, mercury, long-chain hydrocarbons, etc.). In the interest of operational safety and better technoeconomic results, the global small-scale gold mining community is seeking advice and assistance. Additionally, the issue of environmental pollution by mercury contamination is preparing the way for alternative processing techniques with which to solve, or at least mitigate, the problem.

This book describes in condense, compact form the various gold-specific problems and technical strategies for their solution.

The target group of this publication comprises relevant umbrella organizations, small-scale miners, consultants to the small-scale and cooperative mining sector, NGO and GO planners, and craftsmen/workshops involved in the local manufacture of pertinent equipment.

1. PROBLEMS

1.1 Global Distribution of Small-scale Gold Mining by Regions

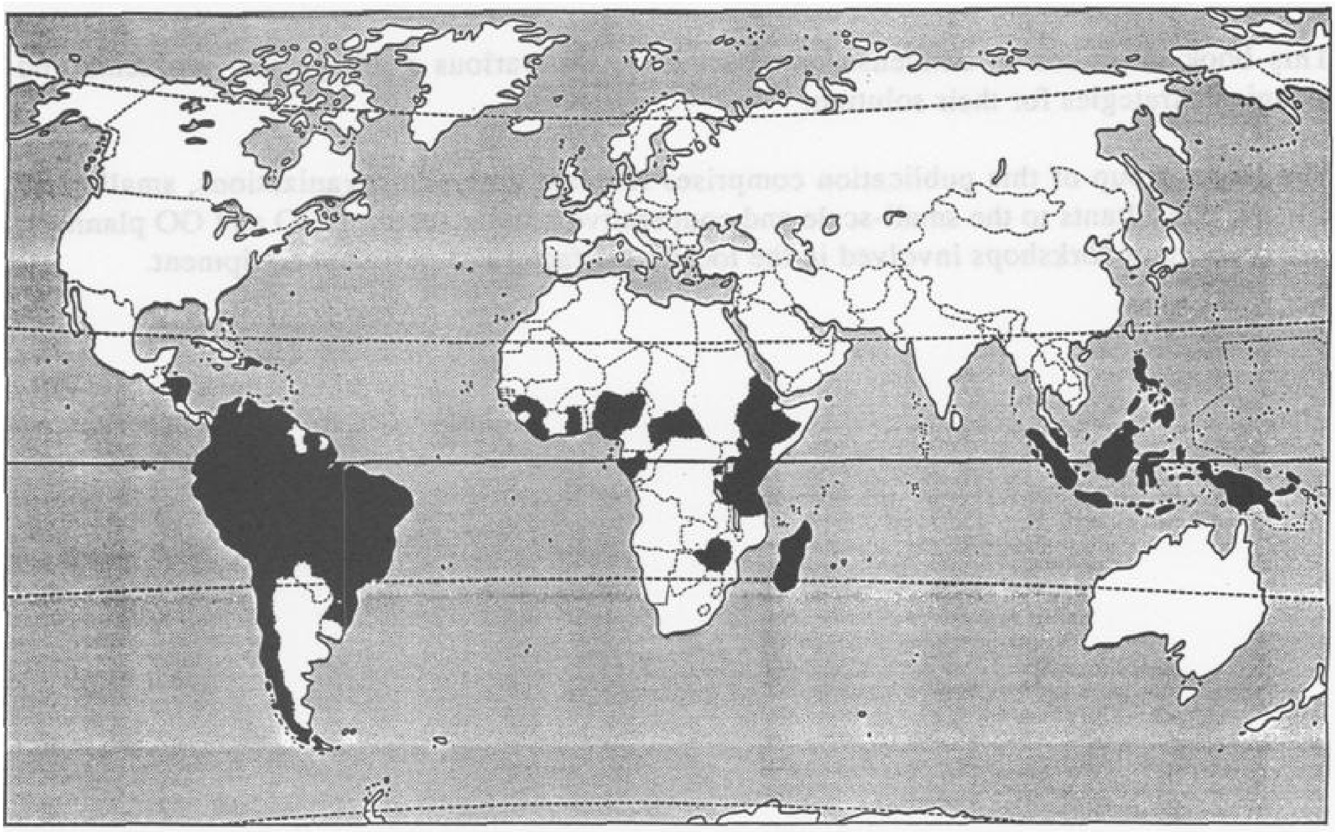
Most of the world's small-scale gold mining operations are situated in

Latin America, i.e., Honduras, Nicaragua, Colombia, Ecuador, Peru, Bolivia, Chile, Brazil, Suriname, Venezuela and the Domonican Republic

Africa, i.e., Ghana, Kenya, Tanzania, Zambia, Zimbabwe, Ethiopia, Guinea, Liberia, Nigeria, Gabon, the Central African Republic, Burundi and Madagascar

Asia, i.e., India, China, the Philippines, Papua New Guinea, Indonesia and Malaysia.

The aforementioned countries are shown in dark color on map 1.



Map 1: Countries in which substantial small-scale gold mining activities occur (dark colored areas)

In Brazil alone, more than a million people are directly involved in gold mining. Hence, the gold mining sector often constitutes a major source of employment in rural areas.

1.2 Generic Problems of Small-scale Mining

Small-scale mining is an essentially artisanal or small-industrial form of raw material extraction. It is characterized by the following phenomena:

little or no mechanization in the form of machines and engines, resulting in a large proportion of heavy manual labor,

low safety standards,

generally low level of training,

lack of technicians, resulting in deficient planning and organization of extraction and processing activities,

relatively poor exploitation of available resources due to selective extraction of rich ores in combination with low re-recovery rates, i.e., specific yields,

- low wages,
- low labor productivity,
- in part only seasonal employment in mines, or only as long as the world market prices are appropriately high,
- little awareness of environmental hazards,
- chronic shortage of capital,
- widespread illegal activities as a result of unfavorable mining laws and due to a lack of mineral rights/licensing.

Such problems frequently lead to a situation in which small-scale miners find themselves caught up in a vicious circle of economic bottlenecks, e.g., lack of working capital, with external assistance as the only possible path of escape.

A frequent and crucial technical and economic problem complex is the dressing of gold ore into markteable products. As a rule, the ores are of very low grade (cf. ch. 2.2) and require concentration by a factor of as much as 1 : 1 000 000.

1.3 Ore Dressing Problems in Small-scale Mining

Gold miners have a choice of three options for processing their raw ore:

- dressing the ore with their own processing equipment,
- employing the services of mobile dressing equipment or
- taking their ore to a central processing plant.

As closer analysis reveals, however, each of those three alternatives has certain drawbacks:

Operating one's own processing equipment

Most operators of small-scale and artisanal gold mines are liable to face major financing and dimensioning problems, if they decide to invest in their own dressing equipment. Basically, the situation can be expected to develop as follows:

a) The more modern types of dressing equipment and the individual machines and structural elements of which they are comprises are not available on the market with capacities below about fifty to seventy tons per day. There is a global absence of suitable mini-size equipment for small-scale mining. Many mining operations handle only 5 to 50 tons of raw ore a month and therefore have no use for modern (conventional) processing equipment, which requires ten times higher throughputs. In addition, the installation of new equipment frequently entails follow-up investments that put the operators under such a high cost burden, that financing becomes impossible for the smaller mines.

b) Small-scale and mini-scale mines suffer under a chronic shortage of capital and, as such, are rarely in a position to bare the high initial cost of conventional dressing systems. Due to the limited size of national markets, most equipment has to be imported, and the import duties add to the already high cost of investment.

c) Two other factors which further reduce the marginal earnings of many mining operations are the lack of technical assistance for sizing, planning and optimizing the requisite equipment and, hence, correspondingly low recovery rates.

Intermittent use of mobile dressing equipment

While various of mobile dressing equipment are being developed, most of it is still at the experimental stage and does not yet meet state-of-the-art requirements. The successful, cost-effective use of such equipment depends on:

a) The presence of a substantial number of small-scale mines within a limited area, i.e., relatively short hauling distances that ensure sufficient feed quantities for operating the mobile equipment.

b) Extensive raw-ore homogeneity within the service area (mining district, because most mobile equipment relies on relatively simplified dressing processes and therefore react sensitively to any change in feed.
For isolated mines, the above considerations practically rule out any chance of access to mobile dressing equipment for getting their raw ore concentrated.

Transport to a central processing plant

Raw ore is by nature a relatively low-value item of trade. Considering that the net product derives from concentrating the raw material into valuable concentrates, it would be counterproductive for miners to market their raw ore, particularly in countries with underdeveloped infrastructure, where the cost of haulage is extremely high.

If the raw ore is sold on the world market, the cost of transportation may even be significantly higher. Consequently, miners should always try to avoid the added cost of hauling country rock (valueless waste) by concentrating the raw ore at a point situated as close as possible to the mine.

Gold miners in particular have the option of marketing preconcentrates or tailings, especially if there is a large leaching plant in the near vicinity. In such cases, the miner only has to dress his ore to the extent of extracting the easily recoverable final concentrate from the crude, intergrown material, and then sell the tailings or any preconcentrates he may have produced.

Such an approach calls for appropriate intermediary ore-dressing technology. The techniques described furtheron incorporate historical, traditional and modern elements of gold processing as means of meeting the demand. In addition, the publication tries to show locally manufactured equipment for gold ore processing as socioeconomically appropriate low-cost solutions for technical problems in small-scale mining.

1.4 Siting Considerations

Gold ore dressing equipment should always be located with a view to obtaining a good slope and ample supply of water, the latter mainly for wet-mechanical gravitative dressing processes, but also for leaching plants, amalgamating plants, etc. Additionally, water often serves as a source of energy for small-scale ore processing, and a substantial vertical deflection is therefore useful for exploiting hydraulic energy. At the same time, the dressing processes can be arranged to exploit the force of gravity for moving the product from the top end (feed) of the system to the bottom end (discharge), e.g., in the pulp, without need of outside energy.

2. GOLD AS A RAW MATERIAL

2.1 General Characteristics

Gold, a well-known precious metal, was one of the first products of mining. Being a metal, gold is a good conductor of heat and electricity. It has a conspicuous yellow color and accounts for approximately 0.000 000 5 % of the earth's crust substance; this corresponds to a concentration of about 0.005 g/t. Ocean water contains dissolved gold in a concentration of 0.01 to 0.05 mg/m^3. Metallic gold is nonmagnetic, soft and malleable. In its natural form, gold practically does not oxidize, as evidenced by its bright luster. Because of its mechanical and chemical properties, gold is very resistant to weathering and tends to accumulate in secondary deposits (cf. ch. 2.2). Gold melts at 1063° C. It is a very heavy metal: pure gold has a specific gravity of over 19 g/cm^3. The density of natural gold declines with increasing silver content, resulting in a 15 to 19 g/cm^3 range of variation. The chemical properties of gold impart good alloyability with several other precious metals, including silver and mercury. Gold dissolves only in strong oxidizing agents such as aqua regia or in the presence of halogens or complexing agents, e.g. cyanide solutions.

2.2 Natural Occurrence of Gold

Natural gold occurs mainly in primary deposits called gold-quartz veins, which may be of plutonic or subvolcanic origin and embedded in plutonites, vulcanites, metamorphites or sediments as country rock, and enriched veins may form by mechanical or chemical action in cap zones, oxidation zones or cementation zones. Sulfidic deposits in part displaying rich gold mineralization due to pneumatolytic or replacement processes are of minor significance for small-scale mining. In secondary deposits, gold accumulates as a result of weathering and transport. Differentiation is made between fossil gold placers, e.g., the gold deposits of Witwatersrand, South Africa, or the Reefs, and recent placers, i.e., those of more recent origin. While fossil placers nearly always yield highly consolidated, extremely hard ore with processing properties that hardly differ from those of primary ore, recent placers display practically all gradations between loose, slightly consolidated and consolidated sediments.

2.3 Properties of Raw Ore

The gold content of a given deposit is strongly dependent on the type of genesis. In case of **primary deposits**, the economic extraction threshold is normally assumed as roughly 5 g/t. The maximum geological gold content of any deposit is probably about 100 g/t. That would necessitate natural enrichment processes reaching a factor of 1000 in order to sufficiently concentrate statistically distributed gold contents into profitable gold deposits.

The gold content of **secondary deposits** range between approximately 0.1 g/t as the lower economic limit and 5 g/t as the upper geological limit.

With regard to mineralogy, small-scale mining is almost wholely dependent on deposits containing free gold as the ore mineral. Indeed, these are the most frequent from the standpoint of economic geology.

Some primary deposits of subvolcanic origin contain gold compounds resulting from the formation of mineral alloys between gold, silver and selenium, e.g., in sylvanite, nagyagite, petzite and calaverite.

There is an even more frequent occurrence of minerals in which gold has been incorporated into the crystal defects of other minerals, e.g., tetrahedrite, argentite, polybasite, pyrargyrite, proustite and freibergite. Normal ore processing techniques are inadequate for recovering such gold.

Placer gold is almost exclusively free gold, the other minerals already having been removed in the course of translocation by reason of their solubility, cleavage or weathering behavior.

2.4 Intergrowth of Gold-bearing Minerals

In primary deposits, the gold-bearing mineral constituents are usually found to be irregularly scattered throughout the quartzose matrix ore, and the gold and/or quartz may be intergrown with accompanying minerals. As a rule, particularly large particles of mountain gold are found embedded in quartz. Crystals measuring more than 10 mm across sometimes occur. Particularly small particles of gold (1 - 100 µm) are mostly situated in the intergranular voids of sulfidic accompanying minerals.

Intergrowth is extremely rare in placer deposits, where the normally already liberated gold occurs in particle sizes ranging from a few tens of microns and several centimeters. In some exceptional cases, individual nuggets weighing more than 1 kg have been found. The conditions of transport are yet to be fully clarified. It is presently assumed that this type of gold enrichment results primarily from the electro-physical precipitation of dissolved gold. Very infrequently, the gold is encrusted by or intergrown with iron minerals.

2.5 Accompanying Minerals

The most significant accompanying minerals in the unweathered veins of primary deposits include the following:

- quartz	- calcite
- feldspar	- siderite

(as gangue material).

And:

- pyrite	- arsenopyrite
- chalcopyrite	- pyrrhotite
- marcasite	- molybdenite
- galena	- sphalerite
- stibnite	- bismuth minerals
- free silver	

(in addition to the aforementioned silver and gold minerals).

In weathered lodes, the quartz residue is accompanied mainly by limonite and weathered accompanying minerals, e.g., jarosite, cerussite, etc.

The mineral composition of placers is often much more complicated. The following table surveys the essential accompanying minerals in gold placers:

Table: Potential composition of gold placers, incl. essential properties of constituents

Mineral	Chemical formula	Color	Density [g/cm^3]	Hardness (Mohs')	Remarks
Free gold	*Au(+Ag)*	golden yellow	15.6-19.3	2.5	
Magnetite	Fe_3O_4	black	5.2	5.5-6.5	glossy, very magnetic
Ilmenite	$(Mg,Fe)TiO_3$	black	4.5-5	5.6	weakly magnetic
Garnet	$M^{2+}{}_3M^{3+}{}_2(SiO_4)_3{}^2$	red, brown	3.8	6.5-7.5	glassy sheen, rounded crystals
Zircon	$ZrSiO_4$	brown, light yellow, uncolored	4.7	7.5	diamond sheen
Hematite	Fe_2O_3	dark steel-gray, black	4.9-5.3	5.5-6.5	rounded particles
Chromite	$FeCr_2O_4$	iron-black to brownish black	4.1-4.9	5.5	possibly weakly magnetic
Peridote	$(Mg,Fe)_2SiO_4$	olive green	3.3-3.4	6.5-7	glassy sheen, transparent to translucent, good cleavability
Epidote	$HCa_2(Al,Fe)_3Si_3O_1V_3$	pistachio green	3.2-3.5	6.7	
Pyrite	FeS_2	bronze yellow	4.9-5.1	6-6.5	angular particles, metallic sheen
Monazite	$(Ce,La,Di)PO_4 + ThO_2$	yellow	4.9-5.3	5-5.5	resinous or greasy sheen, rounded particles
Limonite	$2Fe_2O_33HO$	dark brown	3.6-4.O	5-5.5	
Rutile	TiO_2	reddish-brown, red	4.2	6-6.5	metallic diamond sheen
Platinum	Pt(poss. also Ir)	steel white	16.5-18	4-4.5	malleable, flour + granules
Iridium	Ir(also Pt, etc.)	silver white, tinge of gray	22.6-22.8	6.7	angular particles
Iridosmine	Ir, Os	tinny white to light stheel gray	19.3-21.1	6.7	tabular particles, tough, good cleavability
Cinnabar	HgS	red	8-8.2	2-2.5	

Wolframite	(Fe,Mn)WO_4	black, dark gray	7.2-7.5	5-5.5	semimetallic sheen, good unidirectional cleav-ability
Scheelite	$CaWO_4$	white, light yellow, brown gray	5.9-6.1	4.5-5	diamond-to-greasy sheen, translucent
Cassiterite	SnO_2	brown or black	6.8-7.1	6-7	brittle, rounded particles
Corundum	Al_2O_3	brown, yellow	3.9-4.1	9	diamond-to-glassy sheen
Sapphire	Al_2O_3	blue	3.9-4.1	9	diamond-to-glassy sheen
Ruby	Al_2O_3	red	3.9-4.1	9	diamond-to-glassy sheen
Diamond	C	white, uncolored, pale	3.5	10	diamond-to-glassy sheen
Free mercury	Hg	tinny white	13.6		small, opaque, liquid globules
Amalgam	Hg, Ag, Au	silvery white	13-14		brittle to tough
Galena	PbS	lead gray	7.4-7.6	2.5-2.7	metallic sheen, very good cubic cleavability, friable
Silver	Ag	silvery white	10.1-11.1	2.5-3	rough, malleable, black tarnish
Copper	Cu	reddish brown	8.8-8.9	2.5-3	tough, flexible
Bismuth	Bi	tinny white	9.8	2.5	brittle, metallic sheen
Cerussite	$PbCO_3$	uncolored, white	6.5	3-3.5	diamond sheen
Columbite	(Fe,Mn) $(Nb,Ta)_2O_6$	iron black, gray,	5.3-7.3	6	iridescent, semimetallic
Tantalite		brownish black			sheen, good cleavability
Quartz	SiO_2	uncolored	2.6	7	glassy or greasy sheen, uncleavable
Feldspar	silicates with K, Na, Ca, Al etc.	uncolored, white, light yellow, creme pink	2.5-2.7	6-6.5	good cleavability, glassy sheen

Bold-face minerals may be economically exploitable and separately marketable.

3. GOLD ORE DRESSING TECHNIQUES

The term dressing covers all the various processes for enriching raw ore to obtain marketable concentrates. This includes mechanical processes, e.g., wet mechanical dressing, which do not alter the mineral constitution, and chemical processes, e.g., leaching, in which the valuable mineral is transformed into a new chemical compound. In addition to separating the valuable mineral from the valueless waste for the purposes of concentration, also referred to as sorting, dressing also includes preparatory processes and afterprocessing, e.g., size reduction, classification, drying, etc.

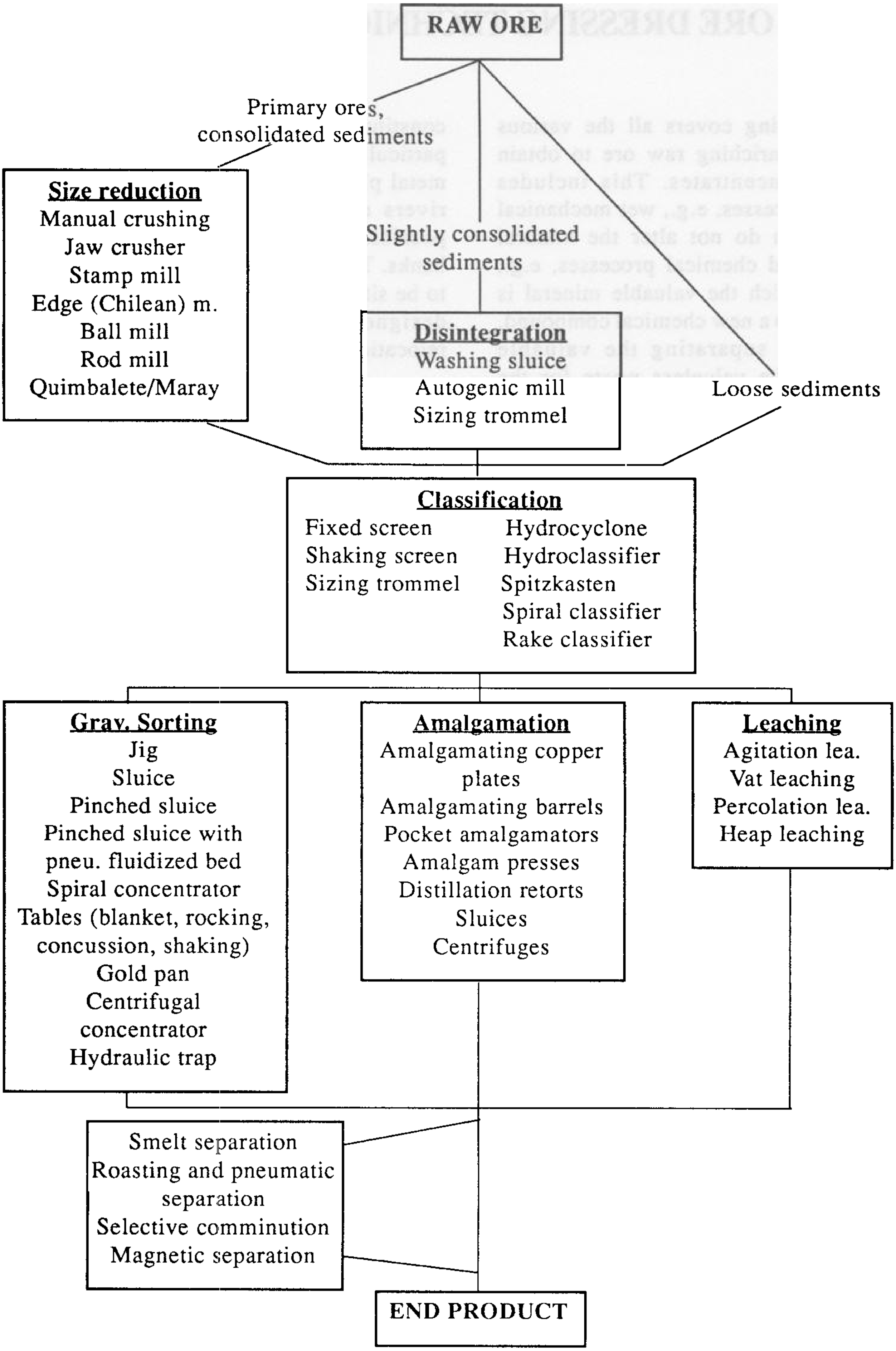
Crucial identifiers for successful dressing are the concentration factor, recovery rates for mass and values, and the concentrate contents.

The task of dressing precious-metal raw materials imposes strict standards on the quality of sorting. This is due to the physical and geochemical/economic-geological properties of gold. As a rule, primary deposits have raw ore contents ranging from 100 - 200 g/t at maximum to 1 - 2 g/t at minimum. The bottom limit represents the cutoff point for deposits deemed just barely exploitable. Sedimentary deposits have raw ore contents starting at about 0.2 g/t and reaching 20 - 50 g/t. Consequently, the dressing process must achieve concentration factors in the area of 100 000 and higher. At the same time, comparatively large quantities of raw ore must be obtained and processed in order to cover the cost of extraction and dressing. In most developing countries, a combination of low wages and large proportion of manual labor leads to a situation in which mining may be regarded as economically acceptable down to a level of 0.3 g recovered gold per man-shift. The fact that many mining operations are subject to seasonal shutdown constitutes a further gold dressing problem, particularly for ore obtained from precious-metal placers. During the rainy season, the rivers often run so deep, that it is not possible to work on the riverbed or the banks. The processing equipment either has to be situated above the high-water level or designed for semimobility to allow its relocation at the onset of the rainy season.

Finally, the particle shape found in sedimentary deposits and even after size reduction is frequently unfavorable for hydromechanical gravity. Small tabular or flaky particles are difficult to separate by gravitative means, despite the high specific gravity of gold.

In simplified terms, the dressing processes used for obtaining gold concentrates comprise three separate phases, the first of which is raw-ore size reduction (to the extent necessary) for the purpose of liberating the valuable minerals, in most cases the gold. The second phase consists of classifying the crushed and/or liberated feed products into various particle-size fractions, and the last phase consists of actually separating the valuable material by means of various sorting processes. The following chart shows in simplified form the individual dressing processes and the devices which may or may not be used to achieve the desired results. The next chapters describe the individual processes and techniques of gold ore dressing.

First, however, let us take a look at some processing problems specific to small-scale mining and some suggestions on how to improve the results of processing in small-scale gold mining operations by better organizing the work involved.


RAW ORE
Primary ores, consolidated sediments
Slightly consolidated sediments
Loose sediments
Size reduction
Manual crushing
Jaw crusher
Stamp mill
Edge (Chilean) m.
Ball mill
Rod mill
Quimbalete/Maray
Disintegration
Washing sluice
Autogenic mill
Sizing trommel
Classification
Fixed screen
Shaking screen
Sizing trommel
Hydrocyclone
Hydroclassifier
Spitzkasten
Spiral classifier
Rake classifier
Grav. Sorting
Jig
Sluice
Pinched sluice
Pinched sluice with pneu. fluidized bed
Spiral concentrator
Tables (blanket, rocking, concussion, shaking)
Gold pan
Centrifugal concentrator
Hydraulic trap
Amalgamation
Amalgamating copper plates
Amalgamating barrels
Pocket amalgamators
Amalgam presses
Distillation retorts
Sluices
Centrifuges
Leaching
Agitation lea.
Vat leaching
Percolation lea.
Heap leaching
Smelt separation
Roasting and pneumatic separation
Selective comminution
Magnetic separation
END PRODUCT

The main problem complexes are, for one, the modest capacities of the processing plants and, for another, their frequently very low recovery of values. The throughput capacities are mostly limited by the very energy-intensive size-reduction process. In Nariño, Colombia, for example, the individual gold ore crushing capacities of roughly two tons per day are significantly lower than that of equipment in open-cast mining operations. Consequently, the size-reduction stage acts as a bottleneck that slows down the entire dressing process. The productivity of gold-processing equipment is highly dependent on the shape and size distribution of the gold particles within the feed. Generally, however, small-scale gold processing plants can be expected to recover about 50 % of the valuable minerals, at least as far as gravity recovery equipment is concerned. The recovery rate can be improved by including additional processing steps (retreatment of tailings), e.g., by cyanide leaching. The following section offers some suggestions on how to better organize the work in order to improve the processing capacities and results of processing, i.e., the recovery-of-values level.

Some simple changes in the processing sequence can, even without a nominal investment, improve the economic results of gold preparation via the following considerations:

- **Avoidance of idle time.** In many processing plants, particularly those in which a large number of different particle-size fractions are processed and/or tailings or middlings have to be resorted, the practice of intermediate piling prior to a second run costs substantial amounts of time. By targeting a continuous mode of operation, with the feed kept in the pulp as it runs through the entire system, from the size-reduction stage to final sorting, the equipment operator can minimize the idle time.

- **Avoidance of overmilling.** Mill operators must be aware of the danger of overmilling, particularly when ball or rod mills - in which high fine and ultrafine fractions are produced - are used for grinding gold ore. Such mills may be equipped with a hydraulic discharge setup such that the mill product is carried off by a flow of water. Since the heavy gold resists being discharged longer than the other material, it is inevitably ground down to a selectively finer particle size than are the country rock particles. Such an effect is undesirable, leading as it does in the downstream processes to very high fine-particle losses, particularly through gravity separation. The problem can be countered at an early stage by equipping the mill with a suitable discharger. Parallel processing analysis provides advance information on the particle size spectrum of the disintegrated material as a basis for determining the milling time and, hence, the size of feed for the sorting equipment.

- **Processing of narrow grain-size spectra.** Some dressing processes, particularly those based on gravity transport, have a classifying effect. With regard to optimal separation efficiency, the feed material should be separated into narrow particle-size fractions On the other hand, dressing processes such as amalgamation demand separation of oversize, which necessitates preclassification of the feed material. A table in the appendix compares the optimal operative ranges of gold processing equipment with regard to size of feed.

- **Increasing throughput by use of parallel equipment.** The throughput rate of ore processing plants always depends on that of the slowest individual step. In primary gold preparation, the slowest step normally is the size reduction of raw ore. By contrast, what tends to limit the throughput of processing equipment in placer gold mining is the sorting stage. By suitably arranging several pieces of low-throughput equipment in parallel, the throughput of the overall plant can be substantially increased.

- **Hydroclassification of feed ore.** Particularly for wet mechanical processing, the material sizing stage dictates the selectivity of the sorting process. Hydroclassification is much more suitable than sieving for the ore fed to sorting devices like tables and sluices. This is because hydroclassification works on the principle of equal falling rates, i.e., a single fraction contains both large, light particles and small, heavy particles. When that kind of feed is put through one of the above types of sorting devices, the incident flow impinging on the individual particles does a better job of separating the heavy material from the light material than any screen could achieve. The fact that hydroclassifiers have a continuous mode of operation is an additional advantage for nonmechanized small-scale mines.

Most conventional small-scale mining operations in developing countries still depend mainly on screen classifying (grading) with all of its drawbacks, e.g.:

- lower throughput rates,
- less selectivity,
- higher labor requirement and
- intermittent operation.

Hydroclassification eliminates such problems.

- **Production of preconcentrates.** Especially if the material in question has already been liberated, one of the first steps of processing should be to produce preconcentrates in order to reduce the through-put volume for the subsequent sorting steps. In placer gold mining, for example, a suitable form of coarse-particle classification can yield a strong concentration of gold in the fines fraction. This requires, of course, exact knowledge of the particlesize distribution of the valuable mineral (gold) in the feed ore. In some cases, the economic geology of primary ores, e.g., formal veins in unimpregnated country rock, allows hand sorting as a means of obtaining preconcentrates.

- **Retreatment of middlings.** All conventional processing techniques produce middlings. Such products belong to one of two basic categories:

- Middlings from sorting equipment with relatively low selectivity. While the individual components are liberated, i.e., the valuable mineral is present as free grain instead of being intergrown with country rock, gangue or accompanying particles, it has not yet been separated into gold-bearing and non-gold-bearing fractions. This type of middling occurs frequently in connection with mechanical gravitative sorting processes, particularly when the valuable mineral and the tailings have similar specific properties, e.g., specific gravities;

- Middlings that have not yet been liberated by prior size reduction, i.e., which still contain intergrowths of valuable mineral and valueless waste in one and the same particle. This type of middling naturally can also occur despite extreme selectivity and remains inseparable by further sorting without prior liberation by size reduction.

Both types of middling products may occur together, of course. In any case, it must be determined which type of product is involved, since it would be too expensive to put already liberated material through a size-reduction step (high cost of milling) and also would reduce the recovery rate (due to the poor recovery rates of fine-particle sorting devices).

Gold pans are a simple, inexpensive means of rapid analysis for quickly determining the type of middling one is dealing with and the necessary nature of further treatment:

- liberated middlings should be resorted, and
- unliberated middlings should be recomminuted and then re-sorted.

The gold processors in the Nariño District often exploit the effects of natural weathering of piled middlings as a means of precomminution. The oxidized, sulfide-rich material first breaks down at the mineral's intergranular voids, yielding better liberation of the gold in subsequent stamping.

In any case, such gold-bearing middlings constitute valuable material that will involve no further extraction costs, but only minor outlays for size reduction and sorting. Only if the middling is very fine, and the mine's own processing equipment will not suffice for resorting should the miner even consider selling the material to a mechanized outfit with appropriate processing equipment, leaching plant, or the like.

- **Exploitation of byproducts.** A further strategy for improving the economic efficiency of gold mines lies in the marketing of byproducts. For primary deposits, the genesis and, hence, the paragenetic mineral composition of the deposit dictates the exploitable byproduct potential. Such byproducts include:

- stibnite	- scheelite
- chalcopyrite	- bismuth minerals
- uranium minerals	- free silver
- galena	- silver minerals
- sphalerite	

and many more.

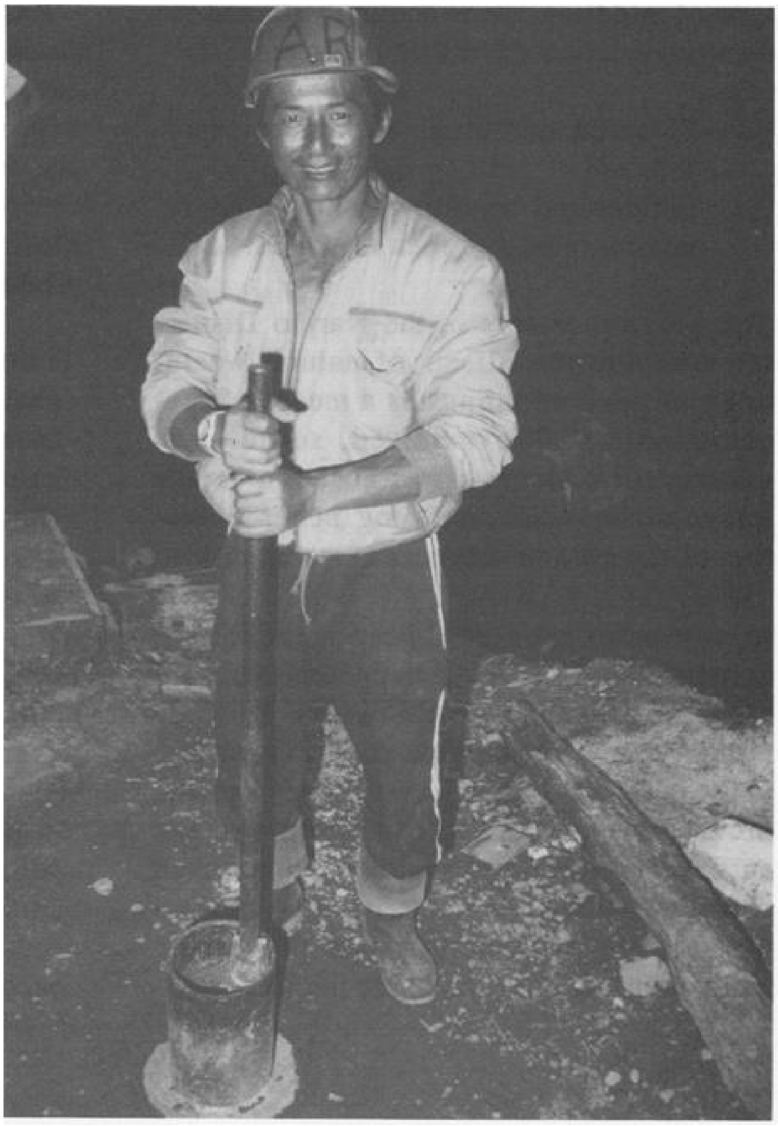
Fluviatile heavy-mineral placers, including many major gold deposits, are the result of physical deposition of heavy, weather-resistant minerals, meaning that the gold is in the paragenetic accompaniment of several additional minerals. Sometimes, the accompanying minerals can be exploited as byproducts for separate marketing (the table in chapter 2.5 lists in bold face the mineral designations of such products).

3.1 Size Reduction

The size reduction of intergrown ores serves a double purposes: to liberate the valuable material (gold) by breaking down the grain structure and separating it from the worthless waste, and to obtain feed material of the proper size for the intended sorting process. Both goals are achieved by pounding, shearing and milling the feed material.

Accordingly, size reduction is only relevant for primary gold ore, consolidated and, to some extent, slightly consolidated sedimentary ore. Loose material from placers requires no size reduction, because the gold is already liberated. The simplest form of size reduction is to crush the material by hand with a hammer or in a manual stamp mill (or mortar, cf. photo 1).

Photo 1:
Manual stamp mill (porrón) used for milling high-grade gold ore in Guaysimi, Ecuador



The specific throughput rates are extraordinarily low: less than 0.1 t/h for rough crushing with a hammer and approximately 10 kg/h for fine-grinding hard ore in a manual stamp mill. Consequently, the economic efficiency of such processes limits their application to very high-grade ore.

This illuminates the general problem of size reduction. Very hard country rock, gangue minerals and accompanying minerals require high comminuting energy inputs. With the conventional means of small-scale mining processes, i.e., manual labor, this means hard work for minimal results. The specific energy requirement for comminuting raw ore is heavily dependent on the type of ore being processed and may in some cases exceed 50 kWh/t. Reef ore, for example, with its large proportion of tough quartz can usually be expected to require such high specific energy inputs for size reduction. The performance of size-reduction equipment can be roughly estimated on the basis of its power rating - presuming the availability of reference data, either from other gold processing plants dealing with the same type of ore or from laboratory testing to determine the specific energy input requirement for size reduction.

The various means of size reduction most commonly employed in small-scale gold mining are discussed below.

3.1.1 Jaw crushers

Jaw crushers are used for initial size reduction, i.e., primary crushing. They crush the feed in a conical space between two inclined jaws, one of them stationary and the other reciprocating around its own axis. The reciprocating jaw is powered by an eccentric toggle-lever system which effectively enlarges and reduces the size of the crushing space. As the jaw moves in, it breaks the material into smaller pieces, which then slide down into the gradually narrowing space until they eventually reach the intended particle size and fall through the gap at the bottom of the crusher. The jaws are subject to considerable wear. Especially in the case of tough, quartzose gold ore, the outlays for spare parts can be quite substantial. A typical set of crushing jaws may be worn out after crushing about 30 000 t of feed. Hard-facing can significantly enhance the longevity of locally manufactured crushing jaws. With their comparatively favorable operating speeds, jaw crushers can be driven by various types of motors, i.e., electric, diesel and gasoline engines, even turbines. The energy requirement for normal rough crushing ranges between 3 and 10 kW for through-puts of 1 - 2 t/h. The size of the feed material is reduced at a ratio of between 5 : 1 and 10 : 1. The crushers normally used in small-scale mining can handle lumps of material measuring up to 20 cm across, and the minimum product size is about 5 - 10 mm.

If a jaw crusher is to be included in the dressing plant, some means of prescreening the feed should also be included to reduce the amount of fine material and, hence, the total amount of material that will have to be put through the crusher, thus reducing the overall energy consumption.

By reason of their simplicity, jaw crushers are suitable for local manufacture. Circumstances permitting, the customary cast construction may be dispensed with in favor of welded or bolted plates. Arc-welding equipment and suitable build-up electrodes should be on hand for the hard-facing work.

Photo 2: **Jaw crusher** with electric motor and V-belt drive for crushing antimony-gold ore, Mina Luchusa, La Paz, Bolivia.

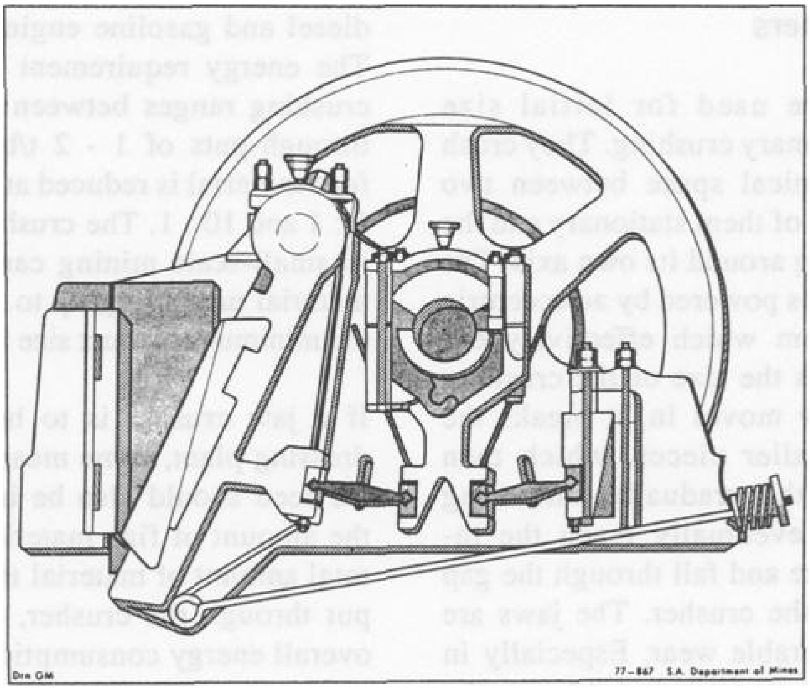


Figure 1: Conceptual drawing of a **jaw crusher** [Armstrong]

3.1.2 Stamp mills

Stamp mills are widely used in small-scale gold mining for wet and dry comminution of coarse, medium and fine feed. Frequently, mercury is added to effect amalgamation at the same time. The crushing effect derives from a piston, which is pushed up by a cam on the drive shaft and then falls down into the stamping trough. On the feeder side, the trough has a raised wall, over which the material is shoveled in. On the discharge side, there are slots, screens or weirs, and the bottom of the trough has a solid floor or a sufficiently thick layer of coarse material to support the feed to be stamped. In a wet stamp mill - which offers the multiple advantage of low dust levels, low wear and less complicated removal of the product - the stamping takes place under water, with the product being carried off by running water. Most multiple-piston stam mills have four to six wooden or metal pistons, which should be of two-piece design, i.e., with replaceable metal shoes rigidly attached to the shafts in order to avoid loss of performance due to slippage. The stamping frequency is very important for good results: stamping too slowly leads to fine-grain sedimentation and excessively fine milling, and the throughput rate remains very low. Optimal stamping frequencies range from about 30 min^{-1} for fine milling to about 90 min^{-1} for coarse milling. Flexible screens, e.g., self-cleaning (non-plugging) perforated rubber sheets, can be installed on the discharge side. It is very important in gold mining to keep the stamp mill pulp free of oil and grease. Care must be taken to ensure that no lubricant, e.g., from the piston bearings, is able to drip into the mortar box, because the immediate result would be fine-gold flotation in the downstream sorting plant.

(To avoid flotation resulting from dissolved lubricants, Colombian miners toss pieces of sisal leaves into the pulp. The leaves contain surface-active substances that saponify grease and oil.)

If the stamp mill is to serve simultaneously as an amalgamator, interchangeable, mercury- or silver-amalgam coated copper plates can be installed on the feed side of the stamping trough. Particularly in the amalgamation of ore containing sulfides and/or soluble ore minerals of the metals copper, bismuth, antimony or arsenic, and in the presence of grease or oil in the pulp, the discharged material may contain substantial amounts of floured mercury that is very difficult to recover. The inclusion of downstream amalgamating copper plates and hydraulic traps is an absolute necessity. The mercury losses can be reduced by adding tensides and lime.

Stamp mills can handle between 0.8 and 2.5 t per day (24 hours) and piston. Each piston weighs between 150 kg and 350 kg and requires a driving power of at least 0.5 kW. Some stamp mills have power requirements exceeding 50 kW. The slow turning speed of the main axle allows the use of waterwheels to drive stamp mills. Such mills can, of course, also be powered by internal combustion engines, electric motors or turbines (via transmission gearing). The size of feed for stamp mills should not exceed 100 mm. From case to case, then, some form of precomminution, e.g., pounding with a hammer, will be necessary. The product of crushing in a stamp mill stays within 50 % < 100 µm.

Stamp mills are very suitable for local manufacture. Steel stamp mills naturally require a good metalworking shop. Railway rails are sometimes used for the piston shafts. In many cases, carpenters build stamp mills in situ. In Colombia, gold miners use wooden stamp mills made of local hardwood driven by large, overshot wooden waterwheels, with three or four cams per piston placed directly on the center shaft of the waterwheel. The typical Colombian stamp mill has three or five pistons and takes about three weeks to build (incl. waterwheel). The cast-iron stamps are replaced every six months or so with new ones from a local foundry. Metal-on-wood friction bearings are preferred.

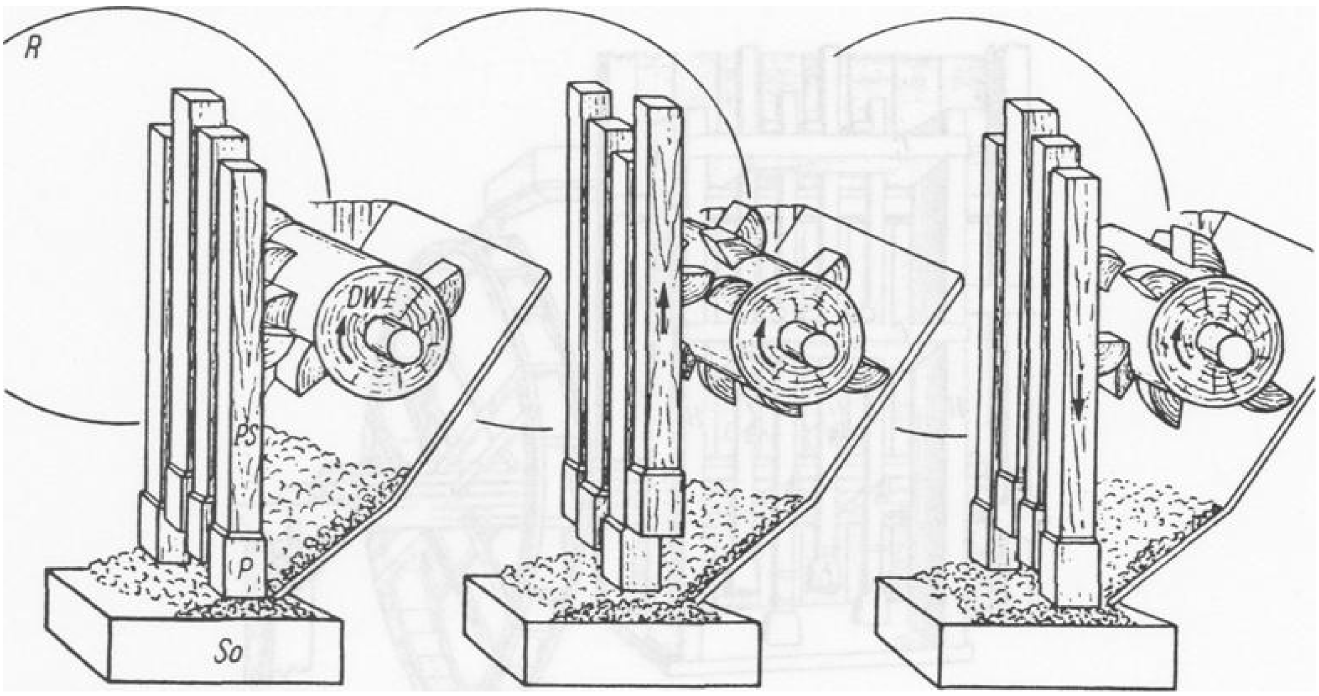


Figure 2a: Functional diagram of a stamp mill [Wagenbreth]

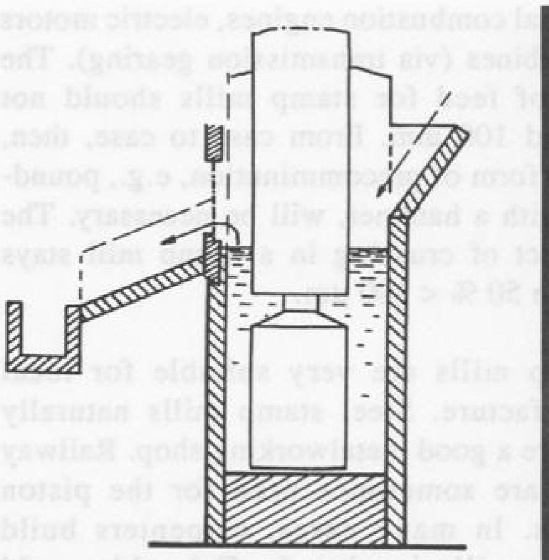


Figure 2b: Schematic diagram of a discharger for a wet stamp mill with a screen bundle

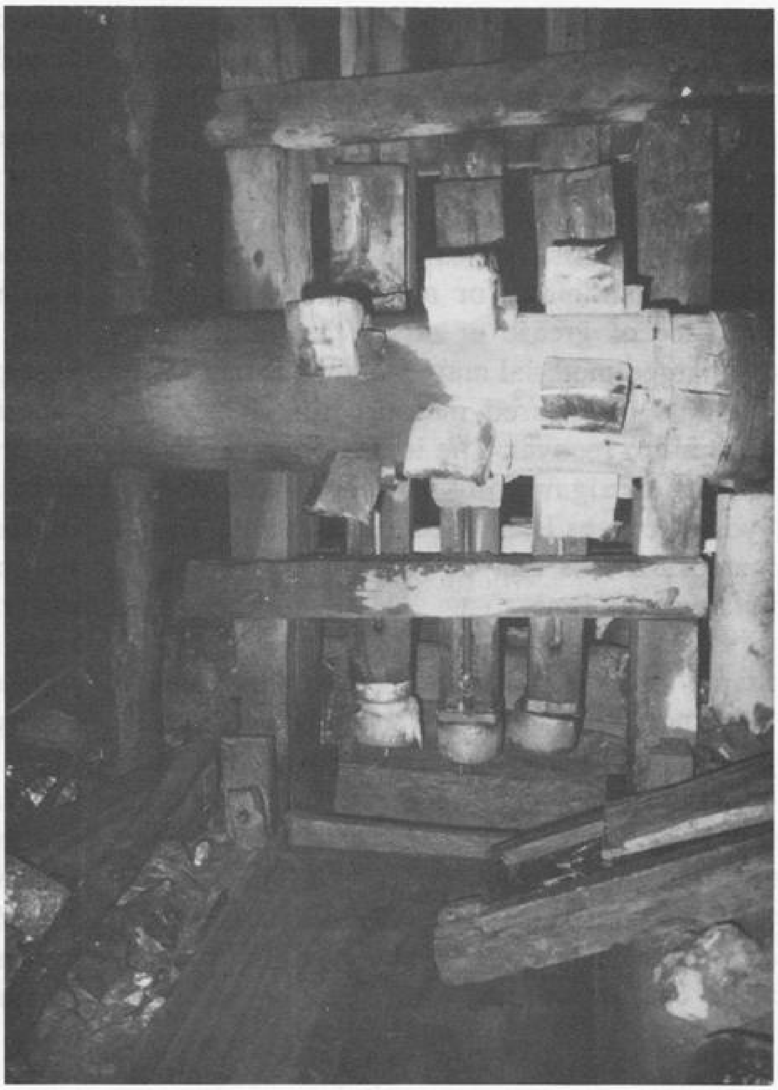


Photo 3:
Three-piston wooden stamp mill with open mortar box. Iron-shoed pistons behind the waterwheel shaft with cams for the pistons. In the foreground, a blanket table with blanket removed. Mina La Llanada, Nariño, Colombia

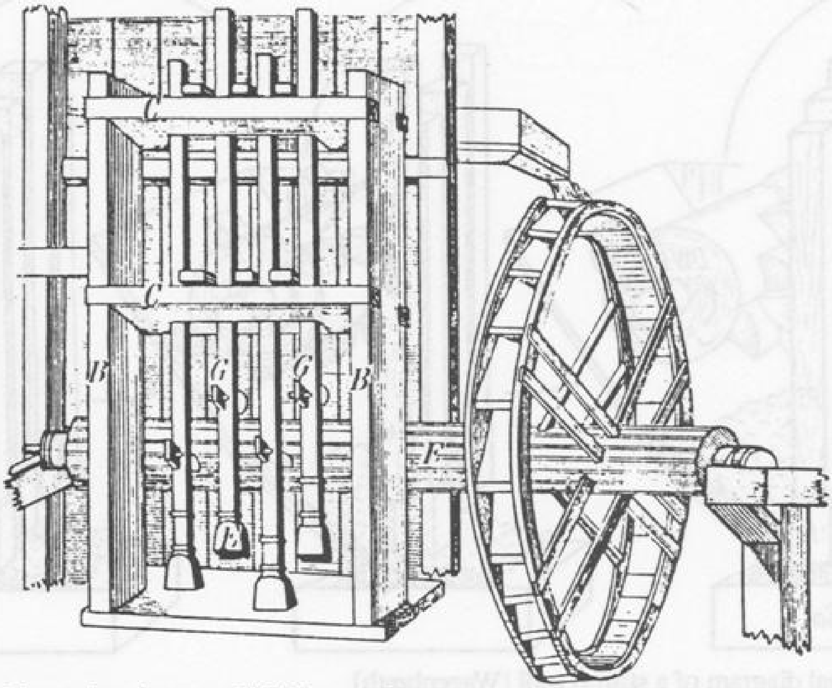


Figure 3: Waterwheel-driven **dry stamp mill** [Schennen]

3.1.3 Edge (Chilean) mills

Edge mills, also referred to as Chilean mills, serve in the medium and fine comminution of primary gold ore, frequently in combination with amalgamation in the grinding track. The principle of operation is that a pair of steel-jacketed concrete wheels (mullers), often weighing more than half a ton each, tumble around a circular grinding track, thereby grinding the product to a fine or very fine consistency. The size of the end product depends either on its retention time or on the velocity of the water flowing through the mill. In many parts of the world, stamp mills, with their comparatively low capacities, have been replaced by a combination of crusher and edge mill. The main advantage of an edge mill as an ore dressing device is that it does a very uniform job of milling and amalgamating gold ore, pressing the fine gold particles into the mercury and ensuring their amalgamation. Normally, they would escape amalgamation due to the high surface tension of mercury. At the same time, the surface of the liberated gold is at its cleanest during milling, i.e., before it has time to corrode. In northern Chile, many miners line the conical sides of the grinding track with copper plates to enhance the amalgamating effect by bonding finely dispersed (floured) mercury. As do stamp mills, edge mills involve relatively high mercury losses for amalgamation. The finely dispersed mercury and amalgam are washed out of the mill and have to be recovered by gravimetric secondary sorting, e.g., in a hydraulic trap or on amalgamating copper plates. Some edge mills have throughputs in excess of 1 t/h. Such capacities necessitate 5 - 7 kW installed power. The extremely heavy wheels make starting difficult and may even require the use of a more powerful starting motor. While most edge mills are driven by electric motors, some are equipped with hydromechanical drives, e.g., water-wheels. The relatively complex design of edge mills, with their mitre gears, complicated main axle bearing and hardened steel grinding faces on the mullers, precludes local manufacture. Edge mills can only be built in specialized metalworking shops with adequate experience in the processing of hardened steel. The muller jackets have to be filled with concrete for final installation. Alternatively, individual segments of the jackets can be positively but interchangeably connected with the inside of the mullers.

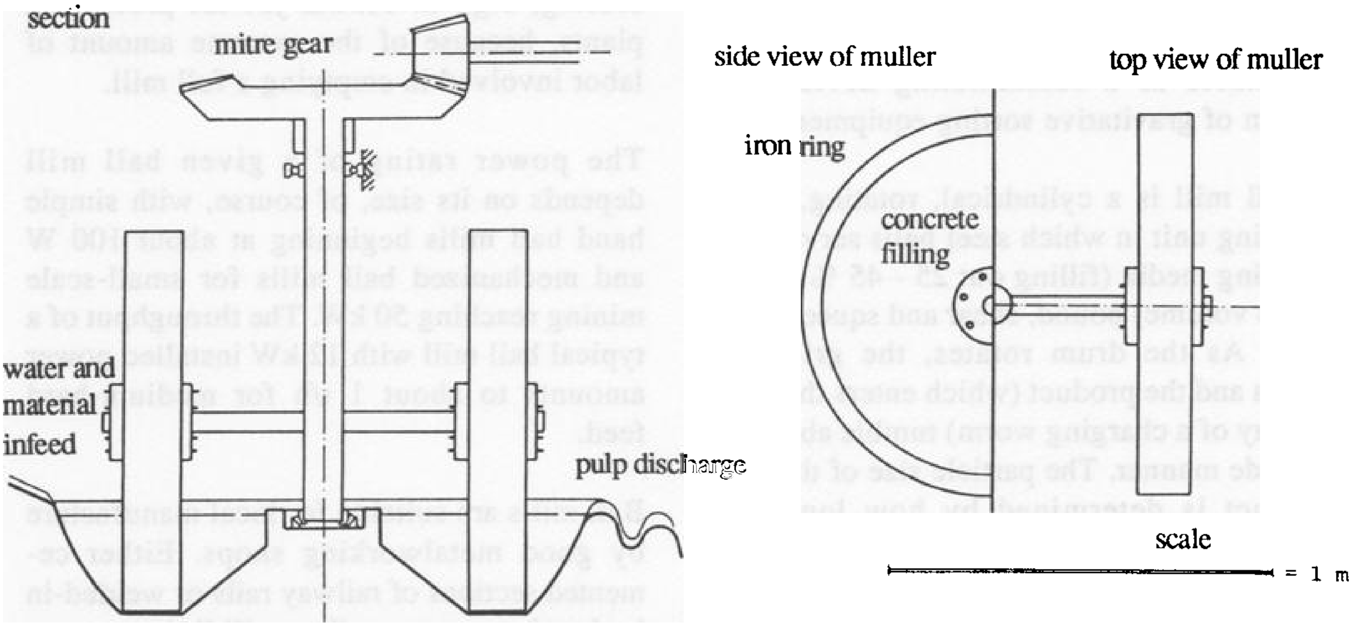


Figure 4: Constructional sketch of a Bolivian **edge mill** [Priester]

Photo 4: General view of an **edge mill** powered by an electric motor with belt transmission and mitre gear; Copiapó, III Region, Chile

3.1.4 Ball mills

Ball mills are used for fine grinding gold ore. Their main area of application is to effect size reduction of feed material for agitation leaching. The large share of fine-grained product makes most ball mills unsuitable as a comminuting device upstream of gravitative sorting equipment.

A ball mill is a cylindrical, rotating, fine-grinding unit in which steel balls serving as grinding media (filling out 25 - 45 % of the mill's volume) pound, shear and squeeze the feed. As the drum rotates, the grinding media and the product (which enters the mill by way of a charging worm) tumble about in cascade manner. The particle size of the end product is determined by how long the material is milled. Flowing water carries the fine fraction off over wires or through screens, leaving behind any material that is heavy enough to withstand the flushing effect. The mill case has an anti-wear lining consisting either of hardmetal with catches for the product and milling balls or of rubber (mainly for regrinding middlings). Ball mills are very unsuitable for batch processing, e.g., in central job-lot processing plants, because of the extreme amount of labor involved in emptying a full mill.

The power rating of a given ball mill depends on its size, of course, with simple hand ball mills beginning at about 100 W and mechanized ball mills for small-scale mining reaching 50 kW. The throughput of a typical ball mill with 12 kW installed power amounts to about 1 t/h for medium-hard feed.

Ball mills are suitable for local manufacture by good metalworking shops. Either cemented sections of railway rails or welded-in leaf springs serve well as mill linings.

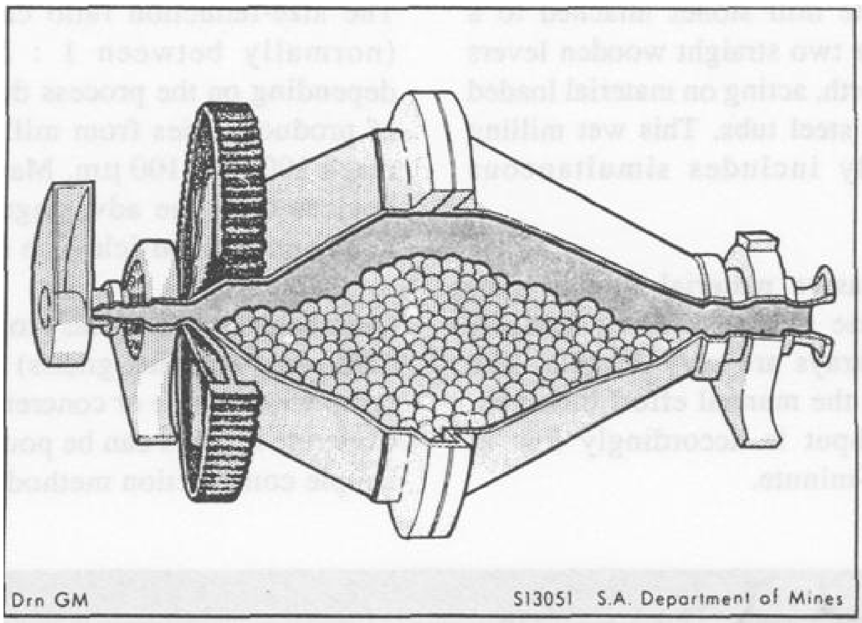


Figure 5: Schematic diagram of a ball mill [Armstrong]

3.1.5 Rod mills

Like ball mills, rod mills are used for fine grinding of gold ore. Since rod mills produce less microfine material, they are more likely than ball mills to be chosen as size reducers in preparation for gravitative concentration. Rod mills are similar to ball mills in design and construction, except that the grinding media consist of steel bars instead of balls.

Apart from round mill housings that rotate with the charge, there are various other design alternatives for rod mills. For example, a mill using steel bars with a diameter of 5 - 6 cm in a vibrating, funnel-shaped trough could be used for special job-order milling and in batch processing plants. The feed spreads out uniformly around the inner surface of the trough and migrates toward the bottom by reason of its own weight and the natural sorting process, thus becoming increasingly comminuted. The funnel-shaped trough also keeps the milling bars separated according to diameter, with those showing the greatest amount of wear gathering at the bottom to do the finest grinding. The product trickles out through a slot at the bottom of the trough. If no new material is added, the mill eventually runs empty by the effect of gravity. This type of construction allows the use of standard hardmetal plates as a mill liner. Such mills are driven by powerful unbalance motors.

3.1.6 Quimbalete, Maray

Manually operated comminuting equipment like the Bolivian Quimbalete or the Chilean Maray are in widespread use among small-scale gold miners, who use them for grinding coarse to medium-size material.

Quimbalete: Very heavy rocker-crushers with stone or steel heads are rolled over the product on a stone slab or metal plate by means of a lever. The Quimbalete's position on the milling slab/plate is altered by rotating it slightly as it reaches the reversing position.

Maray: Concrete mill stones attached to a forked branch or two straight wooden levers rock back and forth, acting on material loaded into concrete or steel tubs. This wet milling process usually includes simultaneous amalgamation.

Even hard, abrasive material can be processed with these manual milling devices. Quimbaletes/Marays are very durable. The only problem is the manual effort involved, and the throughput is accordingly low at about 1 kg/man-minute.

The size-reduction ratio can be very high (normally between 1 : 5 and 1 : 20), depending on the process duration. The size of product varies from mill to mill but can reach 100 % < 100 µm. Manual pulverizing devices offer the advantage of achieving a homogeneous particle-size distribution.

Quimbaletes are very easy to make out of, say, hard stone (granite, gneiss) or welded metal scrap with a stone or concrete stuffing. Concrete Marays can be poured in situ using simple construction methods.

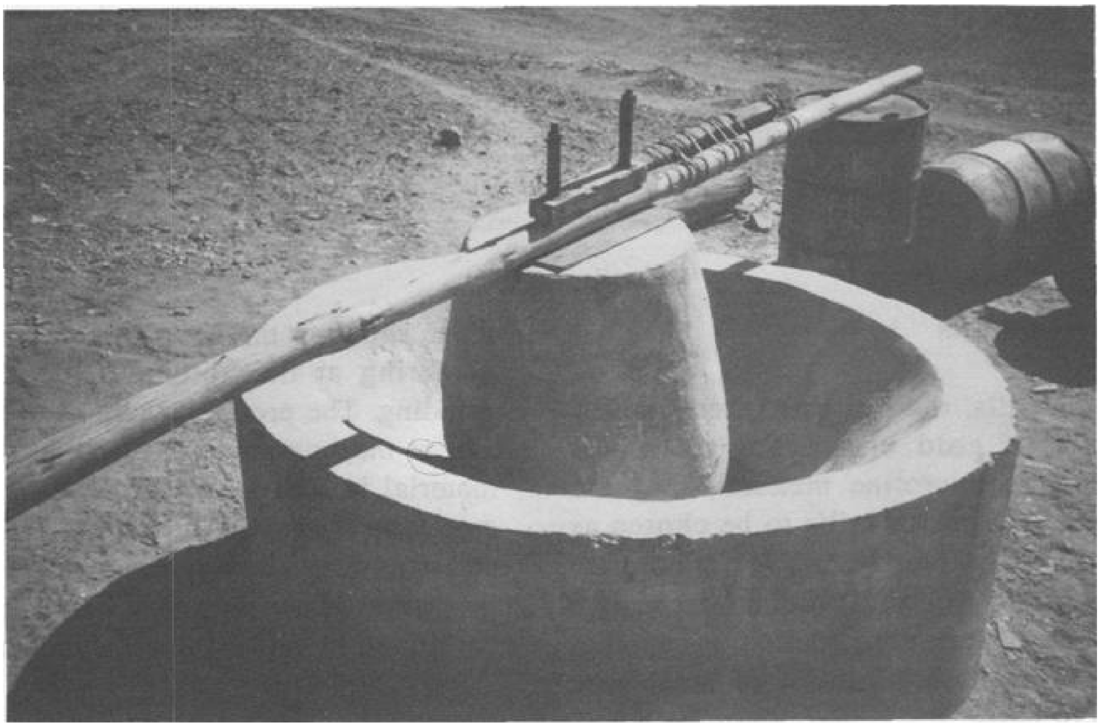


Photo 5: **Maray** (rocker-crusher) for simultaneous size reduction and amalgamation of gold ore in a concrete grinding bowl with a concrete pestle attached to a wooden lever, near Copiapó, III Region, Chile

3.1.7 Washing sluices

Washing sluices are used for separating and/or comminuting partially consolidated gold ore, in particular conglomerates, fossil river placers, etc. Washing sluices effect separation of the feed material with the aid of running water and artificial agitation. For clay-rich feed, the material is first mixed with water in a tank or pit and thoroughly kneaded by trampling or plowing through with a rake in order to break down the adhesive and cohesive cementing power of the clay fraction. Together with subsequent washing in the sluice, this often constitutes the only alternative for mining operations with little mechanization to process argillaceous gold ore. Most washing sluices are simple dug channels, usually with a number of small weir-like terraces. Separation of the partially consolidated material can be improved by shoveling.

Photo 6:
Washing sluice for size reduction and preconcentration of material from the "Alter Mann" gob. The miners repeatedly shovel the oncoming pulp back up behind the next-higher weir in order to improve the separating effect by washing away the less-heavy material. Leadsilver mine in Potosi, Bolivia.

3.1.8 Autogenic milling

Autogenic comminution is employed for some slightly consolidated gold ore, the object being not to reduce the size of the individual particles, but to break down the caked material in order to liberate the particles of gold. Particularly in the case of caked conglomerates, the coarse sediment can serve as grinding media for autogenic comminution. The simplest of autogenic comminution takes place in sizing trommels. If liberation requires a higher level of size-reducing energy, the use of a separate autogenic mill, in which the coarse particles take the place of the grinding media, may be appropriate.

3.2 Classifying Equipment

The process of classification comprises the separation of different particle-size fractions. The need to separate the feed material results from the fact that gravitative sorting processes always involve a sizing effect, i.e., the material is separated not only on the basis of density, but also of particle size. Thus, in order to avoid that effect, the material must be classified in advance. That way, a narrow grain-size spectrum can be fed onto the sorting equipment, which then separates it extensively on the basis of density variation. In addition, certain sorting processes require that the feed material be of a certain particle size, thus entailing preclassification (cf. table 2 in the appendix). In placer gold mining, classification can bring about both the separation of coarse material and preconcentration of the fine-grain fraction.

3.2.1 Fixed screens

Fixed screens are good sizing implements for coarse-to-medium particle sizes (minimal separating boundary: approx. 100 -

200 µm). Fixed screens can be used for dry or wet classification. They are always used when only small amounts of material need to be classified.

Wet classification is accomplished with the aid of a ladder-type screen system, i.e., an open channel with a series of consecutively finer built-in screens. The process water propels the undersize onto the next-finer screen, and a spatula is used to spread the material over the screens and remove the oversize.

Another basic form of rigid-screen classification is employed on the discharge side of stamp mills and various other comminuting devices, e.g., edge mills: the pulp splashes up against the screen, which may be made of flexible, and therefore self-cleaning, material such as perforated rubber sheet or nylon fabric.

Grizzly's, heavy steel-bar grates installed at an angle of 25-30^{0} and consisting of, say, railway rails, are used in gold placer mining as a means of coarse separation aimed at protecting the classifying screens from overly large or heavy feed.

Rudimentary wood- or metalworking shops suffice for local manufacture of screen frames. Most screen fabric is imported, since high-quality material that can withstand heavy abrasion is needed (e.g., special steel for processing sulfidic gold ore, the pulp of which is characterized by high acidity and corrosivity).

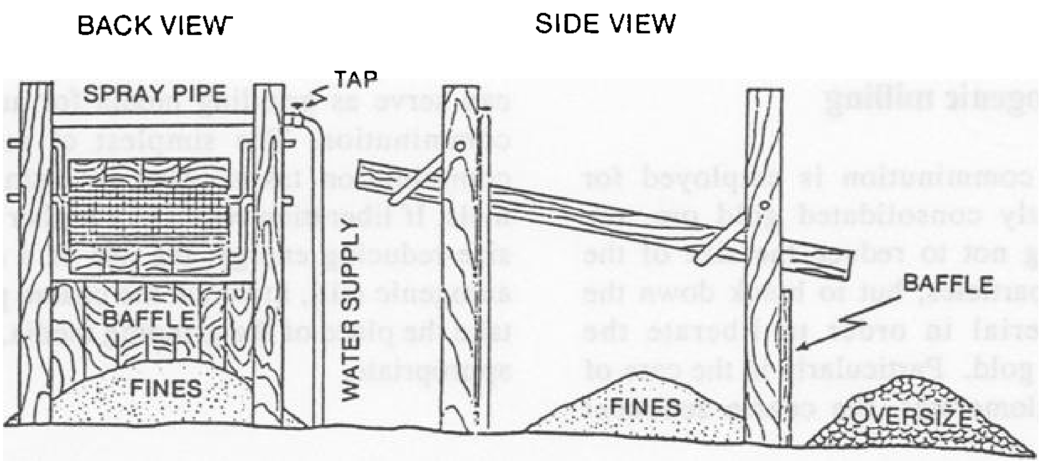


Figure 6: **Simple hand screen** with water sprayer [Stewart]

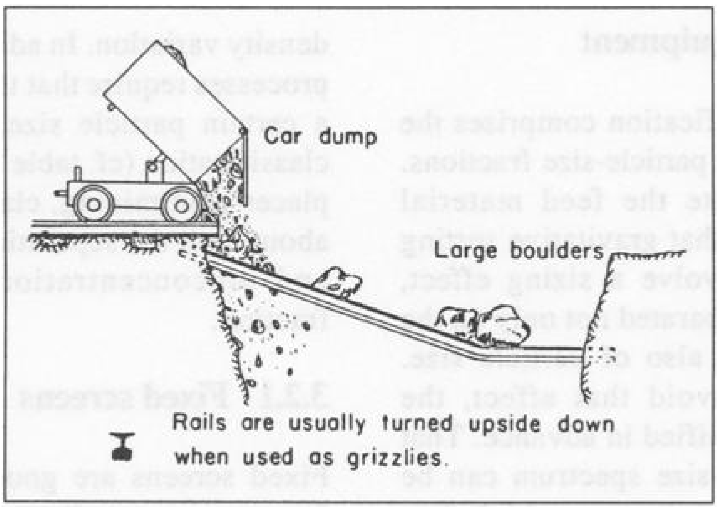


Figure 7: **Grizzly**, a heavy steel-bar grate for eliminating large boulders; made of railway rails [Stout]

3.2.2 Shaking screens

Shaking screens are used primarily in gold placer mining, frequently on dredges, mobile processing plants and, in exceptional cases, in the somewhat larger operations of primary gold mining. Shaking screens can handle high throughput rates. In addition to merely classifying the material, the shaking motion also helps break down consolidated, cemented and caked conglomerates. The feed material is spread with water jets. The size of feed should stay roughly within the 50 mm to 50 µm range.

In contrast to fixed screens, shaking screens are made to move in both horizontal directions, keeping the feed in motion and greatly facilitating its passage through the screen to effect separation in oversize and undersize material. The shaking motion can be generated by an unbalanced flywheel, unbalance motor or the like. Manual shaking screens are rarely encountered. While manual shaking screens are simply hung on ropes or cables, mechanized shaking screens have more sophisticated suspension systems, i.e., vehicle springs. The screen excursion amounts to anywhere from 25 to 80 mm. The device should be designed with a width-to-length ratio of 1 : 2.7. A special advantage of shaking screens is that they have a very modest space requirement.

Shaking screens are well-suited for manufacture by local machine builders.

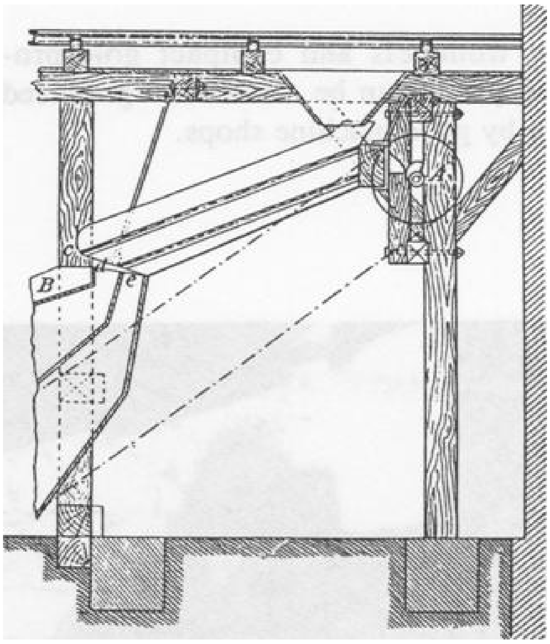


Figure 8: Two-stage **concussion screen** [Schennen]

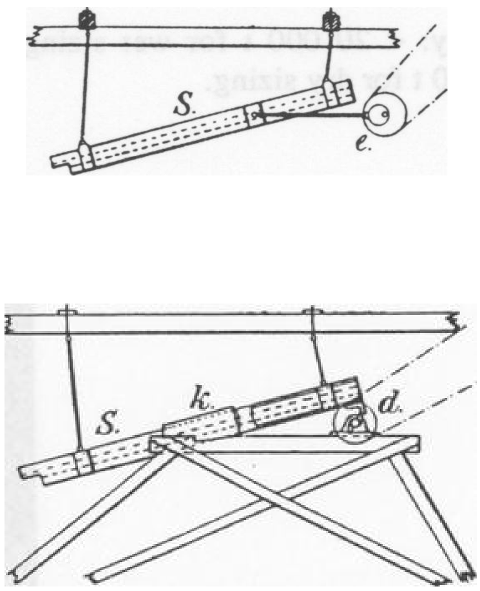


Figure 9: Conceptual diagram of a **shaking screen** (top) and a **concussion screen** (below) [Treptow]

3.2.3 Sizing trommels

Another typical classifying device used in gold placer mining is the sizing trommel. As in the case of a shaking screen, the motion caused by tossing the feed onto the screen helps break down agglomerated material (cf. autogenic comminution, ch. 3.1.8).
Sizing trommels are a mechanized form of wet screening, with a series of increasingly coarse screens or perforated sheeting forming a cylindrical drum - or trommel - on the discharge end. The trommel is powered by an external drive unit with belt transmission. The feed (< 50 mm) passes over the individual screens from fine to coarse, and the undersize drops through discharge cones onto different sorting devices.

Water jets situated 100 - 120 degrees from the bottom serve to keep the screens clean and accelerate the passage of feed.

Sizing trommels have very good running characteristics. Their throughput capacity is strongly dependent on the size of the screen perforations. Coarse screens have higher throughputs. Double- and triple-screen trommels use less electricity and water despite superior throughput rates. In addition, they are subject to less wear. There are two main types in use:

- sizing trommels with cylindrical screens and inclined axis and
- sizing trommels with conical screens and horizontal axis.

Angle of inclination: 4 - 5^{o}.

Longevity: < 20 000 t for wet sizing and < 100 000 t for dry sizing.

Sizing trommels do have certain drawbacks. For one, their energy consumption is substantial, and their self-cleaning effect is minimal. Inclusion of a spraying system (see above) can improve the latter.

If the material to be processed is too tough, sticky or argillaceous, the first trommel should have dogs instead of screens to promote autogenic size reduction of the feed.

The compact construction of sizing trommels predestines them for inclusion in small, mobile gold processing plants (e.g., Denver Goldsaver), which often consist of a drive unit, a pump and a sizing trommel that empties onto a vibrating riffle sluice that sorts the gold-bearing undersize.

Sizing trommels and compact gold-processing plants can be - and are - produced locally by good machine shops.

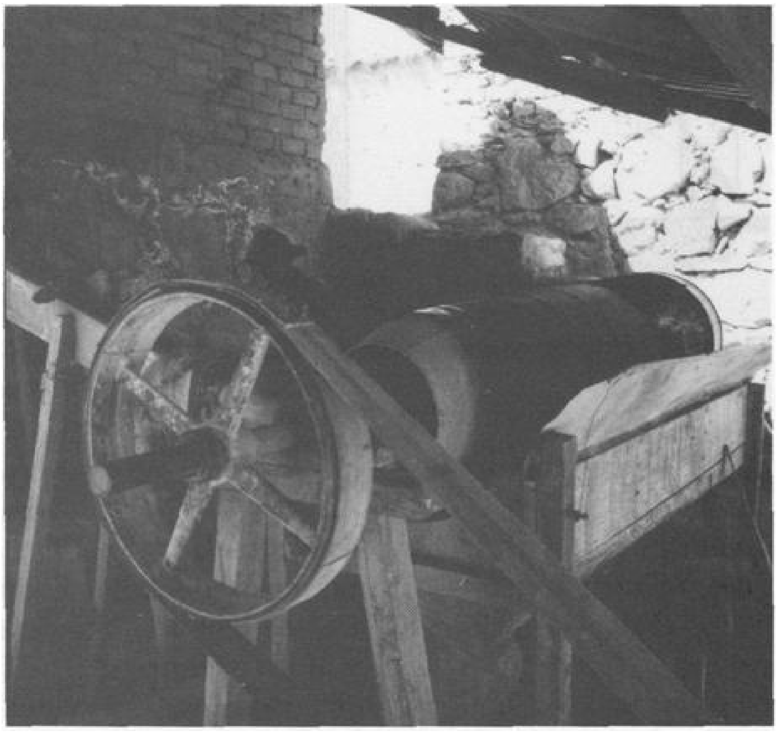


Photo 7:
Sizing trommel with belt drive powered by a small internal-combustion engine. Behind the flywheel: the chute used for feeding the material into the drum. Porco, Bolivia

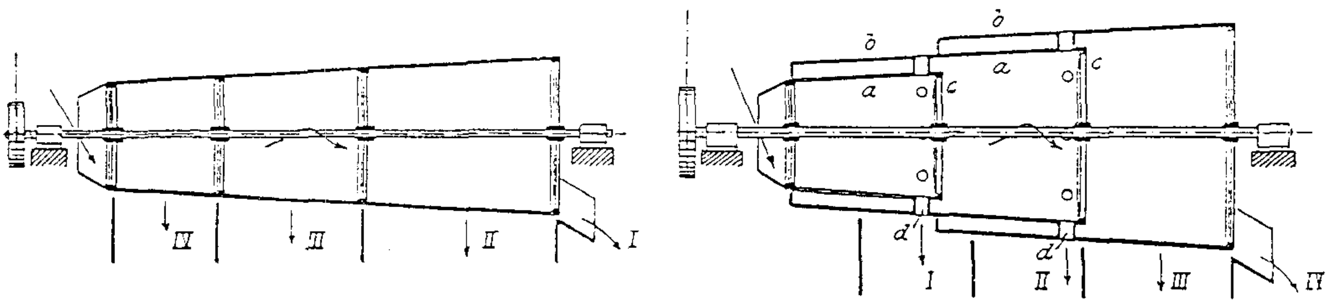


Figure 10: **Conical sizing trommel** with screen jacket shown at left and concentric screen configuration at right [Fischer]

3.2.4 Hydrocyclones

Hydrocyclones have a very wide range of application. They can be used for desliming feed material and pulp, e.g., in gold leaching plants, in which case they exploit the high density of the gold to separate it from the lighter material, with the gold and other heavy material collecting in the underflow as a preconcentrate.

The principle of hydrocyclone operation is somewhat complicated:

Hydrostatic pressure in the feed section accelerates the pulp as it enters the cyclone through a tangential inlet. The throttling effect in the lower, conical section separates the rotating slurry into an outer, descending portion and an inner, ascending portion, with the coarse, heavy material being carried outward and gathering in the underflow, while the lighter, finer material migrates inward for discharge through the overflow (cf. fig. 11).

Hydrocyloning upstream of sorting devices such as spiral concentrators or tables that tend to sort on the basis of grain size (impinged area) yields a much more selective form of separation than that achieved by screens.

Due to potential abrasion problems and the generally high demands on quality of manufacture, hydrocyclones are hardly suitable for local manufacture, despite their simple design. Most hydrocyclones used in Latin American countries are of industrial origin.

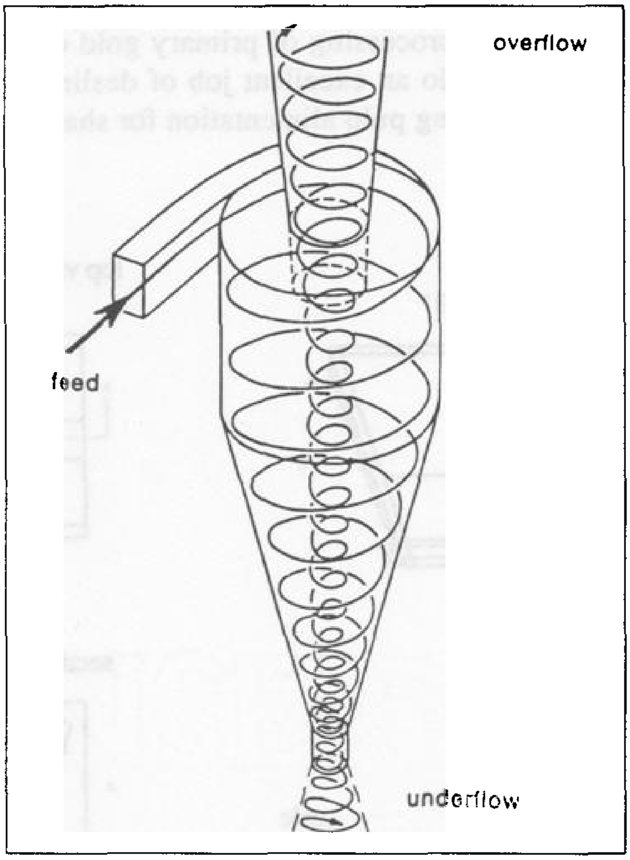


Figure 11: Conceptual diagram of the two-part flow situation in a **hydrocyclone** [AKW manufacturing bulletin]

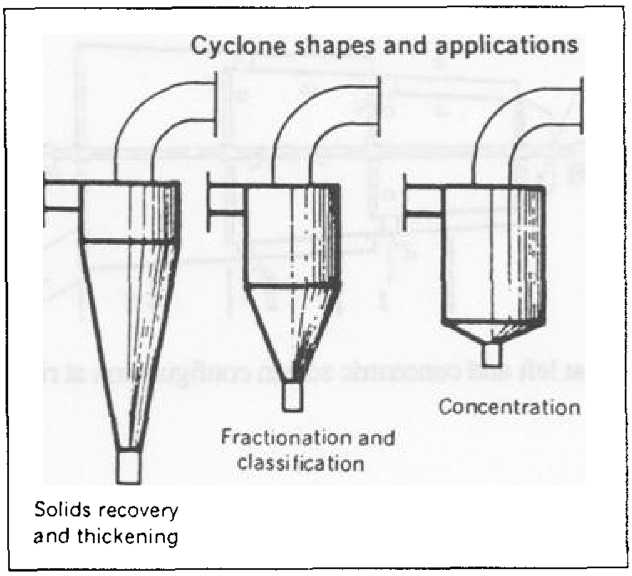


Figure 12: Three types of cyclone [AKW manufacturing bulletin]

3.2.5 Hydroclassifiers, spitzkastens (box classifiers)

Hydroclasssifiers of the countercurrent or box type and simple spitzkastens are sometimes used in the processing of primary gold ores, where they do an excellent job of desliming and classifying pulp alimentation for shaking tables.

A spitzkasten comprises several settling tanks in the form of inverse pyramids with drains installed at the bottom. The pulp flows through the series of tanks, remaining in each tank for a duration dictated by the tank volume, the throughput ratio, etc. Grain collectives settle out according to the principle of equal falling and are separated out while the rest of pulp flows into the next tank, where the next-finer fraction is removed in an analogous manner.

In a countercurrent classifier, separation is effected in three or more fractions plus slime overflow in a sizing chamber with a number of more or less high partitions and, for each fraction, a screen with an underwater feed arrangement that effectively forms a fluidized bed. The products of classification are drawn off through central discharge pipes equipped with hand-actuated regulating cone valves. In the direction of pulp flow, the products become increasingly fine, i.e., they display increasingly slow fall rates. The process is controlled by way of the underwater feed and/or valve adjustment.

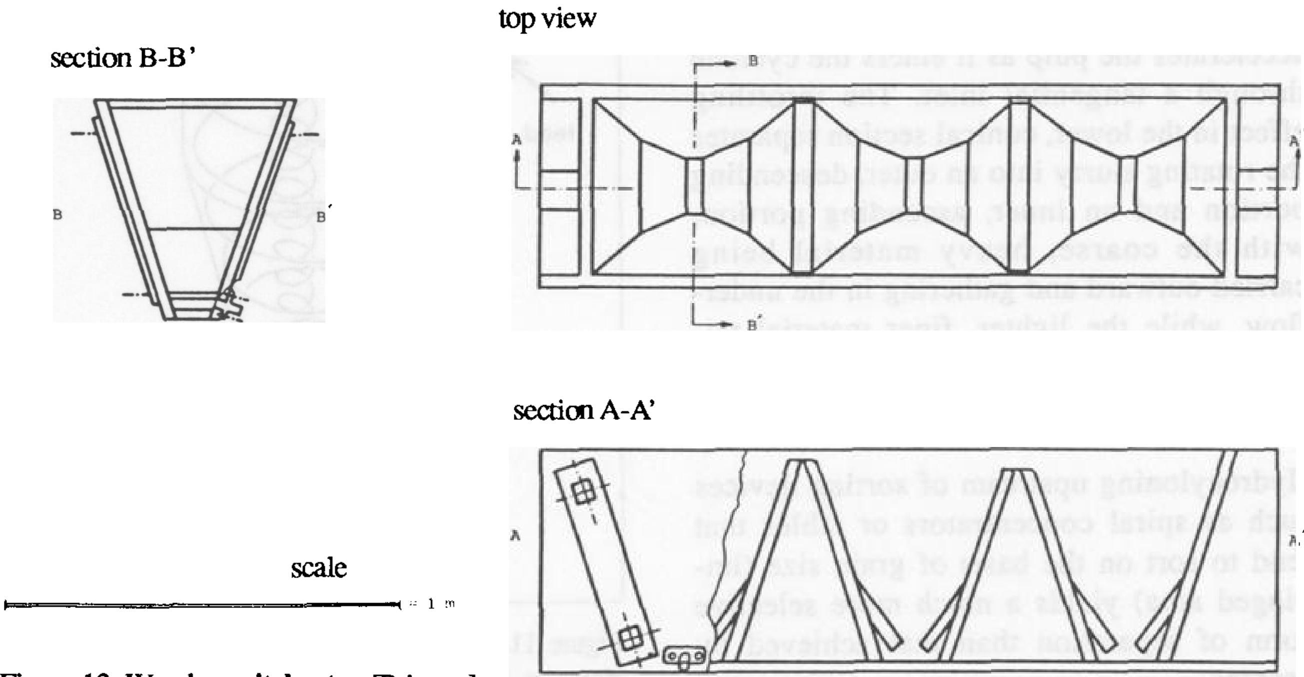


Figure 13: Wooden spitzkasten [Priester]

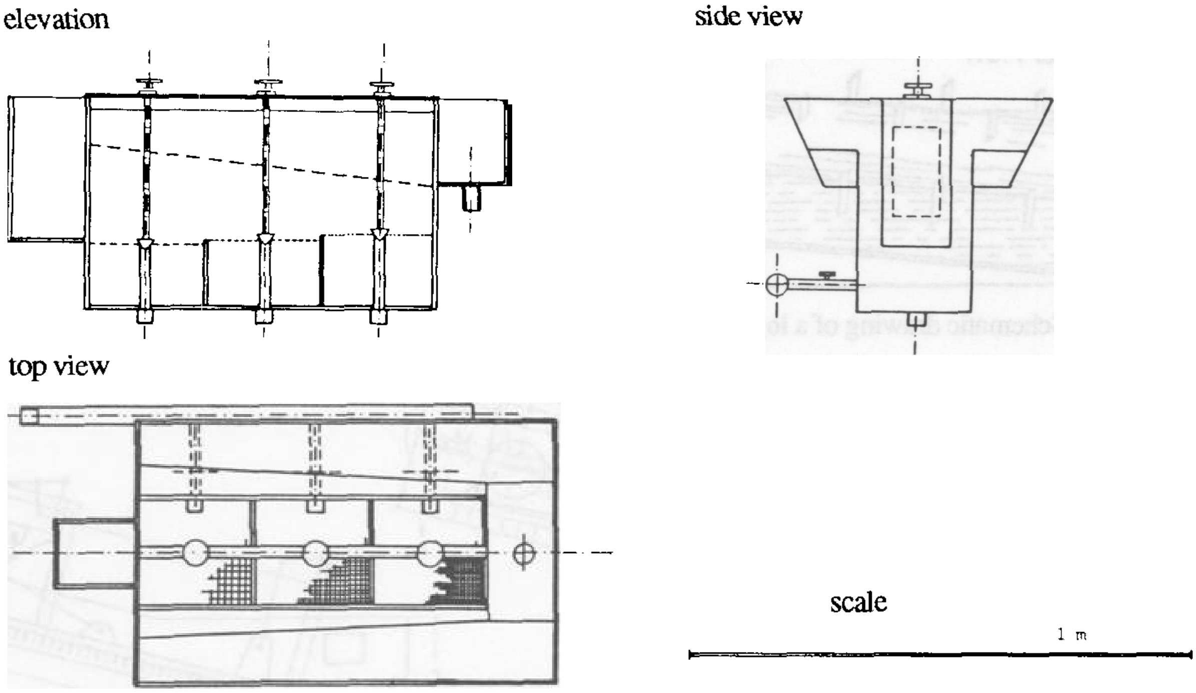


Figure 14: Constructional drawing of a **hydroclassifier** of the type Taller Metal Mecanico, Potosi, Bolivia [Priester]

The size of feed for such devices is below 1 mm. Thanks to their simple design with no moving parts, spitzkastens and hydro-classifiers are very well-suited for local manufacture, and can even be made of wood.

3.2.6 Spiral classifiers, rake classifiers

Spiral and rake classifiers are used by larger primary-ore processing plants for closed-cycle milling. To avoid overmilling the gold, the mill product should be put through a sorting device, e.g., amalgamating copper plates or a hydraulic trap, before the pulp enters the classifier for remilling of the coarse fraction.

Spiral classifiers have an inclined trough containing a rotating upward spiral that separates the pulp into ascending coarse material and descending fine slurry.
Log washers are of even simpler design. They work much like a spiral classifier in that the blades on the rotating shaft (~log) separate the pulp entering the bottom third of the trough into an ascending flow of coarse, heavy material and an underflow of light material. During the starting phase, the rectangular classifying box gradually fills up with material before the actual classi-fying process begins.
As the name implies, **rake classifiers** depend on a rake to draw the coarse material upward through an inclined trough. As it reaches to top, the rake lifts, returns to the starting point, and then drops onto the floor of the trough. The fines fraction remains in suspension, leaving the trough through an overflow (weir) at the bottom end. The pulp is fed in at a point located about a third of the way from the bottom end of the trough.

A high fines fraction (0.1 - 0.5 mm) is needed for maintaining a stable suspension. Otherwise, rake classifiers would achieve little selectivity.

The aforementioned types of classifier are also suitable for local manufacture, par-ticularly the quite simply constructed log washer, which can be built at any good carpenter's shop. The other devices require experienced metal workers.

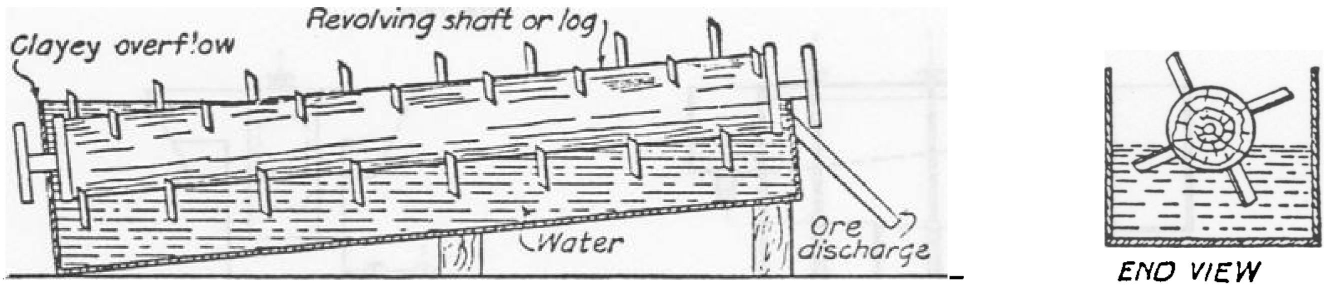


Figure 15: Schematic drawing of a log washer [Bernewitz]

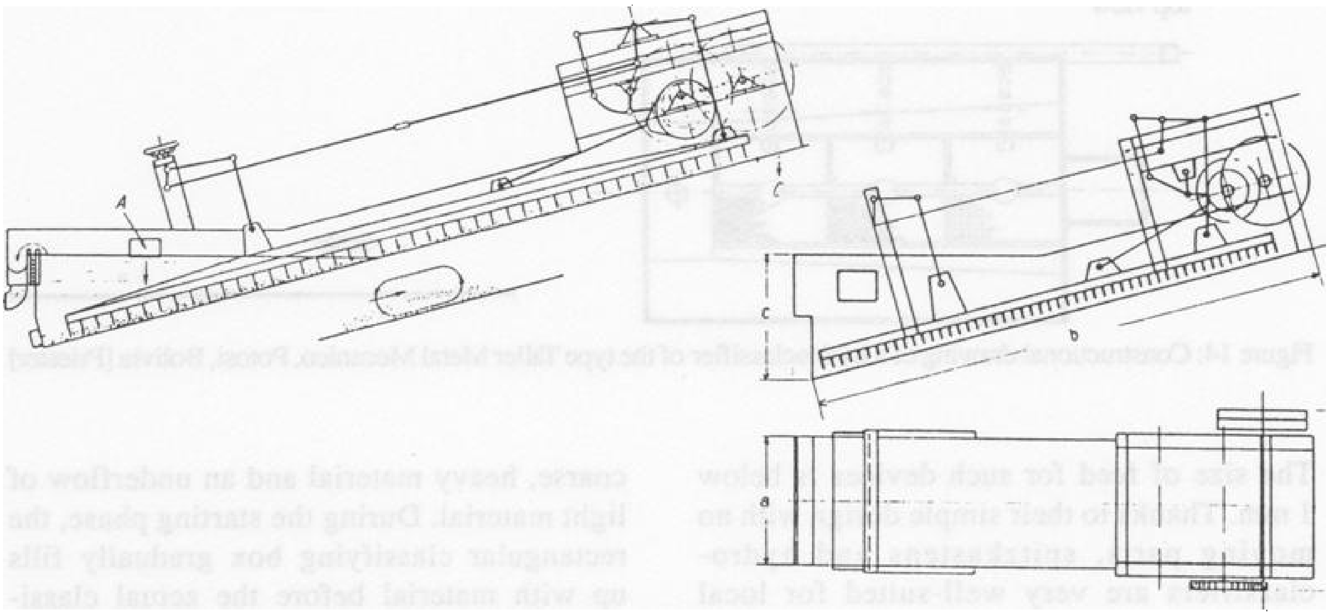


Figure 15: Simplified views of a rake classifier [Schmidchen]

3.3 Wet-mechanical Sorting

Raw ores containing valuable heavy minerals are processed by dry or wet mechanical means, depending on the location, with exploitation of differences in density to effect separation. Gravitative processes are based on density-specific phenomena such as fall rate or radial acceleration for separating the feed within the sorting medium (air for dry sorting and water for wet-mechanical sorting) into two or more component flows, one of which contains mainly ore-bearing minerals and the other mainly tailings.

3.3.1 Gold pans

Gold pans are the first and foremost sorting implement in gold mining. In placer gold mining and primary ore mining alike, small-scale operations rely very heavily on the panning process. Indeed, many small-scale and artisanal mines use no other means of sorting. Thanks to their high selectivity, gold pans are ubiquitous and irreplaceable in all areas of small-scale gold mining, i.e., prospection, exploration and in-process analytics of production and preparation. Moreover, they are frequently used at the dressing stage for purifying preconcentrates. A gold pan is a simple, usually round apparatus with a trapezoidal (North American type) or triangular (South American type) cross section, though some oval and oblong forms are also in use. Pans for placer gold mining are normally much shallower than those used by lode-ore miners. However, the principle of operation is always the same: As the miner shakes the pan, the gold gathers at the bottom. The miner rotates the pan such as to keep its center almost motionless, while a combination of off-flow water and radial acceleration carries the tailings over the rim. The miner keeps repeating the process until only the gold, or the black sand containing the gold, is left in the pan. The last step of the panning process is to hold the pan at a slight angle and lightly tap the back of the rim in the direction of inclination. The result is similar to that of a concussion table, that is, the gold accumulates at the highest point of the concentrate plume. Gold pans are made of diverse materials:

metal	- wood
PVC	- rubber
split gourds	- horn

Wooden pans work best, although pans made of black PVC offer several advantages, i.e., they are crack-resistant, light and generally durable, and the gold shows up well. On the other hand, they are also water-repellent, a characteristic that facilitates gold flotation. Flotation is when gold is lost due to its hydrophobe surface by contacts with air-water-boundaries. The presence of even minute quantities of oil or grease can also lead very quickly to gold flotation. A few drops of detergent or of certain plant juices, e.g., sisal sap, can help avoid flotation.

The throughput of gold-pan sorting operations is generally rather low. If the feed sorts easily, a miner can handle about 100 panfuls weighing about 10 kg each in a day's work, thus putting the throughput at about 1 t/d. The feed should contain no particles exceeding roughly 30 mm in diameter, but the lower size limit for panning - if one exists - could be located at about 20 μm, which is quite invisible to the human eye. On the other hand, the finer the material, the higher the losses due to inadvertent discharge of fine gold particles. One special feature of gold panning is the possibility of simultaneous amalgamation. This entails rubbing mercury into the gold for about an hour with the aid of a boulder and then tapping the rim of the pan to re-unite the finely dispersed mercury. Due to the high surface tension of both natural mercury and mercury covered with microfine particles of oxidized minerals, the floured mercury resists reunification and therefore is carried out of the pan and enters the environment when the amalgam is washed. Consequently, the process is extremely hazardous and should not be practiced. Gold pans are very easy to make. A bit of simple sheet work suffices to turn the cover of an empty diesel drum into a gold pan. The best wooden pans are those made of rootwood. Small-scale plastic processors can produce gold pans made of PVC or PE.

Photo 8: Panning for gold in the Tipuani District, La Paz, Bolivia

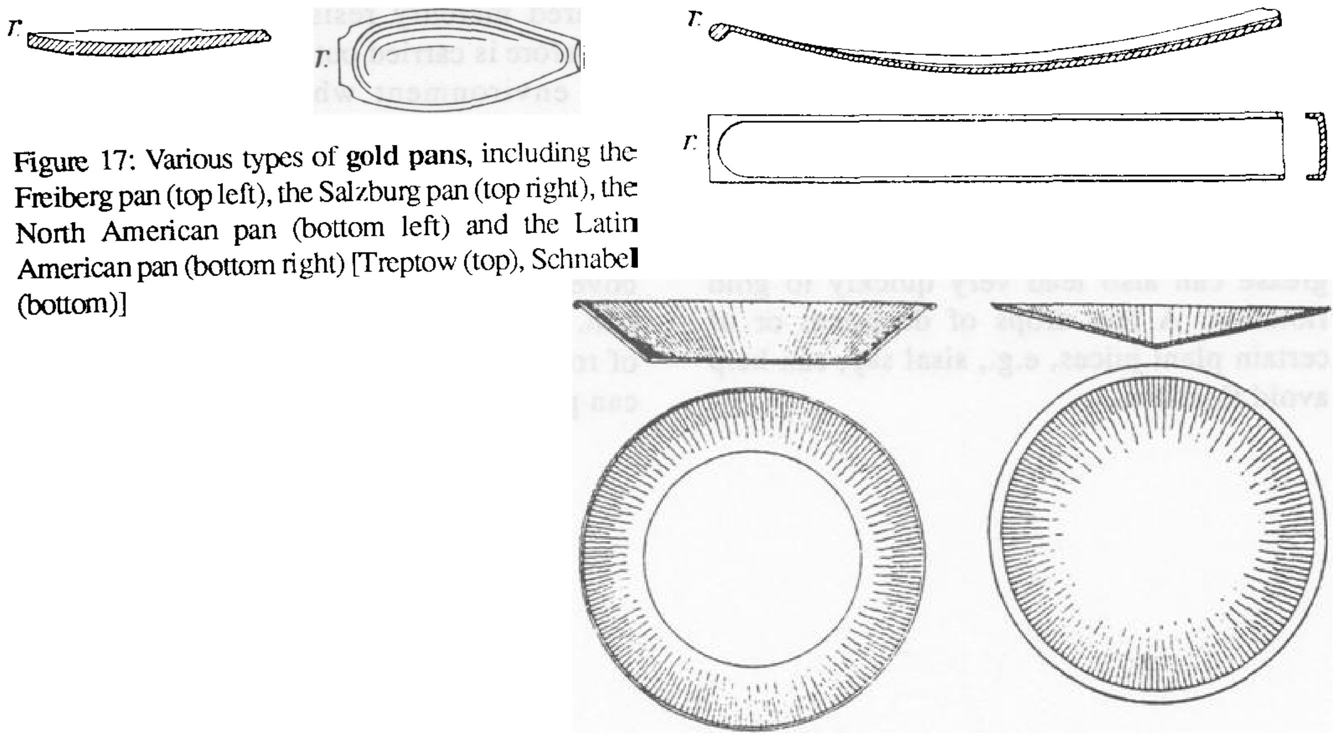


Figure 17: Various types of **gold pans**, including the Freiberg pan (top left), the Salzburg pan (top right), the North American pan (bottom left) and the Latin American pan (bottom right) [Treptow (top), Schnabel (bottom)]

3.3.2 Jigs

Jigs are in widespread use among placer gold miners. Often, they are included in floating processing plants and the larger hydraulic/shovel dredges. A pulsator generates an oscillating flow of water through a fabric screen holding a settling bed of pebbles (ragging). The material to be sorted is fed onto the settling bed. The water flowing up through the settling bed loosens the pile of pebbles and "unsettles" the settled material. Differences in specific gravity and particle size result in different trajectories with different amplitudes and wavelengths. The lighter particles are carried higher and farther. As the water recedes, the heavier gold, which, by reason of its high density, is by now situated at the bottom of the jig bed, penetrates the screen, while the lighter material is carried over the jig bed with the crossflow. The gold accumulating in the hutch is eventually recovered by opening the concentrate discharge valves at the bottom. If the ragging and the jig screen are well-matched, the screen fabric may plug up during the down-pulse, yielding a small volume of concentrate and an accordingly high concentration factor. The pulse frequency and amplitude are adjustable, as is the flow of hutch water, which enters the jig bed from below (through the screen). Differentiation is made between jigs with a constant flow of water, i.e., the suction type, and jigs in which the pump diaphragm's suction cycle determines the infeed of water, with control via a rotating-piston valve (suctionless). While one jig may differ from the next by reason of its pump diaphragm (pulsator) or jig bed geometry, they all operate on the same basic principle. The choice of jigbed material is essential. The use of lead shot (density: 11.3 g/cm^3) yields the best results, though steel shot is also used, if less frequently. The jig bed should be 7 - 12 times as thick as the upper particle size of concentrate for coarse material or about 20 times for fine material (< 2 mm). The jig-bed shot should not be more than 3 or 4 times as large as the upper particle size of concentrate. With a view to avoiding loss of jig-bed material, a hydraulic trap should be installed in the tailings discharge, so that any lost jigbed material can be recovered. Jigs can be operated on a suction cycle (cf. above) that makes the fines fraction trickle through more quickly and selectively, while the coarse fraction wanders very slowly through the ragging and screen. Jigs without suction work in the opposite manner. In any case, the concentrate and back-water discharge valves should always be left slightly open when the jig is in operation. Otherwise, the concentrate would not be drawn off continuously, and too much sedimentation could plug up the drains.

Jigs are suitable for local manufacture by experienced metal workers. The pump diaphragm is the most problematic component and may have to be imported. Another potential problem is the bushing - to the extent necessary - for the driving rods (plungers), which could involve problems with leaks and abrasion. The ragging can consist of normal lead shot, i.e., the commercial type used for hunting.

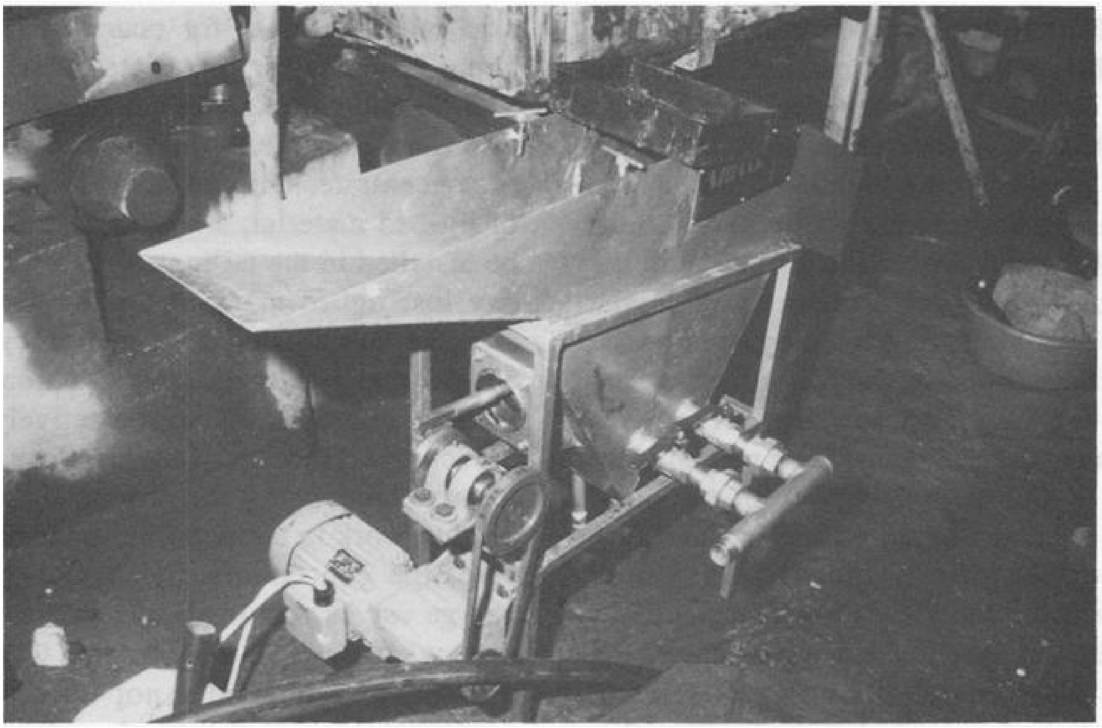


Photo 9: An Ecuadorian **fine-grain jig** with motor and reduction gearing at bottom left, belt-driven cam for the plunger (center) and inlet valves for upstream water at right. The concentrate discharge valves are visible at bottom center. Mina El Canada, Nariño, Colombia.

3.3.3 Sluices

Sluices are very widely disseminated in gold mining, primarily among small-scale placer gold mining operations and in the dressing of milled primary ore. Most miners prefer a continuous-type - or more exactly, a semi-continuous-type - sluice. Some intermittent-type sluices (with no riffles) are used in the purification of preconcentrates, e.g., from blanket tables, in the mining of primary ore.

Semi-continuous sluices extract gold and other heavy minerals from a flow of pulp with the aid of built-in riffles that promote sedimentation. The riffles retard the flow of material, creating turbulence and stagnancy, respectively, in front of and behind them. The heavy material settles out and accumulates in front of the riffles.
Intermittent-type sluices effect gravity separation with much less process water. A small amount of preconcentrate is placed in an inclined channel with a flat bottom and a flow of clear water. The water begins to carry off the lighter material, while the heavy material settles out by reason of its higher weight-to-incident area ratio. The operator continuously shovels the concentrate back to the feed zone and discards the tailings from the lower end of the mineral plume.

The aforementioned riffles are of crucial importance for the sorting performance of sluices. Various types of riffles and linings are used to obtain a "rough" bottom surface and alter the conditions of flow such that gold can collect between the riffles. The more familiar types of riffle/lining material include:

- wooden riffles
- gravel packs
- rubber matting (car mats)
- sisal mats

- fine and coarse fabric, e.g., corduroy, cord velvet
- split bamboo (one third of cross section) arranged in jalousie fashion
- metal section grid
- expanded metal screens
- nets made of meshed hemp or grass cords
- coarse screen fabric
- waste carpet

and various combinations of the above.

The riffles in gold-mining sluices are generally between 1 and 3 cm high and fastened to racks on 1 - 10 cm spacing. The racks can be wedged into the sluice, thus remaining easily removable. Gold recovery by sluicing can be enhanced by frequent clean-ups. Riffle racks tend to gradually clog up with heavy minerals, resulting in the loss of fine gold. The higher the heavy-mineral fraction of the feed material, the greater must be the volume of sedimentation space between the riffles in order to maximize the clean-up intervals.

Proper adjustment of the sluice's inclination is also very important for successful sorting. If the slope is too gradual (a very frequent mistake), the spaces between the riffles or gravel-pack boulders fill up with heavy minerals, resulting in a loss of turbulence and, hence, poor sedimentation of gold. If the sluice is too steep, the water washes the gold out of the sedimentation spaces. It is equally important to avoid fluctuations in material throughput and pulp density. If clear water is allowed to run through the sluice after a separating process, some of the sedimented gold is liable to be lost.

Small tandem sluices, i.e., double sluices, one of which sorts coarse material while the other attends to the fines fraction, accommodate the different sedimentation requirements of fine and coarse gold. The feed rate and/or angle of inclination can be chosen to obtain an optimal separating effect for the respective feed fraction.

Another means of improving the separation efficiency of a sluice is to artificially induce sluice-bed vibrations with the aid of an electric, mechanical or pneumatic vibrator. High-frequency agitation enhances the selectivity and reduces the required minimum length of sluice. Frequently, with a view to minimizing the requisite power input, part of the sluice floor is flexibly supported by rubber gaskets, and only that part is agitated. Such sluices are used mainly where large throughputs are necessary, e.g., in small, mobile gold dressing plants or placer-gold processing facilities.

Amalgamation in sluices using mercury in the riffles' interspaces is a frequent practice. Chapter 3.5 relates the relevant operating experience and sorting characteristics.

Photo 10: Dressing of placer gold ore in a portable **sluice** along Rio Guadas, Valle del Cauca, Colombia

Photo 11: Expanded-metal linings (coarse and fine) in a gold-dressing **sluice** near Copiapó, III Region, Chile

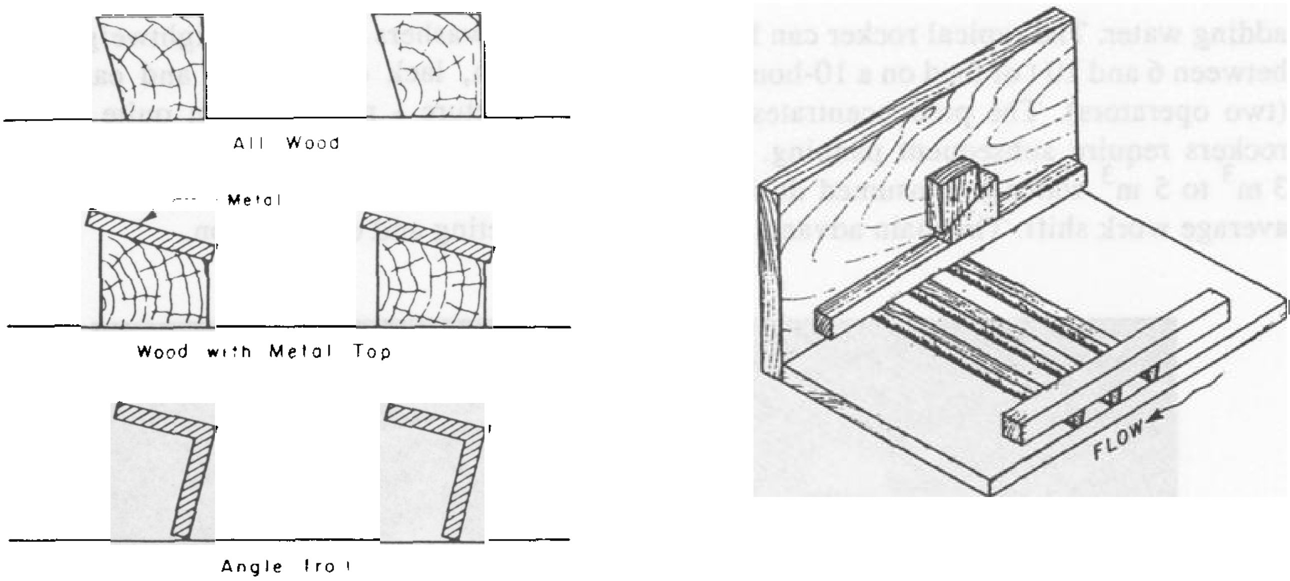


Figure 18: Various types of wooden and metal riffles and their arrangement in a sluice [Silva]

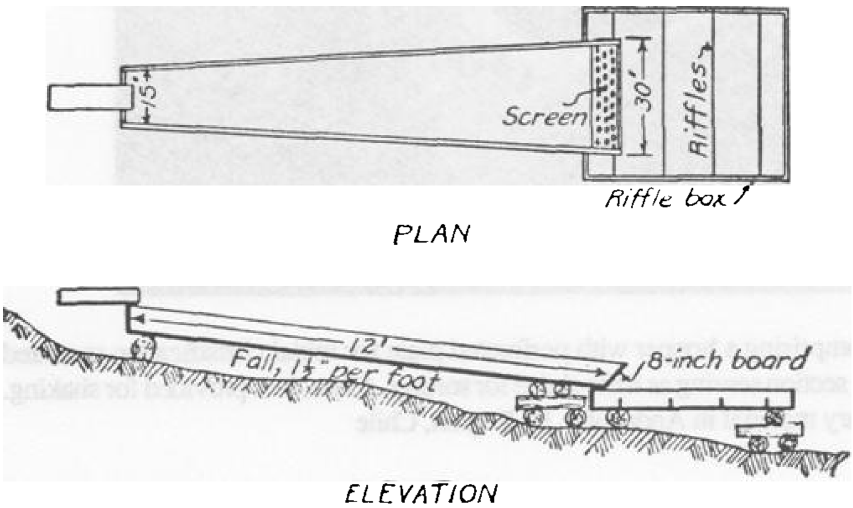


Figure 19: A semi-continuous sluice [Bernewitz]

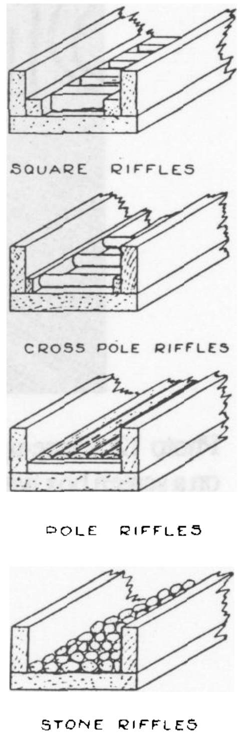


Figure 20 (at right): Various types of wooden and stone riffles [Stout]

3.3.4 Rockers

Rockers, or rocker washers, serve in the dressing of loose and slightly consolidated gold-bearing sediments, primarily in relatively dry regions. Basically, a rocker consists of a classifier and a sluice. The classifier is a deep screening boxcumhopper for accepting the feed. From under the box emerges an inclined wooden riffle sluice, the slope of which varies according to the size of gold recovered.

Clayey feed requires less slope than would be needed for coarser material. The whole unit is mounted on semicircular skids (the actual "rockers"), so that the entire upper section can be rocked back and forth sideways with the aid of a lever. Since the feed material and the washing water have to be shoveled/poured in by hand, as many as four people are needed to operate a rocker: one for extracting the raw ore; one for hauling it to the rocker and loading it into the hopper; one for shaking the rocker; and another for adding water. The typical rocker can handle between 6 and 10 t of feed on a 10-hour shift (two operators). The preconcentrates from rockers require subsequent panning. Some 3 m^3 to 5 m^3 water is consumed during an average work shift. The main advantages of rocker washers are their lightweight construction, lack of a motor and easy local manufacture - making them quite suitable for use as mobile/portable dressing devices for sedimentary gold ore within the scope of prospecting and exploitation.

Photo 12: Close-up view of a **rocker** comprising a hopper with perforated plate for initial classification mounted on a screen box with apron, and a bottom section serving as riffle sluice for sorting. A handle is provided for shaking. Used for extracting gold from sedimentary material in Andacollo, IV Region, Chile

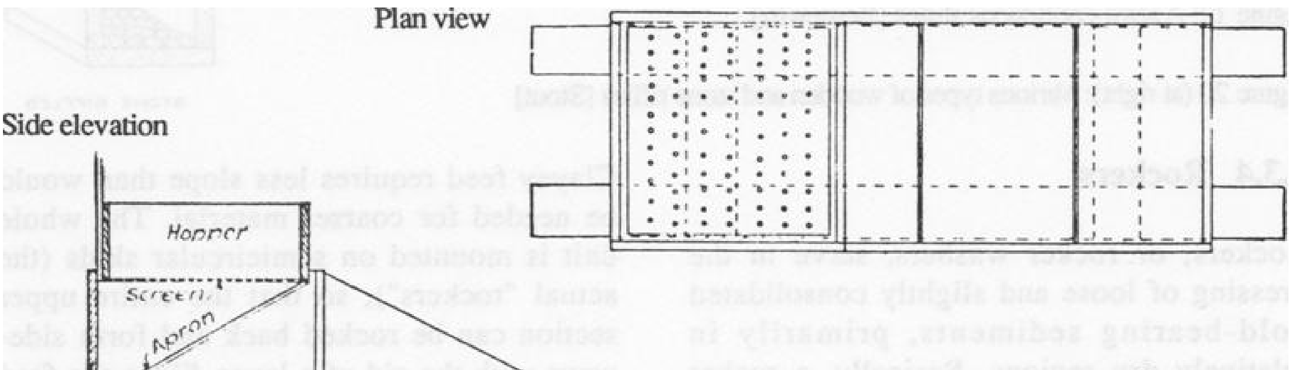


Figure 21: Two views of a **rocker-washer** [Bernewitz]

3.3.5 Dug-channel sluices

Dug-channel sluices are used almost exclusively in placer gold mining. They consist of a trench, with or without masonry walls and with varying floor conditions and angles of inclination. As unclassified feed flows through as thick pulp, some of the material settles and is treated as a preconcentrate. The use of dug-channel sluices involves high gold losses, particularly in the fine-particle range.

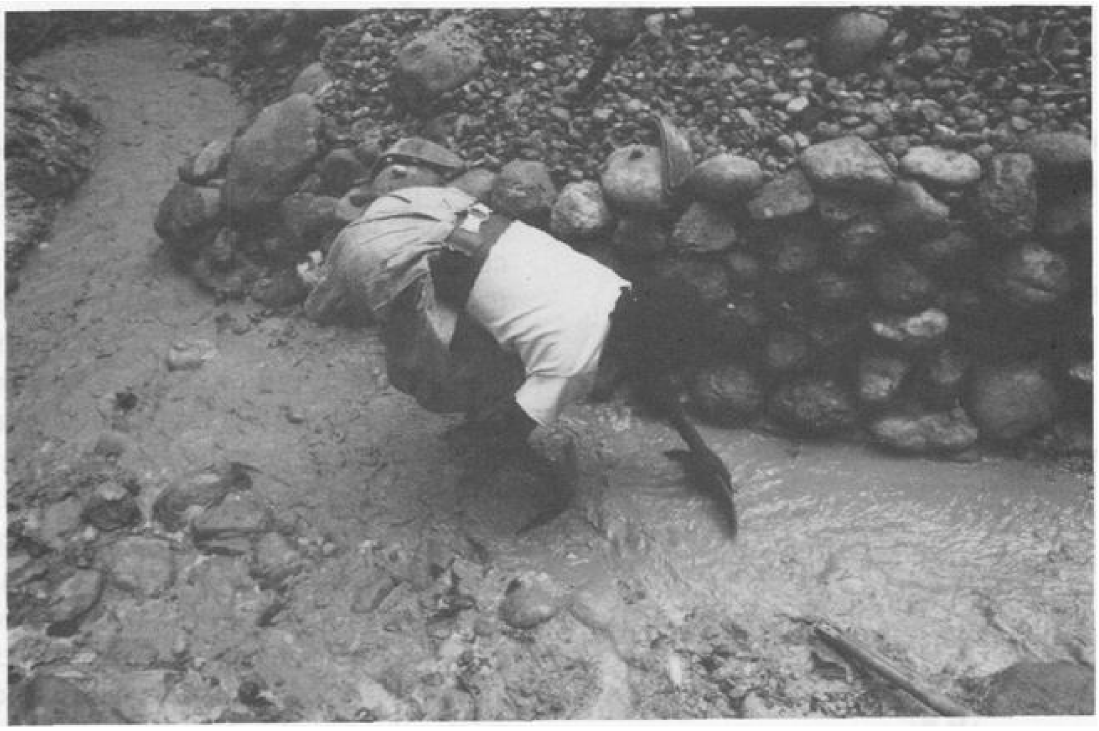


Photo 13: A dug-channel sluice for preconcentrating auriferous placer material. Wooden shells (cachos) are used to remove lightweight sediment. Located along Rio Telembi near Barbacoas, Nariño, Colombia

3.3.6 Pinched sluices

Pinched sluices are used for preconcentrating presized medium, fine and ultrafine particle collectives. They can be used individually or as "packages" in the form of conical concentrators (Reichert cones) or pinched concentrators (Cannon/Carpco concentrators). In any case, the sluices in question have convergent cross sections that turn a wide, shallow flow of pulp into a narrow, deep flow of pulp. Heavy minerals migrate to the bottom. Splitters located at the discharge end separate the mineral bands, with the lighter material on the outside (upper) and the heavier material on the inside (lower) part of the original flow. Like any other sluice, the pinched sluice is highly dependent on an optimal angle of inclination for achieving good separation efficiency. The best angle is one that is just slightly larger than the angle at which the first heavy material begins to settle on the floor of the sluice. The low relative velocity between the feed and the process water allows the use of pinched sluices for selective sorting of fine-grain collectives and material containing tabular and flaky gold. One of the basic characteristics of pinched sluices and conical separators is that they only remain adequately selective for heavy-mineral contents of 70 % or less in the concentrate. Thus, pinched sluices are used mostly for purposes of preconcentration. The splitters can be adjusted for high sensitivity. Simple pinched sluices are well-suited for local manufacture. Most are made of wood or metal with wear-resistant rubber or plastic liners.

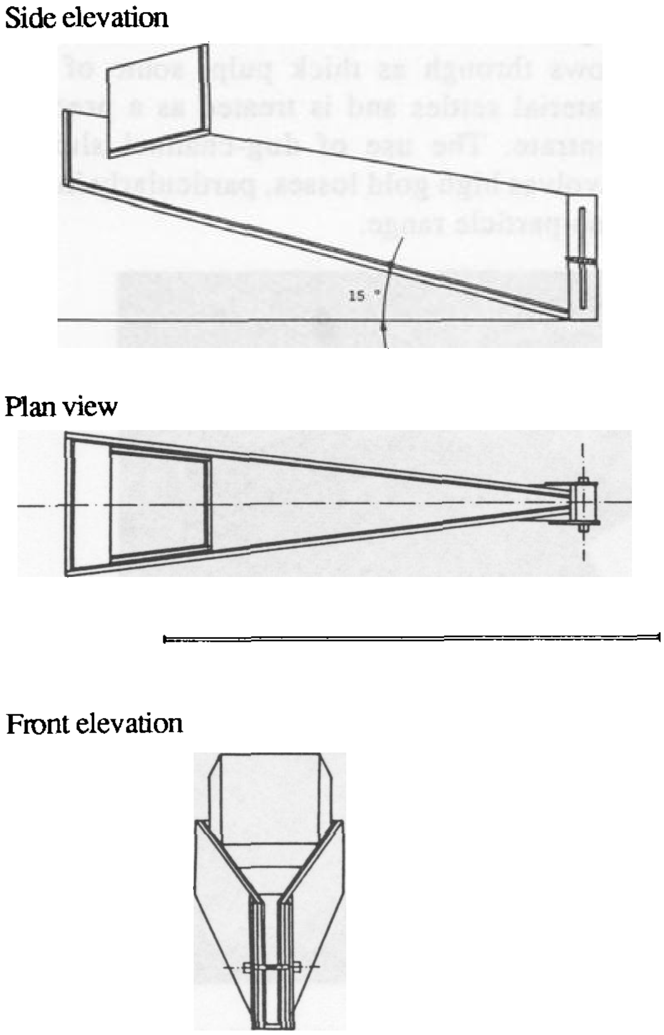


Figure 22: Side elevation, plan view and front elevation of a **pinched sluice** [Priester]

3.3.7 Spiral concentrators

Spiral concentrators are sorting devices used mainly for preconcentrating ores of medium particle size. The typical spiral concentrator consists of a helical sluice with four to six turns. The sorting effect results from vertical separation according to density, with the heaviest particles collecting at the bottom of the spiral sluice section, where friction and drag act to slow the material down. Due to the spiral shape of the sluice bed, the separating effect is enhanced by differences in centrifugal forces that lead to horizontal gravity separation by affecting the lighter, faster-moving material (situated toward the top of the flow) more than the heavier, slower material (near the bottom) as the pulp flows along the spiral-shaped chute. The lighter, faster-moving material thrusts outward toward the rim of the spiral, while the slow-moving, heavy material remains inward, toward the bottom, and the medium-dense particles take up a position in-between. At the end of the spiral, splitters divide the take-out into three fractions: concentrates, middlings and tailings. Modern spiral concentrators are characterized by relatively shallow channel cross sections (for concentrates) that broaden toward the bottom. This shifts the separating boundary toward the outside, thus enhancing the centripetal influence. Simultaneously, the flow of concentrate becomes shallower, giving rise to tangential waves that improve selectivity by removing lighter particles from the concentrate zone. The spiral shape of the sluice makes it possible to combine a number of sluices into a single unit. Most spiral concentrators are now made of plastic or fiber glass-reinforced synthetic resin. Models of relatively simple design such as Reichert spirals have gradually superseded the more complicated types (Humphrey spirals) with wash-water feed lines and concentrate take-off lines at various points along the spiral. Having no motor, spiral concentrators are very well-suited for use in small-scale mining. A single concentrator can handle about two tons an hour. On the other hand, spiral concentrators have relatively limited (feed) particle-size ranges. The best sorting results are obtained for a feed size of 0.03 - 2 mm. When ordering a spiral concen- trator, care must be taken to stress the fact that a low-grade unit is required to match the low gold content of the feed ore. Otherwise, the user will soon have large amounts of preconcentrate to deal with. The somewhat complicated construction of spiral concentrators makes them generally less suitable for local manufacture. Assuming adequate demand, they could be produced on an industrial scale by specialized plastic-processing companies.

Photo 14:
A Humphrey spiral for pre-concentrating ore at the Kalauyo tin mine, La Paz, Bolivia

3.3.8 Tables

Apart from gold pans and sluices, tables are about the most widely used devices for sorting by wet-mechanical gravity separation in gold mining. They are used for obtaining concentrates and preconcentrates from medium, fine and ultrafine particle collectives. Differentiation is made between various elementary types:

- sweeping tables,
- blanket tables,
- endless-blanket tables,
- tilting tables,
- concussion tables, and
- shaking tables.

Round, conical and funnel-shaped tables are not widely used in gold mining. All tables have a large sorting area, across which a shallow flow of pulp undergoes separation.

In the case of static tables (sweeping, blanket, tilting tables), the face of the table is designed to let the heavy material settle, while the off-flow of pulp carries off the lighter material. On a sweeping table, the product is recovered by sweeping it off of the table with the aid of a broom, wiper or wash water at the end of each batch sorting operation. Blanket tables have fabric or canvas liners to enhance the adherence. The product is recovered by washing the concentrate out of the fabric/canvas. Tilting tables, which usually have several decks, are tilted over to recover the concentrate, which is washed off by water into a receiving tank - e.g., the Bartles-Mozley table. Such tables are used primarily for concentrating very fine-grained material.

Moving tables (endless-blanket and shaking tables) separate the heavy material from the light and medium-heavy material by the effects of pulp-flow differentials, induced motion and riffling. The endless-blanket table has an endless fabric deck that travels around a pair of rollers. The concentrate collects between the riffles and drops into a receiving wagon as the revolving deck turns it upside down, while the light and medium-heavy material is carried off with the opposing stream of slurry and wash water. The separating effect of shaking tables derives from the stratification of material with different densities, particularly between the riffles, as a result of longitudinal, asymmetrical shaking in combination with a crossflow of wash water, which separates the particles according to size and density. The riffles become thinner in the direction of movement along the deck, thus first exposing the light, then the medium-heavy and, finally, the heavy material to the crossflow of water. The smallest, heaviest particles resist incident movement most effectively, so the gold plume is easy to see against its black background. The gold is followed, with some overlap, by arsenopyrite, pyrite, various sulfidic accompanying minerals and, finally, the gangue material. Splitters positioned at the end of the table allow separate recovery of the different products.

Concussion tables are something of a cross between a static and a moving table. They essentially consist of a flat, rectangular settling tank. Slurry entering the feed end is distributed homogeneously across the width of the table by the effects of a head-end spreader (Happenbrett). The pulp runs across the inclined table in longitudinal direction, depositing the heaviest material near the starting point and carrying the successively lighter material further on. The deck is subjected to longitudinal jolts in order to enhance the selectivity by repeatedly loosening the sediment. This is accomplished by a camshaft that pushes the suspended table out of line, making it swing back against a buffer. The resultant concussion is transferred in varying degrees to the material to be separated. This causes the density-stratified material to form a number of component flows in mutually opposite directions. The heavy material, i.e., the material situated next to the deck, receives the heaviest impulse and "climbs" the table in the opposite direction as the flow of slurry. The lighter material above it receives a somewhat damped jolt and is more directly exposed to the force of the flowing water, in which it becomes entrained. After a certain length of time, no more feed material is added, and the concentrate can be recovered with the aid of shovels, spatulas, brooms and/or wash water.

All tables rely on extensive homogeneity of feed volume and density - particularly the latter - since any fluctuations alter the conditions of transport by the off-flow of water.

The local-manufacturing options vary according to the type of construction. While sweeping and blanket tables can be of simple wood or concrete construction and built in situ, rocking and concussion tables require high-precision wood/metal work that should only be executed at a properly equipped workshop. Shaking tables involve complex drive assemblies for the asym- metrical shaking motion and therefore require semi-industrial production facilities.

Photo 15: **Concussion table** with a waterwheel as its prime mover for sorting gold ore. The spreader at the front (left) end of the deck helps homogenize and distribute the pulp feed; La Llanada, Nariño, Colombia

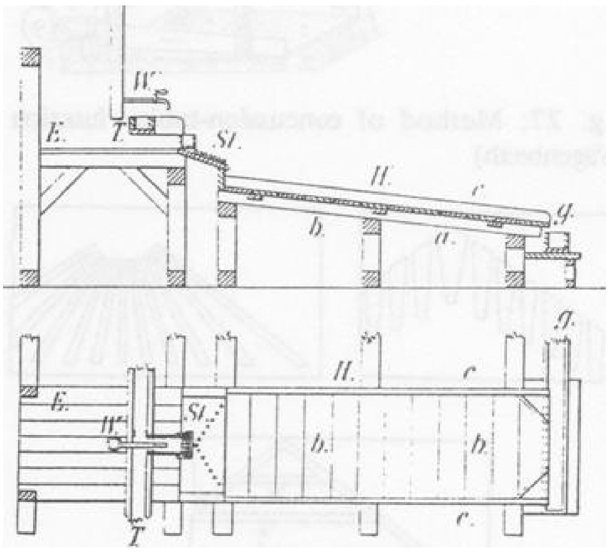


Figure 23: **Sweeping table** [Treptow]

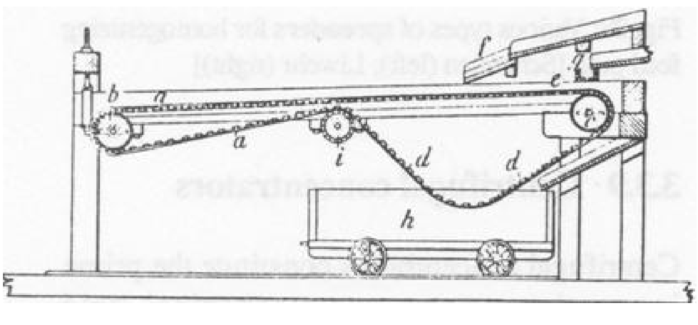


Figure 24: **Endless-blanket** (belt) table [Schennen]

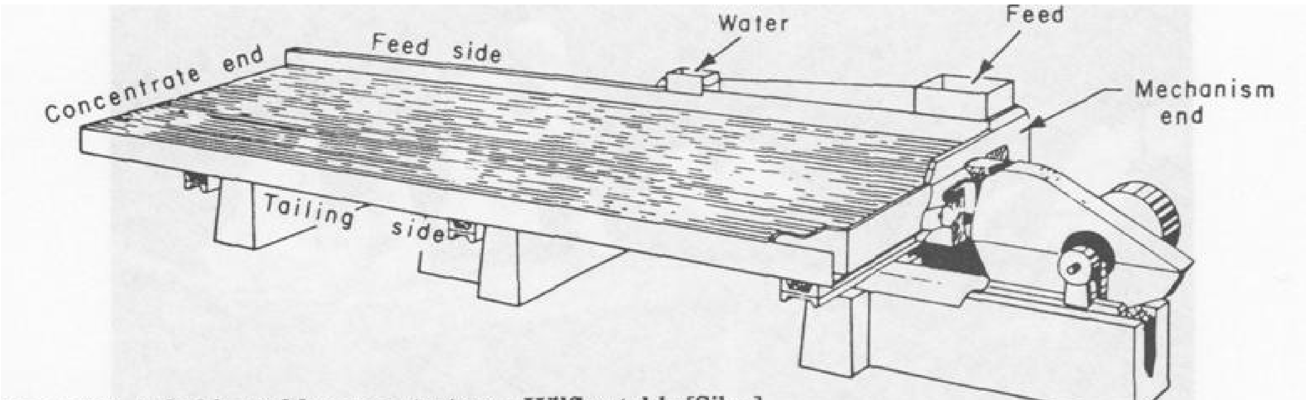


Figure 25: A shaking table concentrator or Wilfley table [Silva]

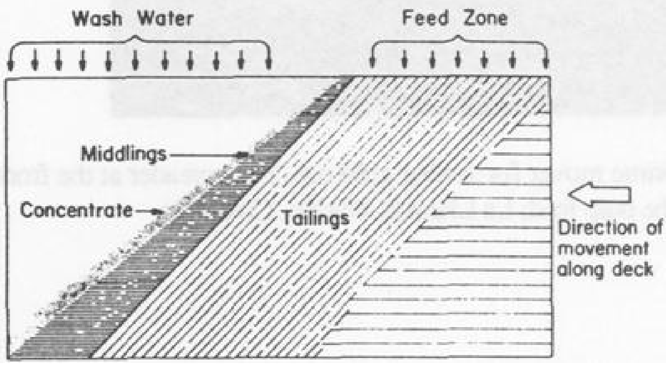


Fig. 26: Idealized mineral separation on a shaking table [Silva]

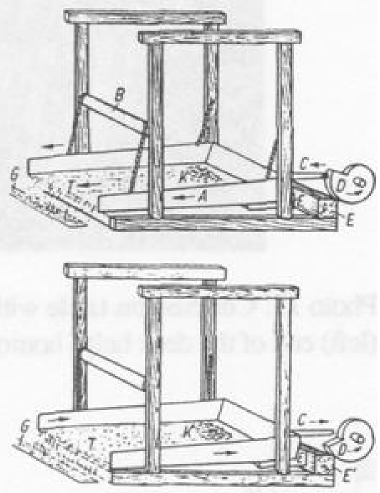
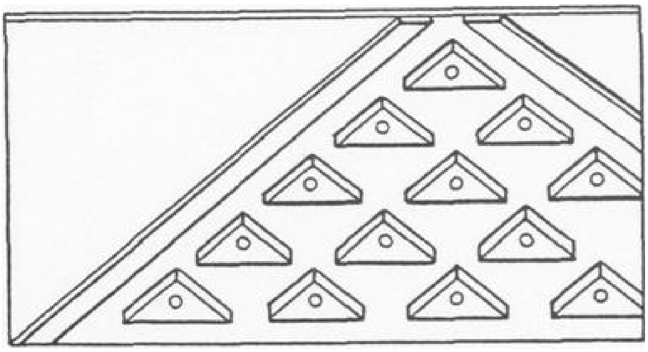


Fig. 27: Method of concussion-table function [Wagenbreth]



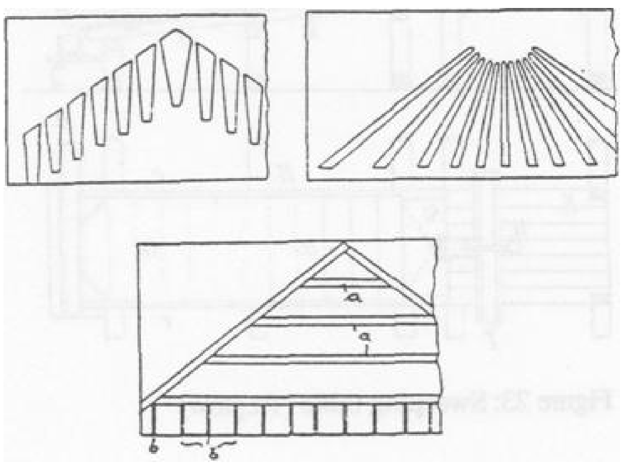


Fig. 28: Various types of spreaders for homogenizing feed pulp [Schennen (left); Liwehr (right)]

3.3.9 Centrifugal concentrators

Centrifugal concentrators constitute the prime innovation among wet-mechanical gold sorting implements. Within a brief time span, they have gained widespread acceptance in the sorting of fine and microfine particle collectives, especially in placer gold mining, although they are also gaining ground in primary ore mining. One special application for centrifugal concentrators is the recovery of gold as a byproduct, e.g., in gravel quarries. Their very high throughput rates allow direct incorporation into the main material flow. All centrifugal concentrators operate on the same principle, namely that a rotating receptacle effects gravitative separation of the feed in a centrifugal field. The more familiar types include:

- ***Knelson centrifuge.*** This is a dynamic concentrator with a radial acceleration force that can reach 60 g. Ore slurry is introduced through a pipe at the bottom of the unit and forced up the annular, slightly conical bowl by centrifugal force. A counterflow of back-pressure water is pressed into the annular spaces to keep the bed of heavy particles fluidized. While heavy particles are forced out against the walls and are trapped between the ribs, the lighter particles are carried by the water flow into successive annular spaces and eventually out over the rim.

- ***Knudson centrifuge.*** The Knudson bowl concentrator is of the same basic design, but without counterflow water, a fact which extensively simplifies its construction. Instead, this type of concentrator employs a stationary eccentric in its rotating bowl as a means of fluidizing the material in the annular spaces by inducing flow turbulence, thus cleaning the sedimented concentrate. Both types of centrifuge are shot down after a certain length of time, normally at the end of a shift, for cleanup, i.e., to flush the concentrate out of the annular spaces. Knudson bowls require more frequent cleanups then do Knelson centrifuges, meaning that the latter yields the more heavily concentrated product.

- ***Falcon centrifuge.*** The Falcon-type concentrator consists of an upright rotating cylinder with a partially conical inner wall. Ore slurry is fed onto a rotating distribution plate through a central feed pipe. Centrifugal acceleration forces the material outward until it meets the conically widening centrifugal wall. Here, the pulp undergoes radial gravitative separation in that the heaviest particles "stick" to the smooth wall while the lighter particles flow up and out of the cylindrical top section of the centrifuge. The retained concentrate forms a wedge-shaped ring that is flushed out with wash water when the centrifuge is stopped. During washing, the solid particles are flushed into a concentrate receiver through a hollow shaft between the distributing plate and the wall of the centrifuge.

All centrifugal-type concentrators are completely reliant on the use of fully liberated/slurried feed. Clayey sediments and gold-bearing laterites, for example, may require elaborate preparation prior to centrifugal sorting. If the feed material contains very heavy minerals such as arsenopyrite, they are recovered as part of the concentrate. If the heavy-minerals fraction is too high, the selectivity of the centrifugal sorting process suffers as a result.

In selecting a centrifuge, close attention must be given to optimal service conditions for each type under consideration. The Knelson centrifuge, for example, requires a supply of clear water for the counterflow that prevents plugging of the small perforations in the cone; otherwise, the mineral grains collecting in the annular spaces would become too tightly compacted. Also, the water pressure must be kept constant, since any increase in pressure or drop in pulp feed rate would result in the loss of fine concentrate, most particularly of particles with large specific surface areas. Simple centrifuges have no such control problems, but Knudson and Falcon centrifuges do have relatively short cleanup intervals that must be taken into account. In addition, compared to a Knelson centrifuge, the Knudson and Falcon types can not recover the same fineness of material. On the average, the recoverable particle size ranges between 30 µm and 1 - 4 mm, depending on the type and model of centrifuge in use. The good results of centrifugal sorting have led to widespread attempts at the local manufacture of gold centrifuges. However, the more elaborate designs, e.g., those based on the Knelson concentrator with its thermoplastic-lined bowl, can only be fabricated in an industrial environment. Moreover, the reproduction or license manufacture of centrifuges involves serious bearing problems. While it is relatively easy to even dynamically balance an empty centrifuge, the presence of inhomogeneously sedimented particles spinning at a high velocity can seriously unbalance the centrifuge to the point of damaging its bearings.

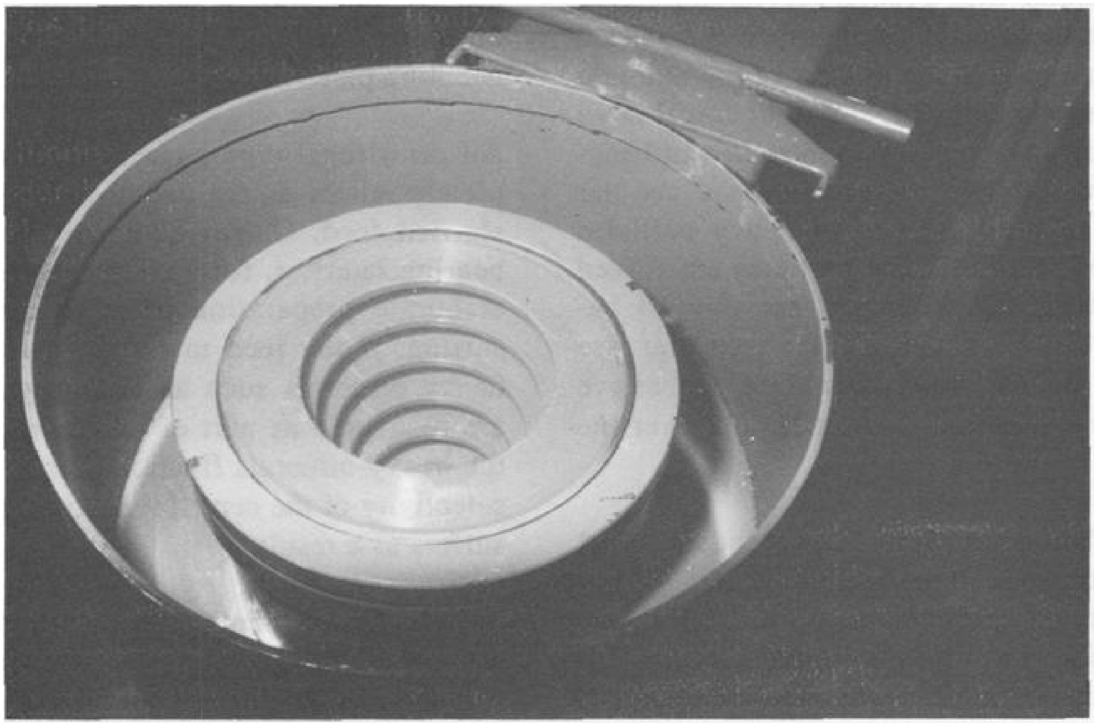


Photo 16: View into the separating chamber - or bowl - of a 7.5" **Knelson fluidized-bed centrifuge.** At center, the polyurethane bowl with its concentric annular spaces mounted in a perforated conoidal steel frustum, through which the counterflow water is pressed in. The bowl can be removed to recover the concentrate. The frustum is surrounded by a collecting bowl of somewhat larger diameter, where the lighter material accumulates [Mining Exhibition in Düsseldorf, 1989]

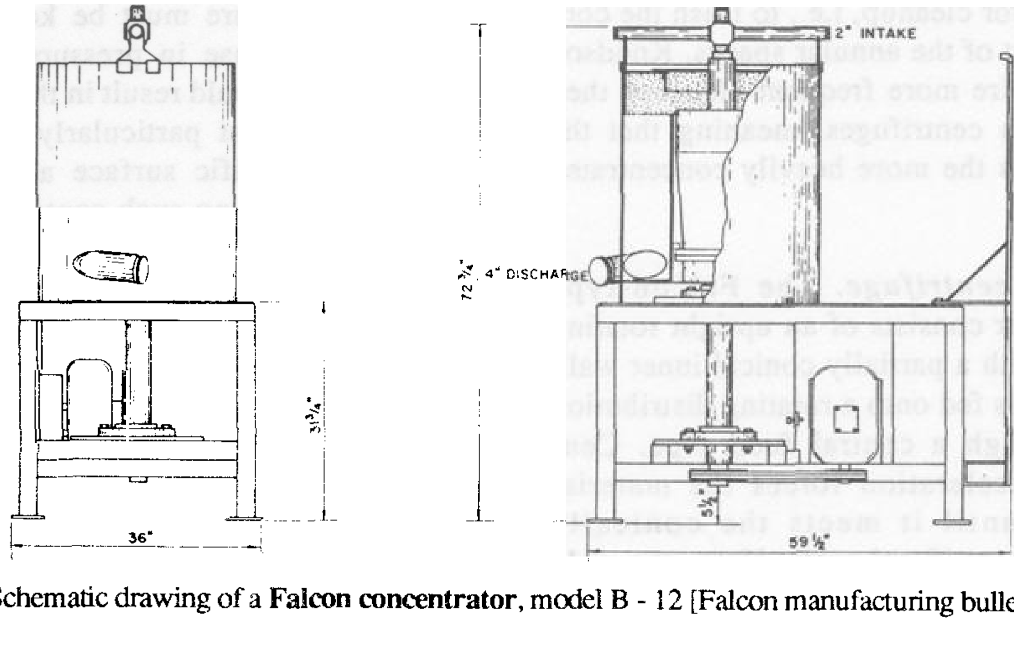


Figure 29: Schematic drawing of a **Falcon concentrator**, model B - 12 [Falcon manufacturing bulletin]

3.3.10 Hydraulic traps

Hydraulic traps are used for effecting the prior separation of coarse gold particles (grains, nuggets, etc.) in order to obviate the need for their subsequent milling, amalgamation or leaching. Hydraulic traps are also widely used for recovering amalgam and mercury downstream of amalgamating stages, e.g., stamp mills or amalgamating copper plates. Hydraulic traps differ according to design and function. Some hydraulic traps are designed for a counterflow of water (cf. fig. 31 and photo 17). They work much like a small, artificial settling tank positioned in the main flow of pulp. They do not interrupt the flow, but only allow heavy particles to settle out. The counterflow of water keeps the sedimentation chamber largely free of lighter particles. In that sense, a hydraulic coarse-gold trap is comparable to a single-section spitzkasten with clear-water counterflow. The sedimented concentrate can be recovered with the trap in operation by opening a discharge valve at the bottom of the trap. In other hydraulic traps, the pulp enters through an inlet pipe and is forced to change directions a number of times before escaping. The gold, mercury or amalgam separates and settles to the bottom. This makes such systems analogous to round or lamellar thickeners.

All aforementioned types of hydraulic traps are well-suited for local manufacture, particularly with access to commercially available components like ball valves, pipe sections, fittings, etc.

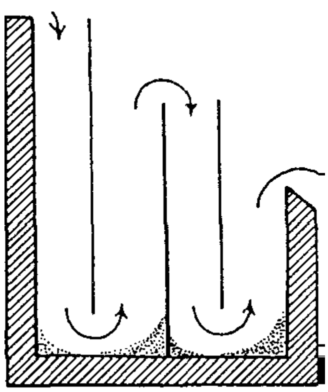


Fig. 30: Functional schematic of a **simple hydraulic trap** for mercury and amalgam [Escobar Alvarez]

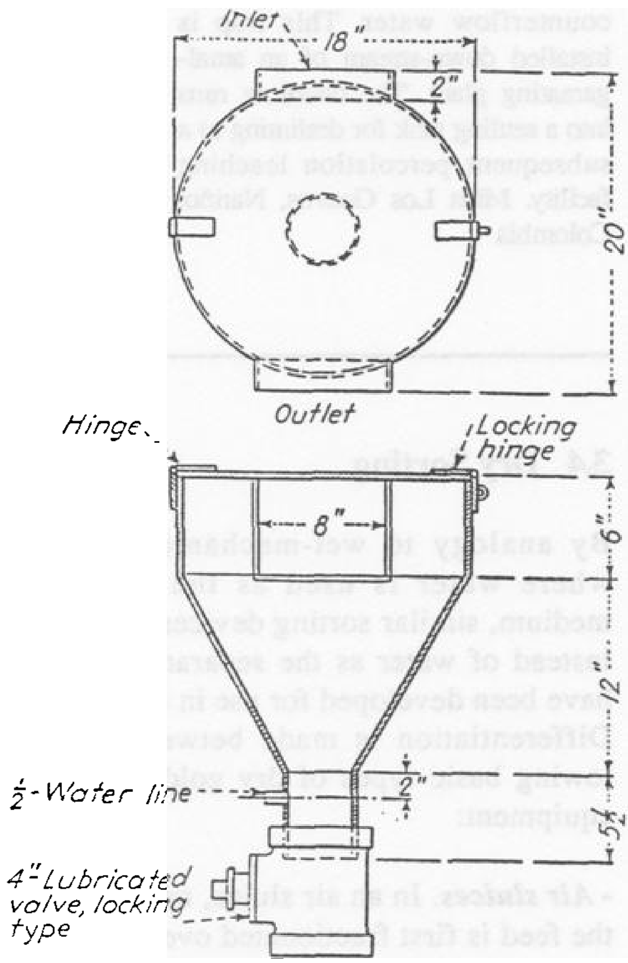


Fig. 31: **A hydraulic trap** [Bemewitz]

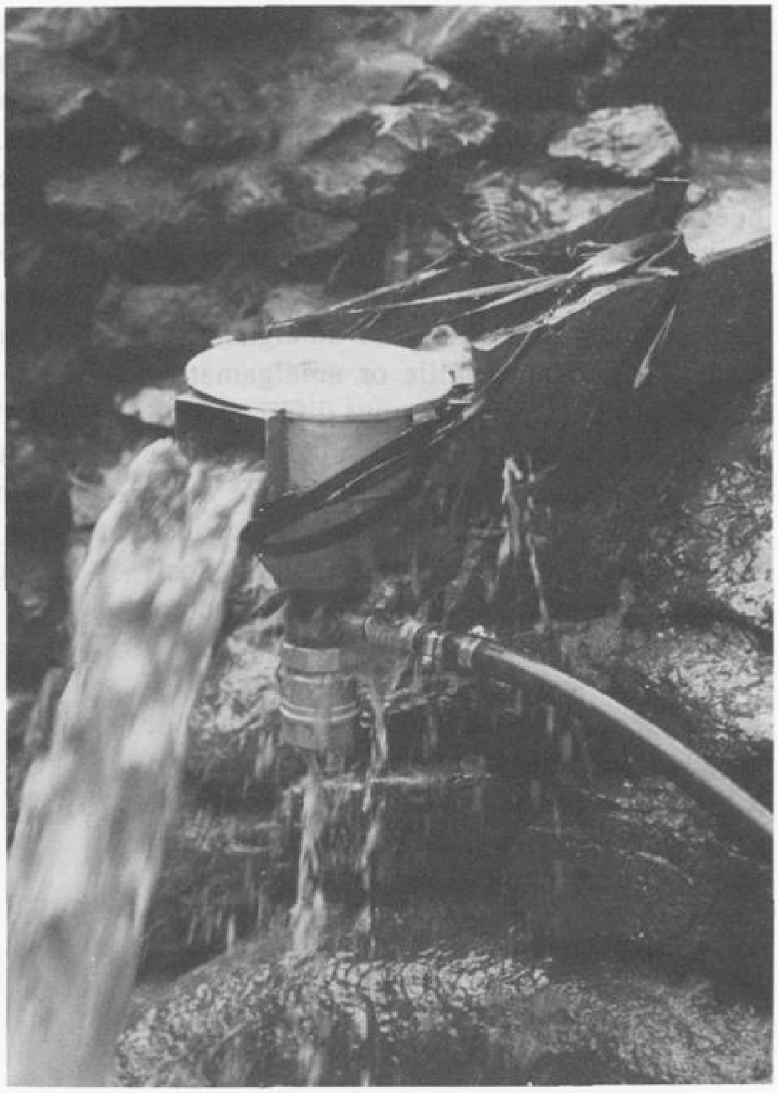


Photo 17:
A locally manufactured **hydraulic trap** in Colombia, equipped with a ball valve for controlling the flow of counterflow water. This trap is installed down-stream of an amalgamating plant. The overflow runs into a settling tank for desliming to a subsequent percolation leaching facility. Mina Los Guavos, Nariño, Colombia

3.4 Dry Sorting

By analogy to wet-mechanical sorting, where water is used as the separating medium, similar sorting devices that use air instead of water as the separating medium have been developed for use in arid regions. Differentiation is made between the following basic types of dry gold-ore sorting equipment:

- ***Air sluices.*** In an air sluice, or dry blower, the feed is first fractionated over a series of inclined screens. A bellows under the box blows puffs of air through the deck of the box, which consists of canvas or some other fabric. The incoming air keeps the oversize fluidized and, hence, capable of vertical gravitative separation. Due to the inclination, the fluidized bed tends to flow downward past riffles that hold back the heavy material, while the waste passes over the riffles and out of the machine.

- ***Dry washers.*** Dry washers similar to air sluices, but without pneumatic fluidization, are also used for dry sorting. Instead of generating a fluidized bed, the material is shaken to effect gravitative separation. Consequently, i.e., due to the lack of loosening action, dry washers achieve less selectivity and recovery of values than do air sluices.

- ***Air tables.*** In many arid regions, air tables, or pneumatic shaking tables, are used for processing gold sand, coal, etc. An air table consists of a deck covered with a porous material over a chamber out of which air is blown up through the tiltable deck. The deck's angle of inclination, together with lateral shaking, divides the fluidized bed into a heavy-material and a light-material zone. Such equipment is characterized by a high power requirement and extensive airborne dust emission. The latter necessitates either a closed-cycle airflow, use of a dust remover in a dedusting chamber, or inclusion of a dry cyclone separator. Air tables have the advantage of producing pneumatic-gravitatively sorted concentrates that require no drying. Their drawback is their poor selectivity.

- ***Winnowing.*** Also known as dry blowing, winnowing is a popular method of dry sorting. The feed falls through a stream of air that deflects the heavy particles less than the lighter ones, since the latter have a higher incident-area-to-weight ratio. Splitters can be used to separate the component flows. Winnowing should be reserved for carefully classified, completely dry, fine feed. The extreme dust emission constitutes a serious environmental problem. Though a cyclone, wet air scrubber or closed-cycle arrangement would substantially reduce dust emissions, it would also substantially increase the cost of operation.

- ***Pinched sluices*** with pneumatically fluidized bed. Pneumatic sorting, e.g., in arid regions, can be effected on pinched sluices with pneumatically fluidized beds. Air blown up through a fabric deck generates a fluidized bed of feed material, and sorting is effected in much the same manner as in a hydrodynamic pinched sluice. A very uniform particle size and substantial differences in specific gravity are the main prerequisites for good sorting. Dry pinched sluices are used mainly for obtaining preconcentrates.

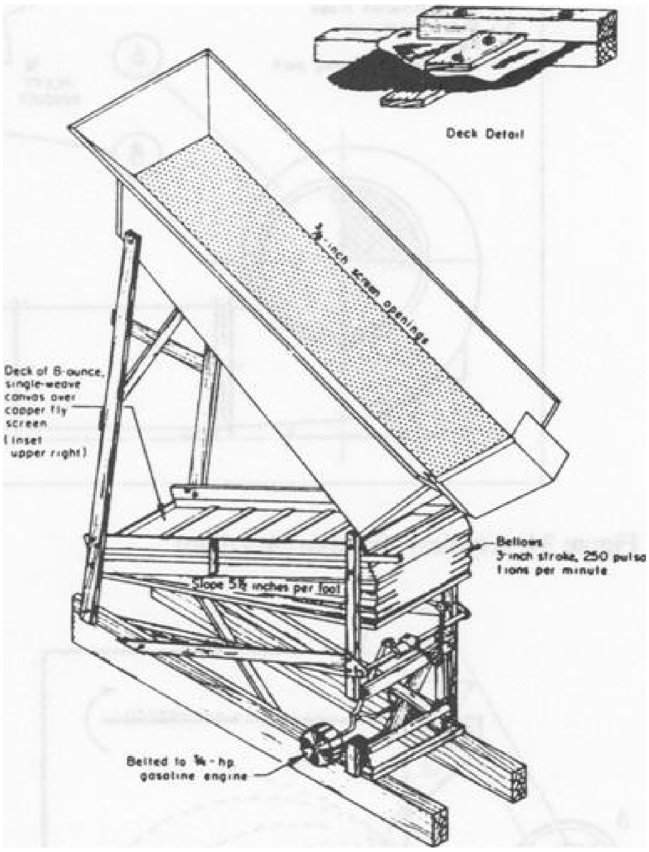


Figure 32: A typical **dry blower** [Silva]

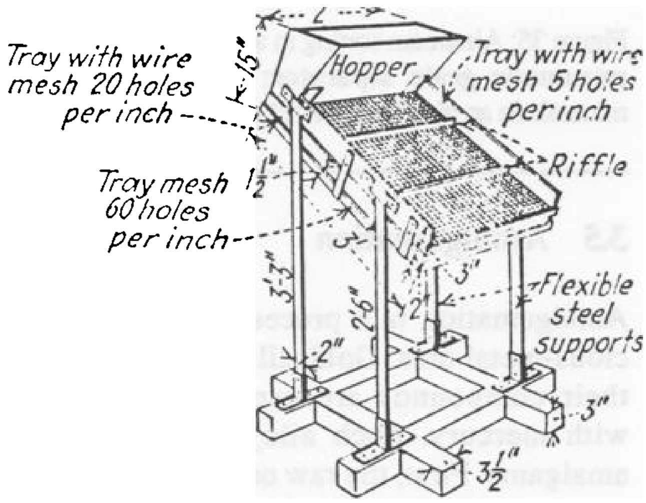


Figure 33: A typical **dry washer** [Bernewitz]

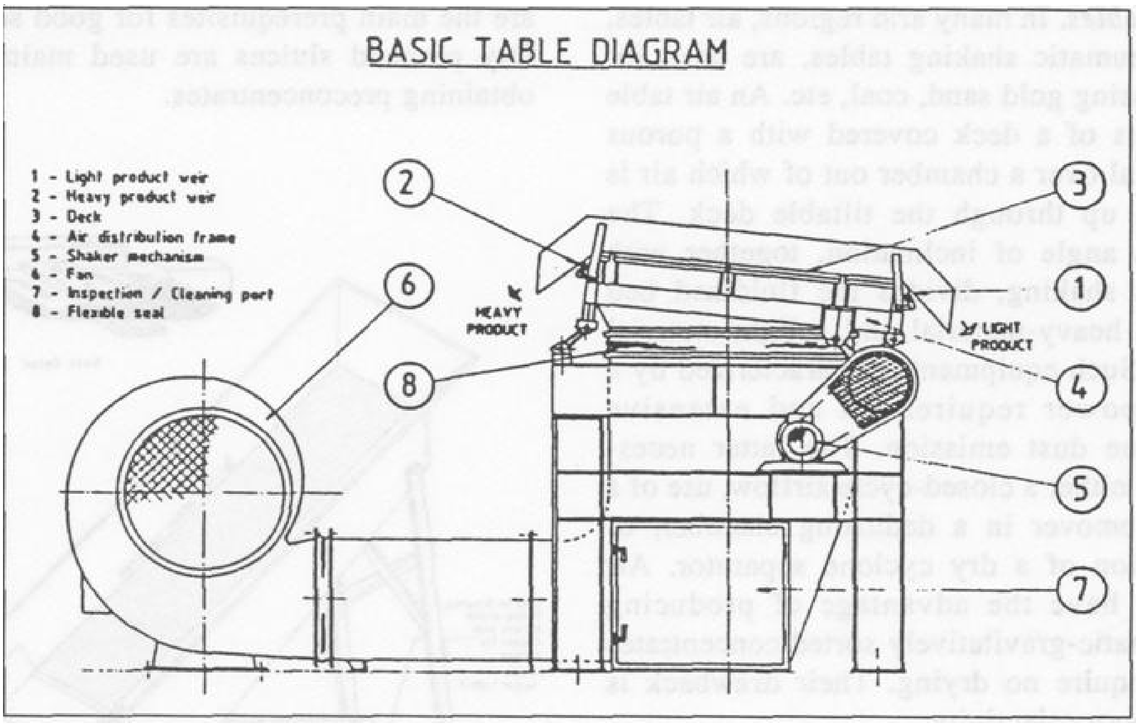
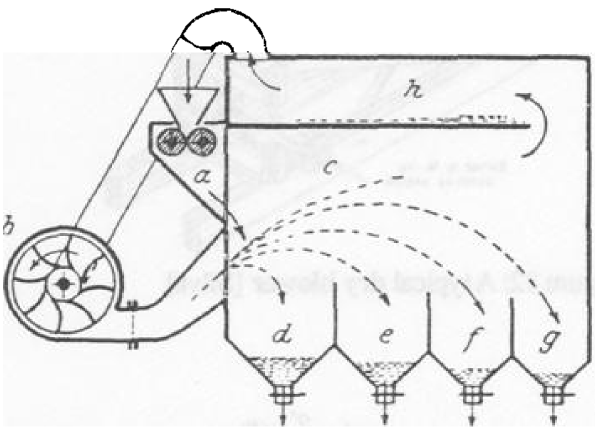


Figure 34: A typical **air table** [Ackthun]



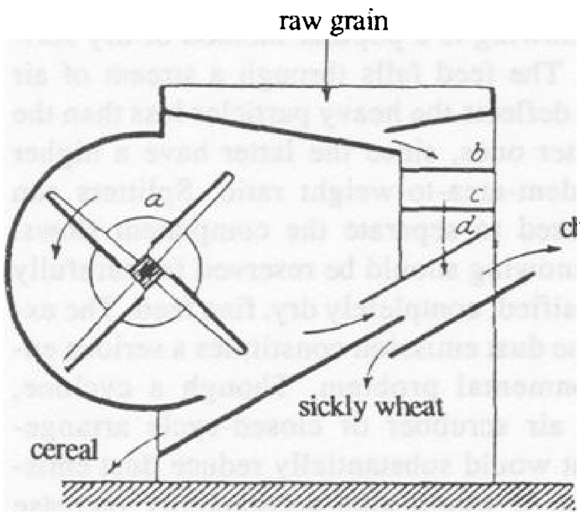


Figure 35: Airstream sorting in a winnower. At right: a **pneumatic grain separator**; at left: sorting of minerals in an **air classifier** [Fischer]

3.5 Amalgamation

Amalgamation is a process applied to precious-metal ores. Gold, silver and several of their compounds are capable of alloying with mercury. Such alloys are known as amalgams. First, the raw ore is processed together with mercury; then the amalgam is separated and distilled (retorted) to separately recover the precious metal and the mercury.

Amalgamation can be accomplished in gold pans, sluices, drums, tubs, amalgamating barrels, edge mills, stamp mills and amalgamating tables.

Since the work involves the use of highly toxic reagents, i.e., mercury, the process poses a serious hazard for the user's health and the environment. Consequently, the use of mercury is prohibited in many regions, e.g., in Alaska, and the ban is rigidly en-forced. Amalgamation is only justifiable as long as the process exploits all technical options for recovering the mercury and amalgam while minimizing mercury losses. For example, the mercury should always be covered with water. The vapor pressure of mercury cannot overcome a water barrier, and, hence, the mercury does not evaporate. Chapter 6 offers additional information on the environmental hazards of amalgamation.

In principle, all free, clean gold will amalgamate. Frequently, however, the feed ore contains certain accompanying minerals and/or impurities with negative effects on the amalgamating process. Some such problems are outlined below:

- Soluble lead minerals, arsenic in arsenopyrite, sulfides of arsenic, etc., antimony and bismuth either react with the mercury, form amalgam or chemical films, or dissolve the mercury or precious-metal amalgam out of the exposed surface on the amalgamating device, thus leading to significant loss of precious metal and mercury. By contrast, fresh pyrite and chalcopyrite have no effect on amalgamation.
- Baryte, talcum, steatite and cohesive hydrated silicates of magnesium and aluminum also may disrupt the process or increase the loss rate.
- Oil, grease and lubricants are extremely troublesome, because they attach themselves to the mercury and tend to attract sulfides. Consequently, the mercury becomes surrounded by a solid film of fine particles. Moreover, the presence of oil, grease or lubricant causes immediate flotation of gold, which then is carried off without having come in contact with mercury. Such factors naturally lower the precious-metal recovery rate of the amalgamation process. Countermeasures include the addition of surface-active agents, e.g., a strong detergent or the sap/juice of a plant such as sisal, the leaves of which are often used for such purposes in Colombia - the idea being to saponify the oil and grease.
- Acid mine drainage as head water also has detrimental effects on amalgamation. The presence of lime neutralizes such effects. Accordingly, amalgamation should be limited to ore from deposits with little or no sulfidic material.

As mentioned above, mercury should only be employed in a closed-cycle process. However, impurities such as the above can lead to contamination of the mercury and amalgam. Dirty mercury is much less reactive than clean mercury. While the latter forms nearly spherical, metallic bright, almost ideal globules, soiled mercury is characterized by a lack of gloss, deformed globules and the tendency of the latter to slightly adhere to a smooth, inclined surface such that they appear to have "tails".

There are a number of ways to clean and reactivate dirty mercury:

- by putting it through a very fine screen (200 mesh)
- by washing it with wood ash and water, whereas potassium carbonate helps saponify the contaminants
- washing in water that contains a surface-active agent or a special plantjuice solution, either of which has the capacity to saponify and dissolve grease and greasy substances
- washing with a reagent such as ammonia, ammonium chloride, cyanide, hydrochloric acid, nitric acid, etc.
- distillation in a retort to leave behind nonvolatile contaminants
- adding sodium amalgam, which, upon contact with water, converts to NaOH, which in turn strips contaminants from the surface of mercury.

Approximately 1 part sodium to 2000 parts mercury is considered a good concentration. Na-amalgam can be obtained by electrolysis as follows: fill mercury into a receptacle and apply a carbon cathode encased in an insulating tube of glass or plastic. Pour a 10-15 % salt solution over the mercury and connect it to a carbon anode; apply direct current from an automobile battery to make the Na^+ ions discharge onto the Hg surface, resulting in the amalgamation of metallic sodium. The concentration should suffice after 10 to 15 min. Store the Na-amalgam away from air, e.g., under petroleum.

3.5.1 Amalgamating copper plates

Amalgamating plates are used for recovering fine stamped or milled gold ore and for recovering mercury and amalgam following amalgamation in a stamp mill or edge mill. Sorting on amalgamating copper plates consists of running a pulp of liberated, ground feed over slightly sloping plates of copper or Muntz metal (60 % Cu, 40 % Zn) that have an electrolytically applied layer of silver and a mercury coating. The slope of the plates should be such that the mineral particles do not sediment (depends on the density of the heaviest accompanying minerals). The gold migrates to the bottom of the pulp, where it comes in contact with mercury and amalgamates. The plates have to be cleaned up several times a day and redressed for return to service. Such preparations are relatively time-consuming. After taking off all amalgam, the plates are scrubbed with fine sand, then degreased with a 1 % cyanide solution, scrubbed again, scoured with an ammonium-chloride solution to remove base metal oxides and then, finally, coated with mercury. Sooner or later, all plates will need a new coat of silver. Either the plates can be coated with a thin layer of silver nitrate, or silver foil can be alloyed with the mercury. Some local approaches have evolved in small-scale mining for preparatory purposes and for cleaning the plates, e.g., scrubbing the plates with urine, sisal sap or washing powder.

As far as the actual operation of amalgamating plates is concerned, close attention must be given to an optimal angle of inclination. The pulp should flow over the plates in a succession of small ripples, and each successive plate should drop slightly. All plates must be completely flat. The feed must be carefully sized, since coarse material could lead to abrasion of the amalgam and mercury as it passes over the plates, thus leading to loss of both metals.

Photo 18: **Amalgamating copper plates** coated with silver amalgam for amalgamating fine gold out of passing pulp; Nambija, Ecuador

3.5.2 Amalgamating barrels

Amalgamating barrels are used for amalgamating preconcentrates. The main advantage of barrel amalgamation is that the feed and the mercury are contained within a closed reactor, thus precluding the loss of substantial amounts of metallic mercury. Amalgamating drums work on the same principle as a ball mill. The feed consists of a high-grade preconcentrate. The barrel is charged with feed ore, water, roughly three times as much mercury as the anticipated quantity of recovered gold, and a load of grinding media. As the barrel rotates, its contents become intimately mixed, i.e., the particles of gold come in contact with the mercury and amalgamate. The grinding media press gold into the mercury, so that even minute particles which otherwise may have escaped amalgamation due to the surface tension of mercury can also be recovered. At the end of the rotating time, tapping and other means of vibration help effect gravity separation, with the mixture of amalgam and mercury gathering at the bottom for recovery after removal of the tailings. The reactor itself is normally cylindrical with a horizontal axis and therefore resembles a rod or ball mill. Frequently, however, small-scale miners simply load boulders into a cement mixer to arrive at an "amalgamating barrel". The American Berdan pan, for example, is a slow-running single-ball rolling mill with a circular-oblique trajectory. As the bowl rotates, the ball strives to remain at the bottommost point and effectively floats upon the mercury. All amalgamating barrels, no matter what the type, have the common advantage of mitigating, if not totally precluding, the loss of metallic (floured) mercury during amalgamation, thus protecting the environment. During operation, a suitable reagent can be added to optimize the surface properties of the mercury with respect to the feed charge in the closed barrel. Suitable reagents for improving the surface activity of mercury include caustic and slaked lime, sodium hydroxide, sodium amalgam, ammonium chloride, cyanides, nitric acid, surface-active agents and similar substances. The basic operating parameters for amalgamating barrels are the type and quantity of feed and grinding media and the speed of the barrel. Particularly large grinding media are needed for simultaneous milling and amalgamation of unliberated preconcentrates and high-grade ores. The speed of rotation for pure amalgamation of liberated free gold is only about half that of an equal-size mill. The low speed extensively prevents the formation of floured mercury. However, if milling and amalgamation are to be effected simultaneously, a compromise will have to be found between loss of performance due to grinding-media slippage and certain drawbacks affecting the results of amalgamation.

Amalgamating barrels are relatively well-suited for local manufacture. Metalworking shops can produce simple types from pipe sections, sheet metal, etc., and commercial-type bearings can be employed. Many simple amalgamating barrels now in use are powered by small waterwheels.

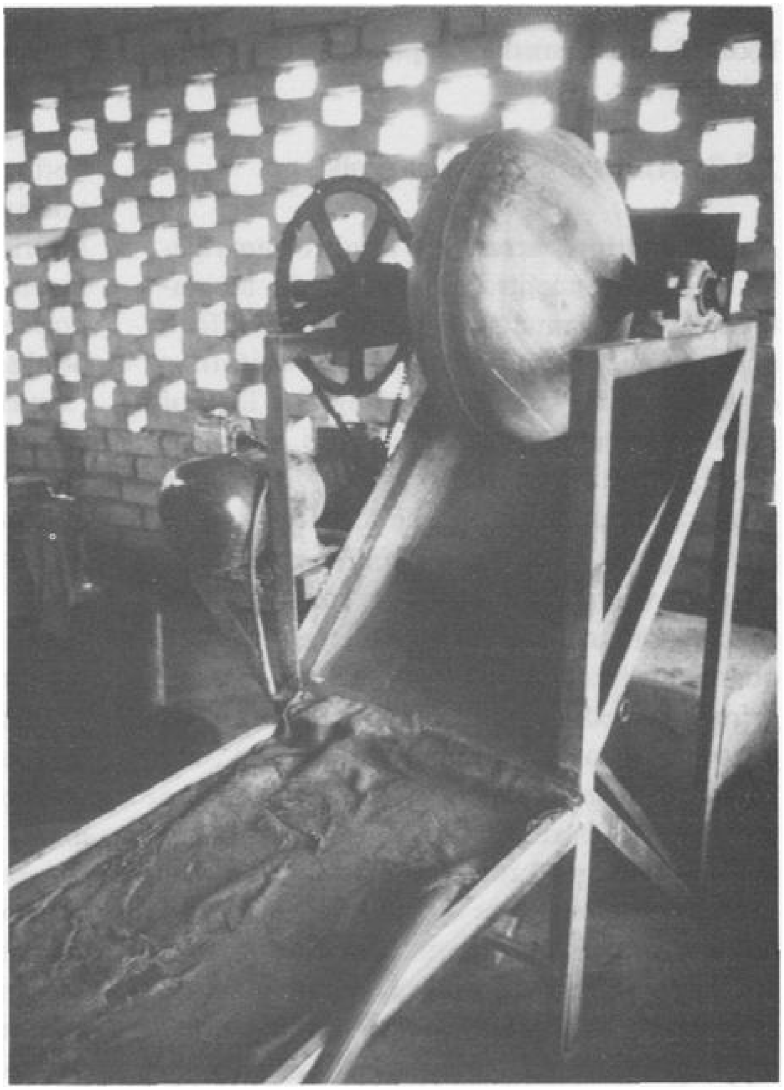


Photo 19:
Barrel amalgamator for shaking-table concentrates mounted above a sluice for washing out the amalgam and a covered set of amalgamating plates for recovering the lost amalgam; Shamva Mining Centre, Simbabwe

Photo 20:
A simple, waterwheel-driven **amalgamating barrel** with iron bars for planking the gold into the mercury; Mina El Canada, Nariño, Colombia

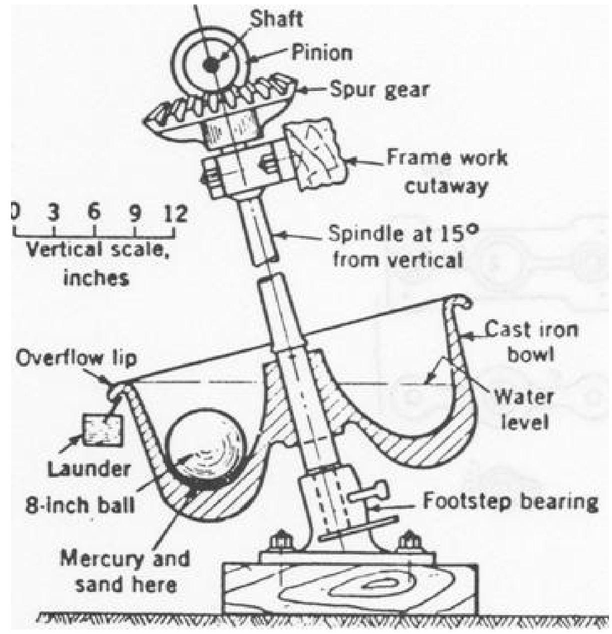


Fig. 36: **Berdan pan**, a single-ball rolling mill for use as an amalgamator [Bernewitz]

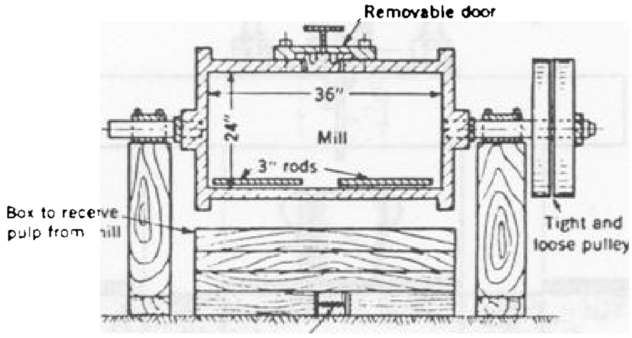


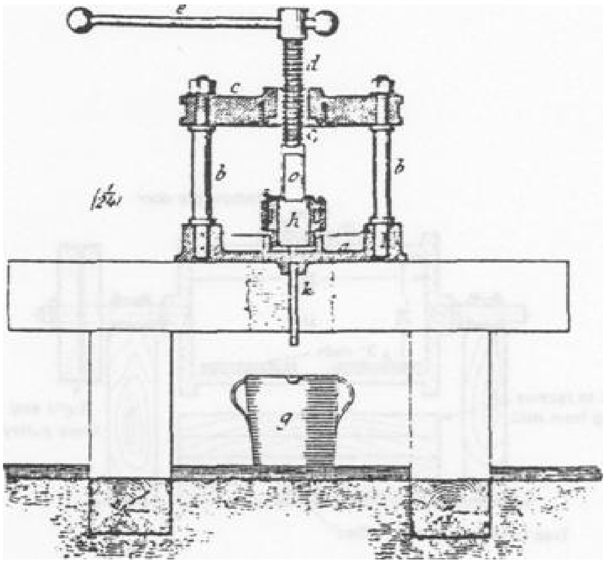
Fig. 37: A rod mill serving as an **amalgamating barrel** [Bernewitz]

3.5.3 Pocket Amalgamators (Jackpots)

Gold ore can also be amalgamated with the aid of pocket amalgamators, or mercury-filled traps. Also referred to as jackpots, such implements operate much the same as a round thickener. The pulp enters the deep, conical vessel through a central feed pipe extending to just above the top of the mercury. The pulp is forced to change directions, causing the heavy particles of gold to sediment, be wetted by the mercury and, hence, amalgamate. Due to its high specific gravity, the amalgam sinks to the bottom of the vessel, so that pure liquid mercury remains at the pulp/mercury interface for further amalgamating action.

3.5.4 Amalgam presses

Amalgam presses are used for separating amalgam from the mixtures of amalgam and mercury obtained from amalgamation processes, the idea being to reduce the feed to the distillation retort. Their principle of operation is based on the difference in viscosity between pure metallic mercury and various gold-amalgam alloys (Au_3Hg and Au_2Hg) and the amalgams of other metals. Amalgams are of a high viscosity and pasty consistency. The mixture is pressed through chamois-dressed leather, linen or some other tightly woven fabric. The amalgam remains trapped in the press, while the pure mercury containing only trace amounts of gold (< 0,2 % at room temperature) passes through the filter fabric and collects on the other side. In small-scale mining, amalgam presses are frequently dispensed with in favor of a simple wet cloth wrapped around the amalgam. Wringing the cloth over a gold pan forces out the mercury, which drips into the pan. The more viscous amalgam remains in the cloth. Such manual means of separation, however, are ill-advised, because mercury is very toxic, and the low applied pressure yields poor separation efficiency. Mercury separates from amalgam more readily if the mixture is heated in hot water prior to pressing. The elevated temperature lowers the viscosities of the individual constituents, yielding better separation of the components through a cloth or press. Amalgam presses, at least the more simple models, can be locally manufactured at modest cost. Any good metalworking shop can make amalgam presses employing commercial-type threaded couplings, etc.



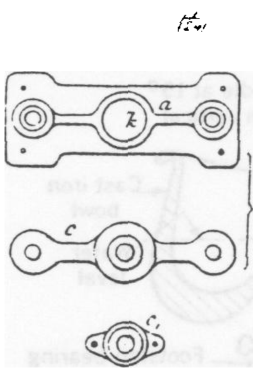


Figure 38: An amalgam press [Rittinger]

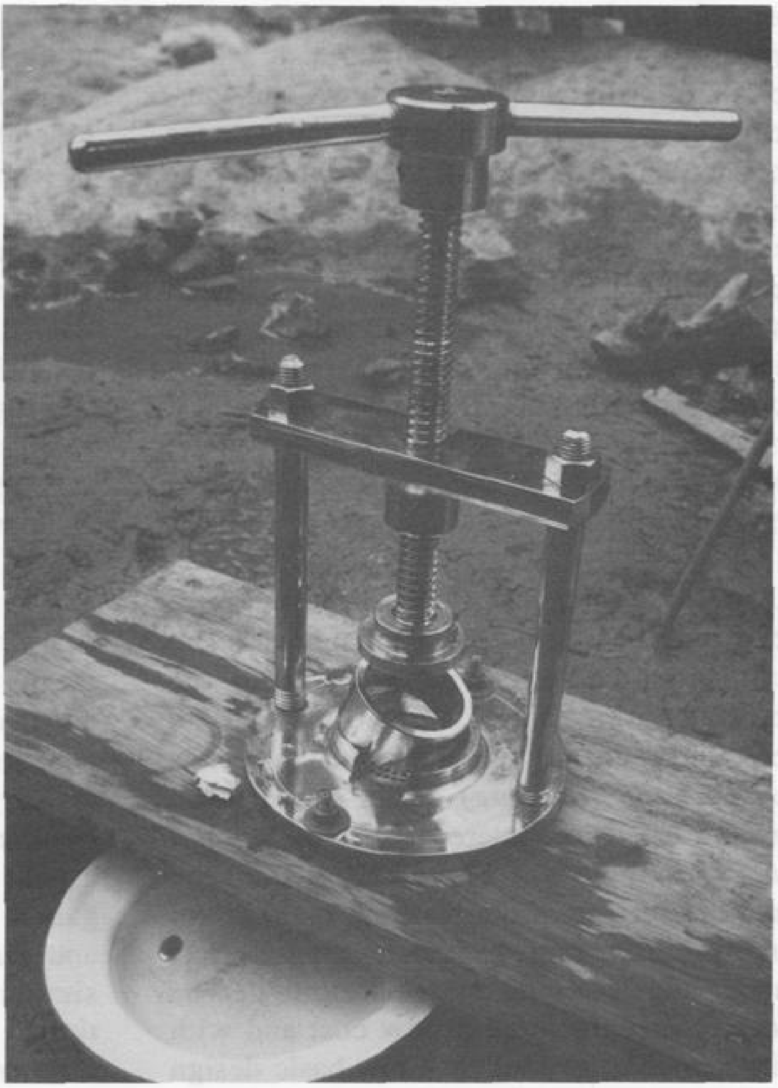


Photo 21:
A nickel-plated **amalgam press** on a wooden stand over a mercury drip pan; local manufacture in Colombia; with floating cylinder for fixing the filter fabric in position above the perforations; cutout in the cylinder mounting ring to facilitate removal of fabric and amalgam

3.5.5 Distillation retorts

Retorts are used for separating amalgam into mercury and the precious metal (gold and/or silver). The separation of amalgam into mercury vapor and gold is the most hazardous part of the entire amalgamation process. It endangers both the miners' health and the environment. Care must be taken to recover as much of the mercury as possible in the distillation of amalgam.

A retort is a crucible-like vessel with an opening/closing mechanism, an outlet pipe at the top of the vessel, i.e., in its cover, and a downward-pointing, pipe-like neck serving as a condenser. The most elementary type of condenser consists of a straight tube wrapped in wet rags. More elaborate constructions include a water-filled jacket or even a counterflow cooler employing water as coolant in an open or closed cycle. The amalgam to be separated is placed in the crucible, which should be lined with paper, the ash of which will form a nonadherent intermediate layer between the gold and the wall of the retort. Better results are achieved by applying a thin coat of graphite, lime, chalk or talcum to the inside of the crucible before it is loaded with amalgam. This keeps the gold from sticking to the bottom of the retort after distillation. (No greasy matter should ever be used, because it would evaporate along with the mercury, inactivating its surface for subsequent use in the process.) Then, the crucible is closed and heated to raise the temperature of the gold-mercury alloy to above 400°C, at which point the amalgam breaks down and mercury evaporates. As it passes through the condenser, the mercury vapor precipitates onto the tube and trickles into a water-filled receiving vessel. The water prevents further evaporation. Retorts must always be heated such that heat is applied to all sides at once, including the outlet riser. Otherwise, some of the mercury could condense before it reaches the condenser, in which case it would run back into the crucible and have to be re-vaporized.

In any case, once the heat has been removed, care must be taken to ensure that subsequent cooling does not draw water into the crucible. Were that to happen, the still-hot crucible could explode due to instantaneous evaporation (flashing) of the water. That hazard can be countered either by using a wet sack to "connect" the end of the neck and the surface of the water in the receiver or by terminating the condenser tube just above the receiving vessel. Retorts can be locally manufactured at low cost and with little trouble, as long as a few basic design details are adhered to: For one thing, the condensing area for the mercury should be kept as small as possible in order to minimize loss of mercury due to cohesion of fine droplets of mercury on the inside of the retort. For example, the cooling tube should be of small diameter and made of iron or steel, since brass would amalgamate with the mercury. The inside of the tube must be very smooth, and all seams should show upward so as not to hinder the trickle of mercury. Despite such precautions, as much as 2 grams of mercury may remain behind in the retort, which has to be washed out to recover it. Consequently, it is a good idea to collect a sizable quantity of amalgam for distillation in a single process. The vessel closure is another crucial detail. No matter what kind of retort is used, special attention must be given to gastight flanges and fits. If a retort is found to have developed a leak, a mixture of moist clay and ash can be used to effect a good seal between the cover and the crucible (prior to distilling!). The clay must be free of coarse grain. The simplest retorts, made of commercial-type pipe sockets, threaded couplings and pipe sections, have certain drawbacks with regard to user-friendliness. In addition, threaded closures are liable to develop leaks due to periodic heating and cooling. The best seal for a retort is provided by a lid with a turned sealing lip and a wedge-type closing mechanism that presses the lid up against the rim of the crucible.

In some cases, techniques other than retorting are used for recovering mercury from amalgam by distillation. However, they are consistently characterized by major loss of mercury and should not be practiced:
Mercury can be separated from amalgam in a gold pan made of metal. First, the amalgam is filled into a small tin can (cf. fig. 39) and placed at the center of a gold pan or similar vessel. Then, a larger tin can is turned upside down and placed over the smaller one at the center of the dish. The remaining dish volume is then filled with a mixture of damp sand and ash. Now, if heat is applied to the bottom of the small can containing the amalgam, the mercury will evaporate, and some of it will condense onto the inside of the larger can and trickle to the bottom of the dish, where part of it will again evaporate, eventually getting into the sand, where (most of) it again condenses. At the end of the heating phase, the sand is removed from the pan and washed, at which point most of the condensed droplets of mercury reunite. An even more rudimentary approach to the recovery of mercury from amalgam is to place a fresh banana leaf over an open evaporating tray. Mercury condenses on the leaf and migrates to the edges. Orange peels, brassica leaves, halved potatoes, etc. serve the same purpose in more or less the same manner.

Photo 22: A technically mature type of **retort** made in Colombia; in operation on-the-spot, with a kerosene **blowtorch** serving as heat source and a cup of water for catching the precipitated mercury; the lid of the pot is fixed in position by a wedge running in a groove; such retorts cost approximately US$ 65

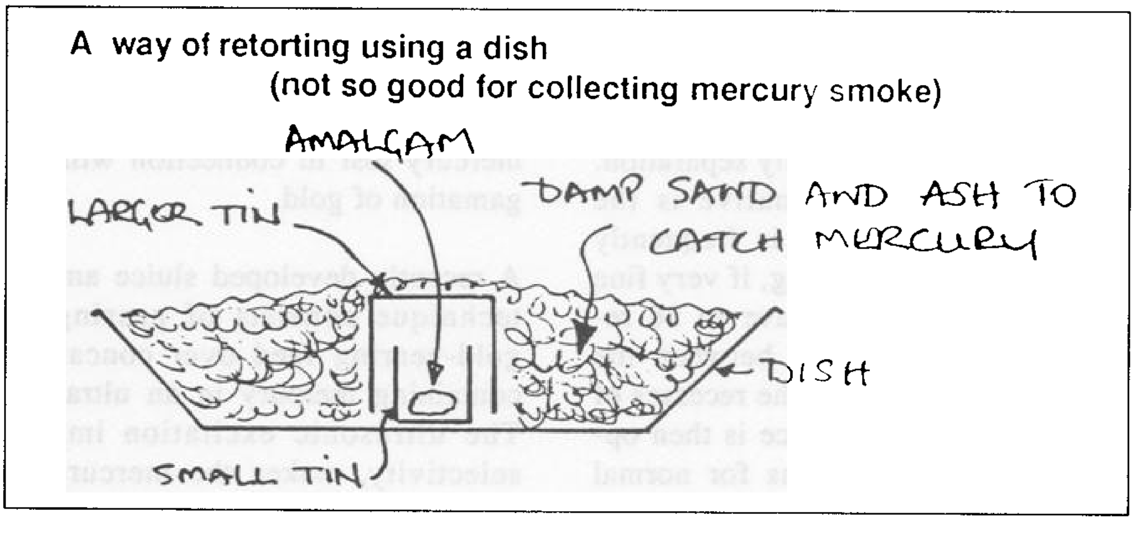


Figure 39: The can-and-dish amalgam retorting principle [Blower]

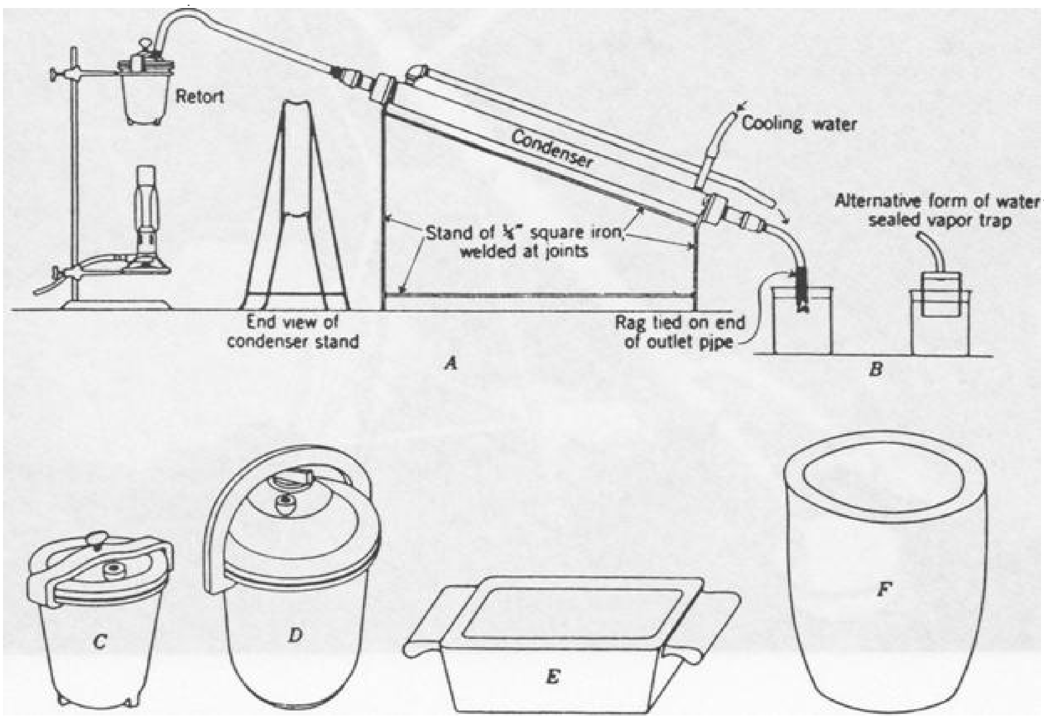


Figure 40: A selection of **retorting vessels** (bottom); a typical distilling arrangement [Stout]

3.5.6 Amalgamation in sluices

Apart from the use of an amalgamating apparatus designed exclusively for the process, amalgamation can also be accomplished in some devices that are normally used for wet-mechanical gravity separation. The most widespread alternative is the sluice. Sluice amalgamation is frequently practiced in placer gold mining, if very fine gold particles (flour gold) have to be recovered. Mercury is placed between the riffles of riffled sluices or in the recesses of rubber-lined sluices. The sluice is then operated in the same manner as for normal gravity separation. As the pulp flows by, the heavier gold migrates down into the space between the riffles and is amalgamated by the mercury. The advantage of this process is that fine gold becomes amalgamated instead of being washed out of the sluice by momentary flow peaks. The drawback is that the process can produce relatively large amounts of floured mercury, particularly if the feed is rather coarse, which is likely to escape to the environment. Losses of metallic mercury - due mainly to sluice amalgamation - account for nearly half of all mercury lost in connection with the amalgamation of gold.

A recently developed sluice amalgamation technique consists of passing the fine-gold-bearing feed over concave grooves containing mercury in an ultrasonic field. The ultrasonic excitation improves the selectivity, makes the mercury more reactive, and raises the process temperature. The aim is to induce microrange evaporation of the mercury and its subsequent selective precipitation onto the particles of gold, thus facilitating amalgamation. Cooling is provided by the normal flow of water. It is somewhat difficult to find the proper slope, i.e., one that will keep mercury from being washed out while still ensuring intimate contact to promote amalgamation.

With the exception of the ultrasonic excitation feature, amalgamating sluices are well-suited for local manufacture. However, only nonamalgamable materials must be used for any sluice intended for contact with mercury. Sluices made of aluminum or galvanized sheet metal are wholely unsuitable for such purposes.

3.5.7 Amalgamation in centrifugal concentrators

In some cases, gold ore is amalgamated in centrifugal concentrators, most notably Knudson centrifuges. The idea is to place mercury in one or more of the inner annular spaces in order to effect the amalgamation of gold during the sorting process by reason of its contact with mercury and the influence of centrifugal force. Due to the high flow velocities occurring in the centrifuge, accordingly high loss of finely dispersed mercury must be anticipated.

3.6 Leaching Processes

Chemical dissolution, transport and precipitation phenomena provide the basis for separation by leaching. In certain Eh-pH environments, certain minerals are dissolved by certain acids, bases or solutions. The presence of bacteria may have catalytic effects. The dissolved metals are precipitated and concentrated at a different location. Leaching is accomplished in vats, on heaps or in situ.

Most relevant for the gold-mining sector are the leaching processes based on dilute cyanide solution and thiourea.

3.6.1 Cyanidation

Cyanidation is the most important leaching process for gold ore. Cyanide leaching can be used for dressing a broad range of gold ores, e.g., ore with small particles of free gold (down to submicroscopic-range), disperse gold, e.g., in vulcanites or carbonates), gold from soluble sulfides, and gold attached to sulfides. Refractory ores, e.g., with gold-bearing pyrites, require supplementary preparation, e.g., roasting, prior to leaching.

Leaching is most suitable for ore containing fine particles of gold with large surface area. If the feed contains coarser gold particle collectives, the leaching rate drops. Consequently, such fractions are normally extracted via upstream gravity separation, with only the tailings undergoing leaching to recover the fines fraction.

Ores displaying different types of intergrowth can be selectively comminuted in order to liberate the gold without overmilling the ore. Ore with coarse gold intergrowth in quartz and fine intergrowth in sulfides and their intergranular voids is frequently encountered. The sulfides can be gravitatively separated from the main milling circuit, e.g., by a jig, and selectively fine-ground in a separate milling cycle.
Agitation leaching is primarily for milled or finely comminuted ore, while vat leaching is reserved more for ground ore, and heap leaching for rough-crushed run-of-mine ore. Heap leaching is inexpensive but has relatively low recovery rates (about 50 %) and is therefore hardly recommended, except for low-grade ore. Rough-crushed heap ore is rarely leachable. Gold particles surrounded by, say, quartz are also unleachable without enhanced exposure by grinding.

All things considered, the best leaching options for small-scale mining would appear to be percolation leaching or vat leaching - and possibly heap leaching for easily leachable ore - because they require less apparatus for leaching, absorbing and extracting the gold from clear slurries.

Differentiation is made between the following processes and applications:

Agitation leaching

The cyanide leaching process exploits the capacity of gold to enter into soluble complexes with cyanide. This takes place according to the following steps: The raw ore is reduced in size to < 0,1 mm, the low pH of the slurry is raised to approximately 10 - 11.5 by adding milk of lime (CaO), and the density of the pulp is adjusted to 40 - 50 % solids. Enough sodium cyanide is added to an agitated vessel full of slurry to yield an NaCN concentration of 100 ppm. All the while, the solids content must be kept in suspension by paddles or blown-in compressed air (in pachucas). With the pH at 10 - 11.5, the dissociation equilibrium shifts in favor of the cyanide ion, and the following chemical reaction takes place:

$$2\,Au + 4\,NaCN + 1/2\,O_2 + H_2O$$
$$\rightarrow 2\,NaAu\,(CN)_2 + 2\,NaOH$$

The leaching process takes between 12 and 24 hours, after which the solution contains gold in a concentration of 4 - 6 ppm.

In a well-equipped agitation leaching plant, the leaching process can be commenced with at the milling stage by wet milling in a cyanide solution. This offers the advantage of exposing absolutely fresh mineral surfaces to the leach solution.

High, cylindrical leaching tanks, e.g., pachucas, can be made by stacking concrete rings, which, if necessary, can be lined with synthetic resin to allow acid leaching, e.g., with thiourea (cf. ch. 3.6.2.).

Vat leaching

In vat leaching, rough-crushed or agglomerated ore is loaded into a vat and flooded with cyanide solution set for a certain pH. The chemical processes involved are similar to those described above. At the end of the exposure time, barren solution is drawn off via a filter pipe.

Percolation leaching

The tailings from gravity concentration processes are frequently subjected to percolation leaching. Large open tanks with contents possibly exceeding 100 m^3 are equipped with a bottom drain, possibly with a double floor and a layer of filter fabric or gravel. Cyanide solution is introduced into a tankful of ore and percolates down through the load at a rate of about 8 - 10 cm/h. The feed material should be deslimed in order to keep the percolation rate at least above 2 cm/h, since less would be very unfavorable. Eventually, the leaching solution collects at the bottom of the tank, and air (which supplies the oxygen needed for oxidation) surrounds the product. This process is repeated daily for a few days to over a month.

Percolation leaching tanks should not be too deep. Excessively deep tanks make it too hard for the air to surround the product as the leaching solution recedes, and the ore may not receive an adequate supply of oxygen.

The leaching solution can be applied in different concentrations, i.e., highly concentrated at first, then more dilute and, finally, as a washing solution.

Heap leaching

Ore crushed for adequate leachability is heaped in a leachtight basin lined with clay, asphalt and/or a tarpaulin and sprinkled with leaching solution. The process yields clear, gold-bearing cyanide complex solutions. Depending on the nature of the gold-cyanide complex solution, further processing can be according to several different methods.

For clear solutions:

- According to the Merrill Crowe process, the solution is separated from its undissolved constituents in fabric-lined vacuum filters or clay-lined filtering candles and then completely deaerated in vacuum cylinders to preclude oxidation and resultant high loss of gold in the following steps. The oxygen concentration is lowered to approximately 0.5 mg/l. Zinc dust and lead nitrate solution are added to induce formation of local elements, and the process of gold cementation proceeds as follows:

$$2\ NaAu\ (CN)_2 + 2\ Zn$$
$$\longrightarrow 2\ Au + Na_2\ [Zn\ (CN)_4].$$

Gold and the excess zinc are filtered off in a filter press, while the cyanide solution recirculates. The solids are treated with dilute sulfuric acid to wash out the excess zinc. The gold precipitate is then roasted at approximately 800°C and subsequently melted down at 1200°C with borax and siliceous fluxing agents.

A much simpler variant is the zinc precipitation approach, which, however, involves higher losses. The leaching solution is forced up through a calotte of sieves loaded with loose, fine zinc shavings with a large specific surface area. The gold precipitates onto the zinc,

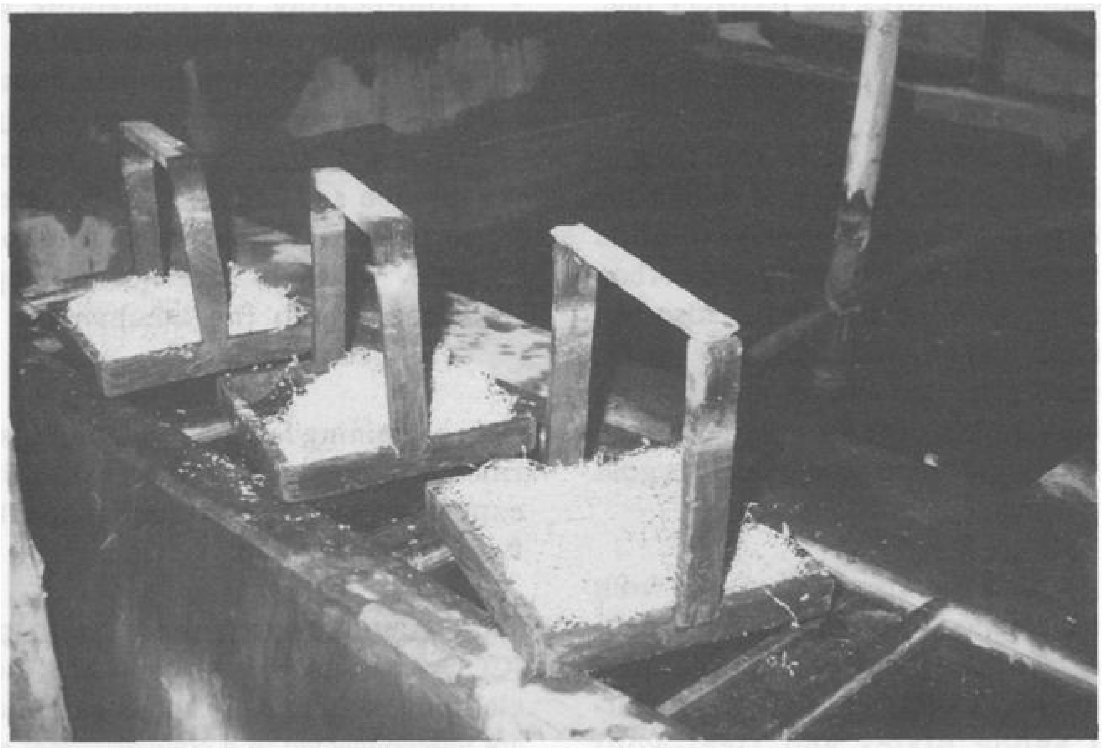


Photo 23: A bank of gold **precipitating vats** for depositing gold on zinc shavings; screens removed to show load of zinc shavings for use in a simple percolation leaching process; the pregnant solution passes through all four precipitation stages in sequence; the solution collecting tank is shown in the background; Mina Los Guavos, Nariño, Colombia

as evidenced by blackening of the zinc. Once all of the shavings have accumulated a film of gold, they are melted down and the gold recovered (possibly being sold as gold-bearing zinc to a professional buyer or service organization).

The zinc wool used for precipitation should be about 0.02 mm thick. This gives it between 10 m and 20 m^2 specific surface area per kg on a volume of roughly 10 l. The optimal measure is regarded as approximately 30 l zinc wool per m^3 solution/24 h. Solutions which have already undergone precipitation are emptied into the reservoir from a considerable height in order to entrain oxygen on the way down.

Recent patent literature describes the cementation of gold from slightly turbid cyanide solutions in vibrating reactors filled with zinc granules. The agitation achieved by vibration accelerates cementation, improves the recovery rate, lowers zinc consumption and substantially simplifies the process sequence.

The third alternative is the carbon-in-column process (CIC), in which the clear gold-cyanide complex solution flows upward through activated-carbon cylinders, with the gold being adsorbed onto the carbon. Completely gold-saturated activated carbon may contain as much as 20 - 30 kg gold/t and can either be incinerated or marketed as gold concentrate.

Finally, gold can also be extracted from cyanide solution by electrolysis. In small-scale mining, locally manufactured electrolytical cells consist of special-steel anode plates and steel-wool cathodes. A continuous flow of pregnant solution is fed through the cells under a 12 V current with an intensity of some 6O A. The steel wool attracts the gold, which later can be shaken off and collected, while the steel wool can be used again.

For pulps:

- The carbon-in-pulp process (CIP) consists of adding granulated active carbon to the pulp. The carbon adsorbs the precious-metal cyanocomplexes, which are subsequently separated by mechanical means (screening) and washed out using a strong alkaline sodium cyanide solution, possibly under pressure and at an elevated temperature. Then the gold, together with any silver or copper present, is electrolytically precipitated onto steel-wool electrodes. After the process, the activated carbon requires cost-intensive regeneration. Alternatively, the gold-laden activated carbon can be incinerated.

- Gold-bearing ore containing organic substances with the potential to absorb gold out of solution (semi-activates), are treated according to the carbon-in-leach process (CIL), in which the leaching solution and activated carbon are added to the ore. The active coal absorbs the gold, which is subsequently separated according to the mechanical means described above.

Pulp containing few suspended solids can be filtered through a bed of gravel as the least expensive and least equipment-intensive form of clarification.

The countercurrent decantation process (CCD) can also be employed as a means of clarifying solutions. A number of thickeners are interconnected in a countercurrent arrangement such that the thickened pulp is returned to the preceding thickener, and the overflow is forwarded to the next thickener.

Cyanidation involves substantial expenses for reagents, particularly for loss of cyanide due to oxidation, the release of HCN and various reactions with the ore's companion substances. Active-carbon abrasion is a problem in the CIP leaching process.

Cyanide solutions tend to decompose at elevated temperatures, and low temperatures retard their reaction rates. From an economic standpoint, the optimal solution temperature is about 20°C. In a cool climate, the solution therefore requires some form of heating. In any case, the tanks and vessels containing leaching solution should be kept covered, because UV radiation has a destructive effect on cyanide solutions. Pregnant cyanide solutions should always be further processed without delay. Otherwise, colloids (chiefly nitrates of aluminum, iron or magnesium) may settle out of the clarified solution and hinder precipitation onto the zinc shavings or dust. Also, the gradual absorption of atmospheric CO_2 can lead to the precipitation of calcium carbonates in high-calcium solutions.

Leached ore can be emptied out of vat-leaching plants and tanks via floor drains or simply by washing with large amounts of water. The tailings should be stockpiled or stored in a tailings pond. Excessive cyanide contents are gradually broken down by ultraviolet radiation.

3.6.2 Thiourea leaching

In addition to cyanidation, gold ore can also be subjected to thiourea leaching - a practice which appears to be gathering momentum. In contrast to cyanidation, thiourea leaching takes place in the acidic pH range, and the complexing agent (thiourea) is more than 10 times stronger, i.e., more concentrated than a cyanide solution. The thiourea leaching process is advantageous in connection with sulfidic/sulfide-rich and partially weathered ores, which show a tendency toward acid reaction, anyway. Consequently, it takes little effort to generate a suitable pH environment. On the other hand, the acidic conditions demand the use of reactors and other plant components that are sufficiently resistant to acids, and that can make the equipment quite expensive. Moreover, the often tight thiourea market can be a killing factor for the process in developing countries. The thiourea leaching process is analogous to the cyanide leaching process, as described in chapter 3.6.1.

3.7 Concentrate Purifying Processes

The products of gold sorting, concentrating and dressing processes are not always directly marketable. They often contain contaminants, companion minerals, etc. Consequently, the various steps of ore dressing are frequently followed by some means of purifying the concentrates.

3.7.1 Smelt separation

To separate gold from heavy minerals or to clean the gold obtained from amalgamating/leaching operations, the concentrate is placed in a crucible with borax and other fluxing agents and heated to 1200°C. The oxidic minerals such as limonite, ilmenite, etc. melt, producing a solid-liquid system in which the liquid gold collects at the bottom of the crucible below the slag. Pure gold melts at 1063°C. The following substances can be used as slag formers and fluxes. Quartz, glass, borax, microcosmic salt, soda, sodium bicarbonate, potash, ammonium chloride, ammonium carbonate, tartar, potassium cyanide, saltpeter, fluorite, and cryolite. The choice of reagent for inducing slag formation and improving the fusibility depends on the mineralogical composition of the concentrate. Even the choice of crucible (fireclay, graphite or oxide-ceramic) depends on the nature of the concentrate composition.

The smelt separation technique cannot be applied to platinum group metals (PGMs), because their melting points are too high (platinum 1769°C, rhodium 1966°C, paladium 1550°C, osmium 2700°C and iridium 2454°C).

3.7.2 Roasting and winnowing

One concentrate purification technique that is widely employed among gold miners in Africa is a combination of roasting and classifying. The concentrate is heated to a temperature of at least 600°C in order to disintegrate certain mineral components in the heavy-mineral sands, e.g., hematite. After cooling, those constituents can be blown out of the concentrate.

3.7.3 Selective comminution

Small-scale gold miners in New Guinea use a method of concentrate purification that exploits the malleability of gold. The concentrates are first classified into as tight a range of fractions as possible with the aid of a set of test screens. Then, each individual particle-size fraction is worked with a hammer. The pounding forges the gold particles into flakes of larger size, while the barren particles break down into smaller pieces due to their brittleness. Renewed screening on the same screen separates the gold from the selectively comminuted concentrate. The process is repeated for all size fractions.

3.7.4 Magnetic separation

Magnetic separation is a method used for removing magnetic heavy-mineral particles from precious-metal concentrates. Strong permanent magnets like those used in loudspeakers serve well. They are installed in plastic cans and calibrated with cardboard, paper, wood or plastic spacers. Separation is accomplished by passing a strong magnet over a thin layer of concentrate lying on a nonmagnetic deck. The magnetic separator is lowered onto the deck, and the magnets attract any magnetic particles. Then, the magnetic separator is lowered onto a different table and the magnets removed from the plastic cans, so that the magnetic mineral particles fall off. Several repetitions can significantly improve the magnetic separation efficiency.

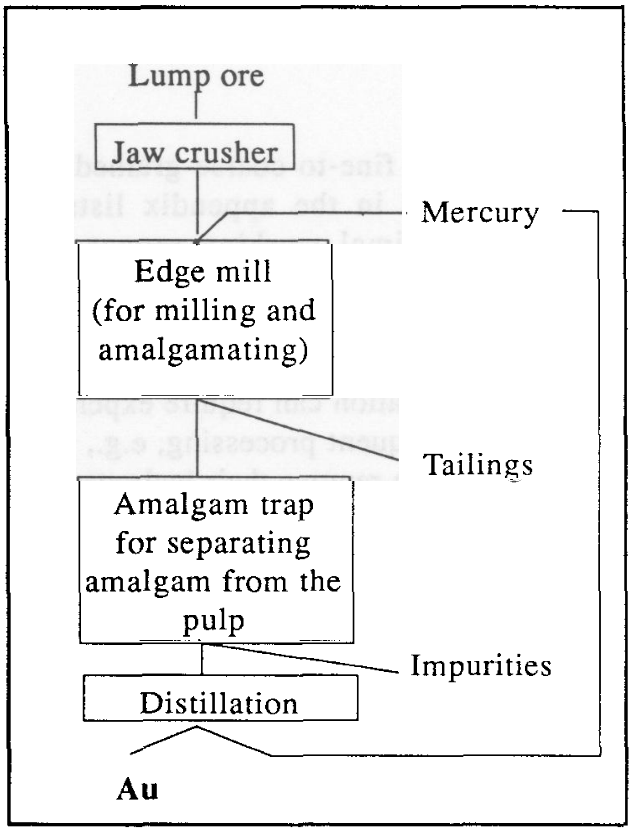
4. GOLD ORE DRESSING PROCESSES

Depending on the nature of the raw ore and the company's investive potential with regard to processing equipment, the sequence of operations comprising a number of the dressing and preparation steps discussed in chapter 3 will vary, but should always culminate in a method of generating marketable gold concentrates. Basically, different dressing techniques are used for loose sediments and primary ores.

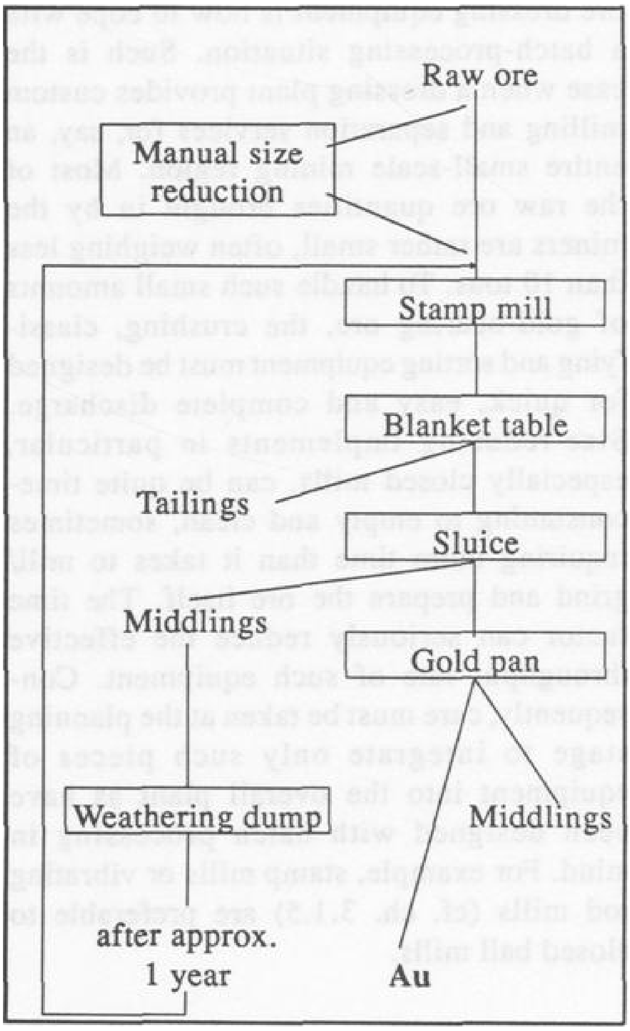
Loose sediment dressing processes require no size-reduction stage, because the raw material is already liberated. Initial concentration is often achieved by screening off the coarse fraction. Dressing processes for gold-bearing loose sediment are practically always designed for higher throughputs, because the raw ore content is normally substantially lower than that of primary ore deposits. Moreover, secondary treatment of middlings is only necessary if inadvertent discharge, e.g., in certain particle-size fractions, detracts from the selectivity of the sorting process. Middlings are never milled as part of a placer gold dressing process.

In contrast to placer gold processing, lode ore requires size reduction to ensure the liberation of the valuable material for dresssing. Accordingly, size reduction is the first step in the processing of primary ore and consolidated sediments. Classifying stages follow, depending on the type of sorting. With regard to sorting, it is difficult to generalize in connection with host ore mining, because all manners of sorting, i.e., gravity separation, amalgamation and leaching techniques are employed under certain circumstances, often in combination. As a rule, leaching is reserved for fine and ultrafine particle collectives, amalgamation for fine-to-medium fractions, and gravity separation for fine-to-coarse-grained material. Table 2 in the appendix lists the respective optimal working ranges. Additionally, some lode-ore mining operations employ a sulfide flotation process to obtain gold-rich preconcentrates. However, the products of flotation can require expensive, elaborate subsequent processing, e.g., pressure leaching, to remove their hydrocarbons and various other problematic minerals. Consequently, such processes are hardly suitable for small-scale mining and therefore not gone into in this publication. Some typical small-scale gold ore dressing procedures are illuminated by way of flowsheets on the following pages.

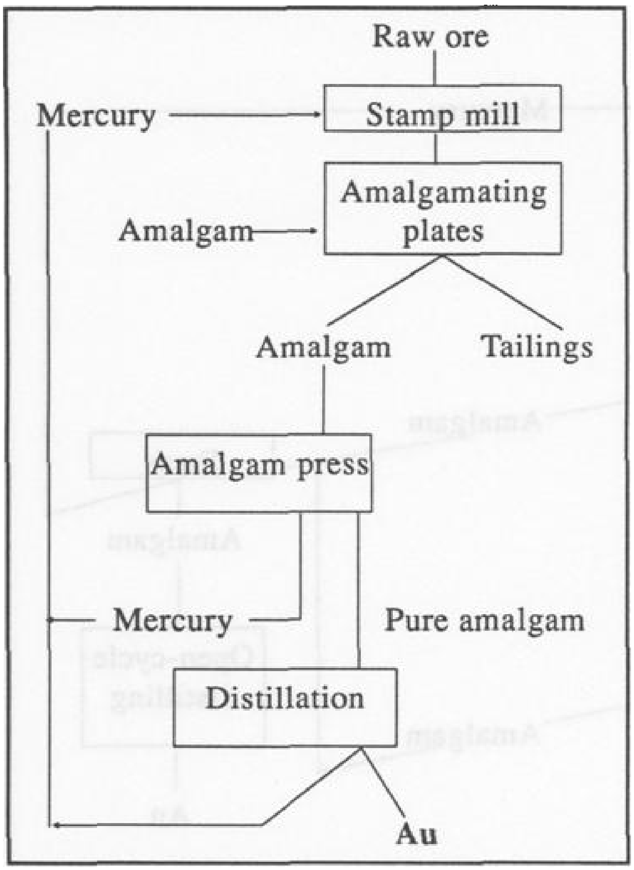
A crucial problem in the engineering of gold ore dressing equipment is how to cope with a batch-processing situation. Such is the case when a dressing plant provides custom milling and separation services for, say, an entire small-scale mining region. Most of the raw ore quantities brought in by the miners are rather small, often weighing less than 10 tons. To handle such small amounts of gold-bearing ore, the crushing, classifying and sorting equipment must be designed for quick, easy and complete discharge. Size-reducing implements in particular, especially closed mills, can be quite time-consuming to empty and clean, sometimes requiring more time than it takes to mill/grind and prepare the ore itself. The time factor can seriously reduce the effective throughput rate of such equipment. Consequently, care must be taken at the planning stage to integrate only such pieces of equipment into the overall plant as have been designed with batch processing in mind. For example, stamp mills or vibrating rod mills (cf. ch. 3.1.5) are preferable to closed ball mills.



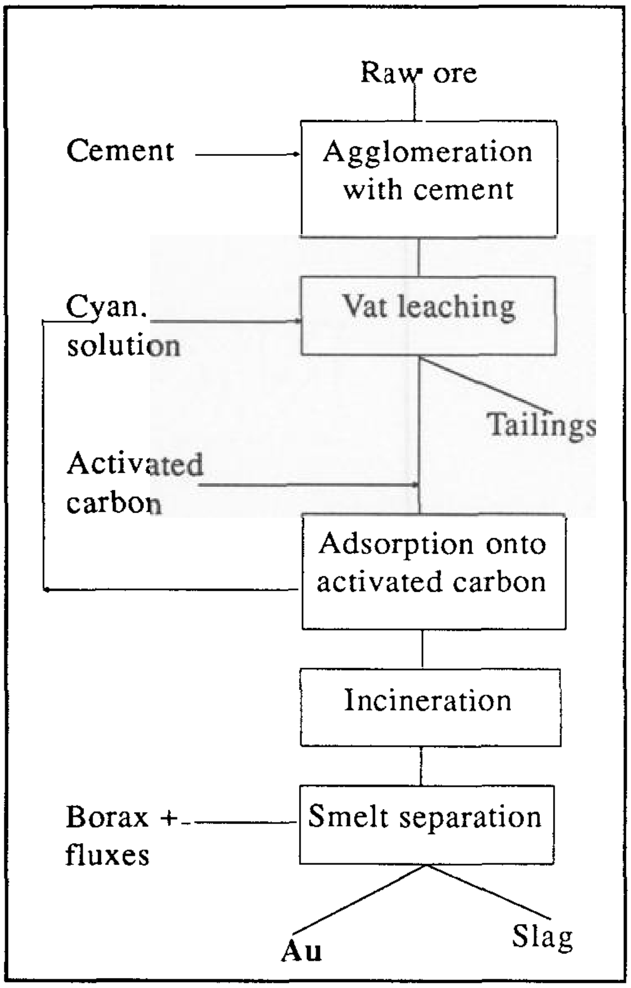
Flowsheet for a mechanized primary gold ore dressing plant in Bolivia (Mina Luchusa/Dept. La Paz) serving here as a typical representative of small and medium-size operations in Chilean gold mining.



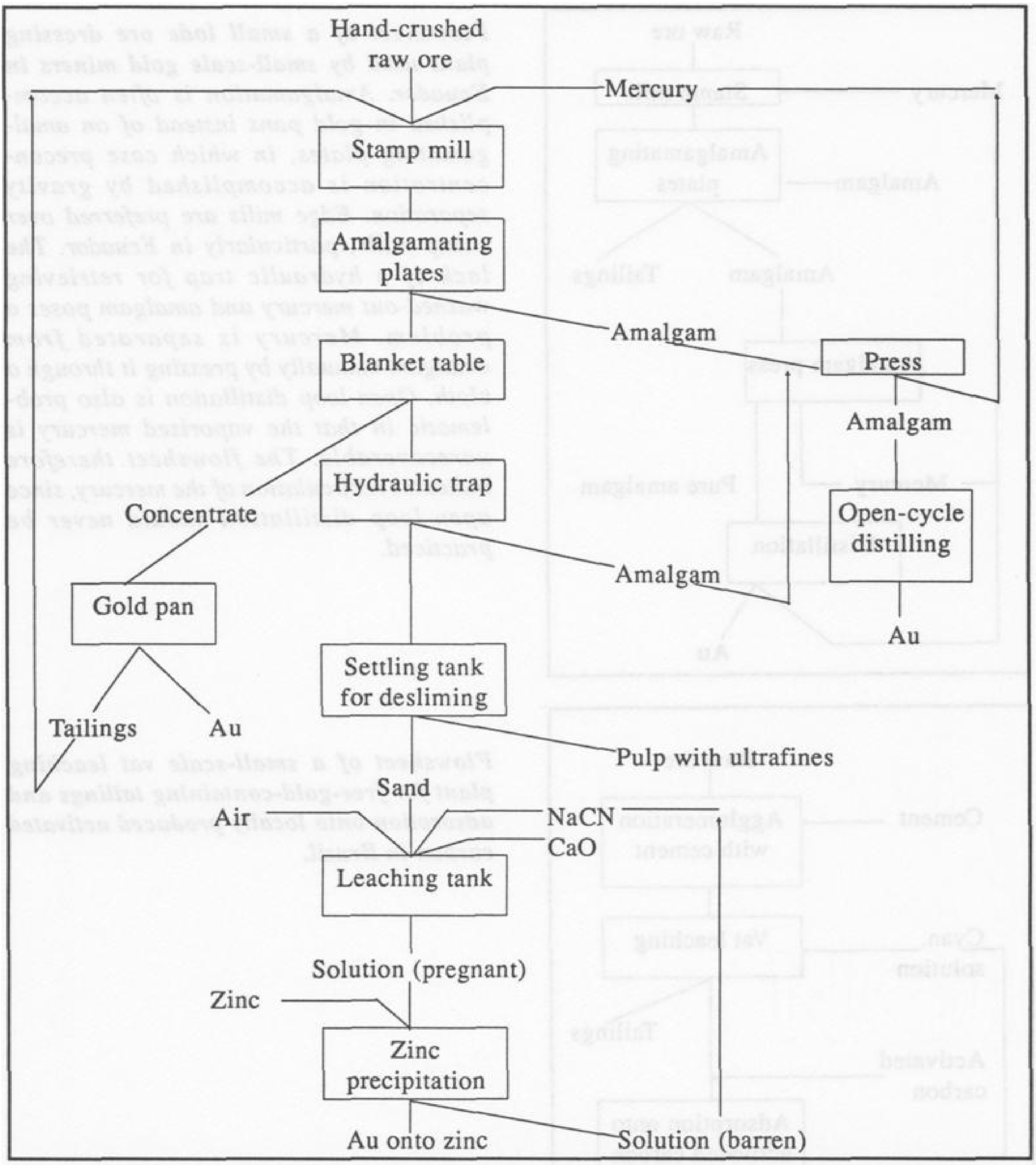
Flowsheet for a small gravimetric processing plant for primary gold ore in the Andes (Nariño, Colombia). *This type of plant is used in Nariño for dressing rich, intimately intergrown primary ore from sulfidic deposits exhibiting substantial arsenopyrite contents that preclude amalgamation. The hand-crushed ore is milled to roughly 50 % < 100 µm in wooden or metal stamp mills, most of which are powered by overshot wooden waterwheels or, more rarely, by small internal- combustion engines with V-belt power transmission. Blanket table operation is problematic in that high throughput calls for frequent cleaning of the blankets or, alternatively, acceptance of their plugging up by sedimentation and accordingly high gold losses. The sulfide-rich middlings of the preconcentrates are stockpiled for weathering and exposure to oxidation, particularly along the grain growth boundaries, in order to facilitate liberation of the values upon renewed milling.*



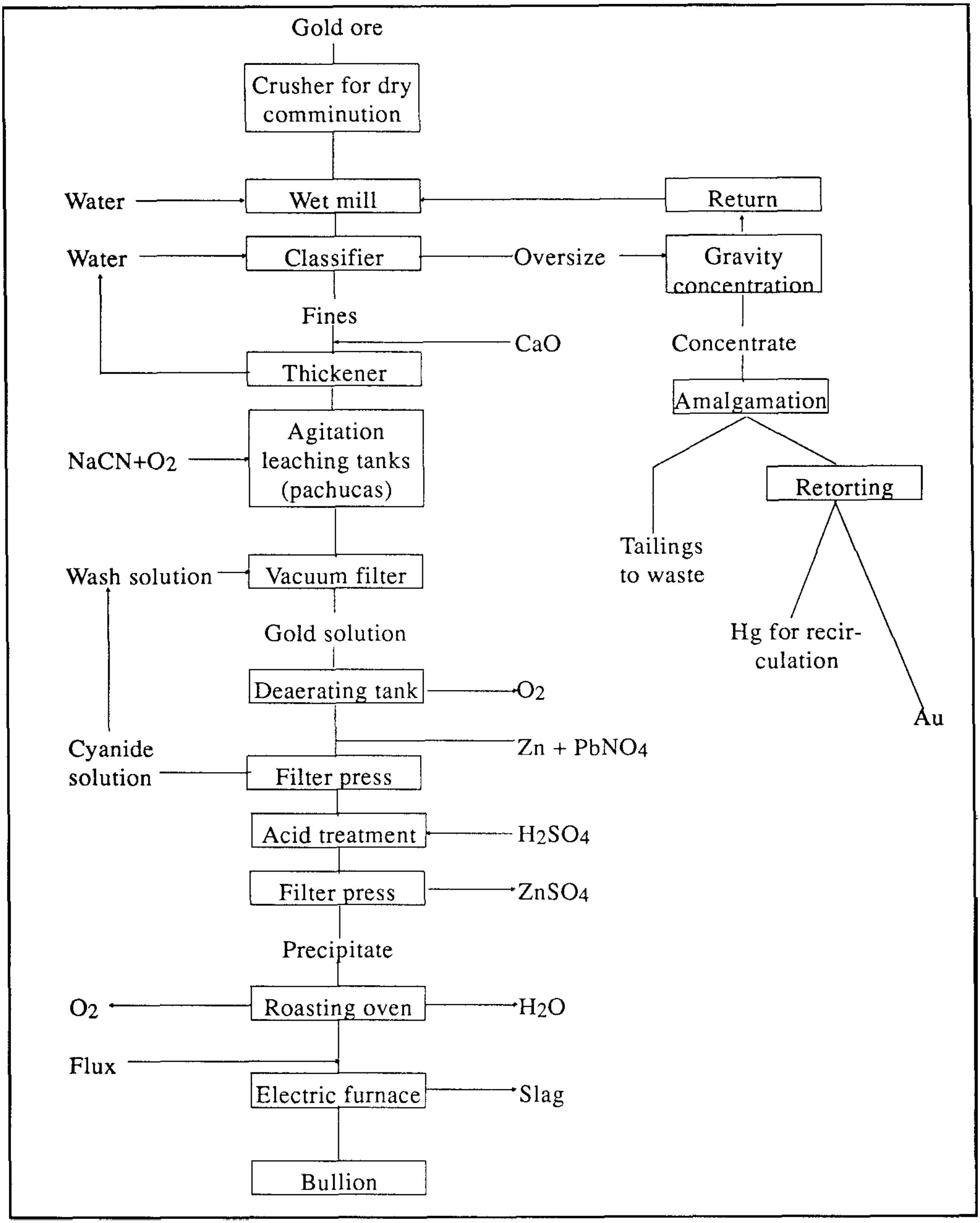
Flowsheet of a small lode ore dressing plant used by small-scale gold miners in Ecuador. *Amalgamation is often accomplished in gold pans instead of on amalgamating plates, in which case preconcentration is accomplished by gravity separation. Edge mills are preferred over stamp mills, particularly in Ecuador. The lack of a hydraulic trap for retrieving washed-out mercury and amalgam poses a problem. Mercury is separated from amalgam manually by pressing it through a cloth. Open-loop distillation is also problematic in that the vaporized mercury is unrecoverable. The flowsheet therefore indicates recirculation of the mercury, since open-loop distillation should never be practiced.*



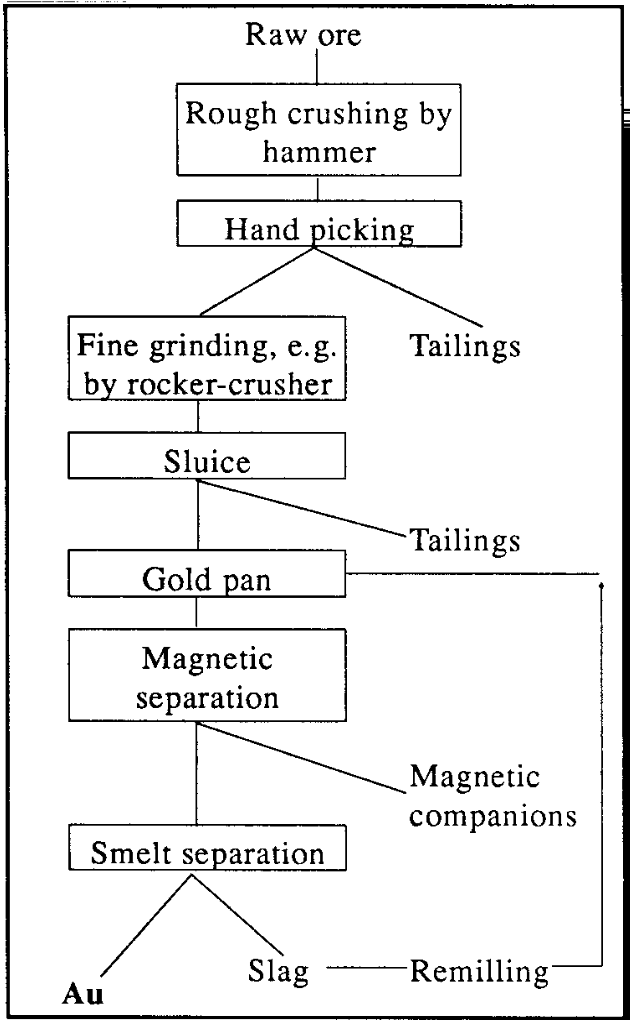
Flowsheet of a small-scale vat leaching plant for free-gold-containing tailings and adsorption onto locally produced activated carbon in Brazil.



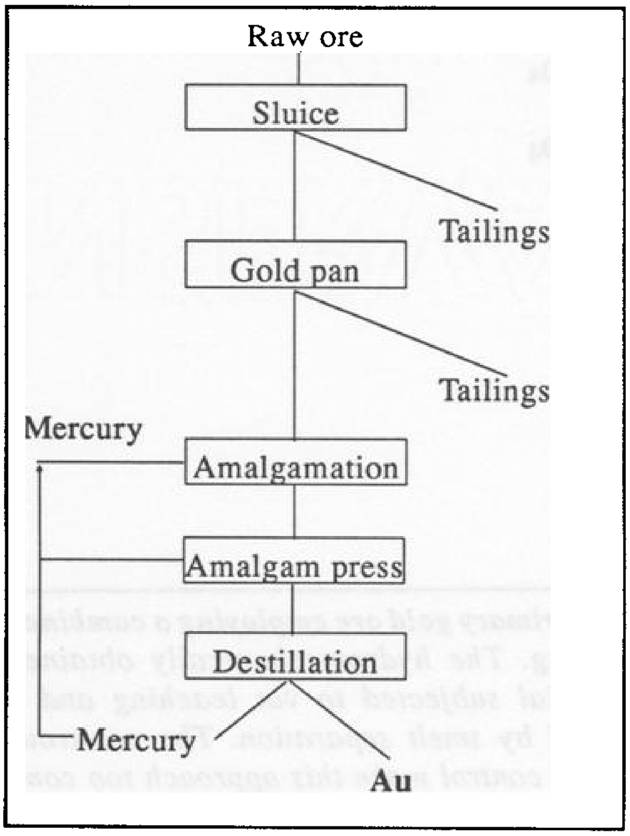
Flowsheet of a dressing plant for primary gold ore with amalgamation and leaching at Mina Los Guavos, Nariño, Colombia. *At this plant, amalgamation begins in a waterwheel-powered stamp mill, where up to 90 % of the recoverable gold amalgamates. After the stamping process, the mill is shut off and the amalgam collected by hand out of the mortar box. The amalgamating table is used for amalgamating and both tables together serve in recovering washed-out, floured mercury and amalgam. Separation of amalgam and mercury is manual, i.e., by squeezing through a cloth. The distillation setup is not equipped for recovering the mercury. A simple hydraulic trap is provided for catching any mercury or amalgam that may have escaped the amalgamating process; otherwise, the mercury could bond gold in the subsequent leaching plant. The product of leaching (gold on zinc wool) is sold to a refinery.*



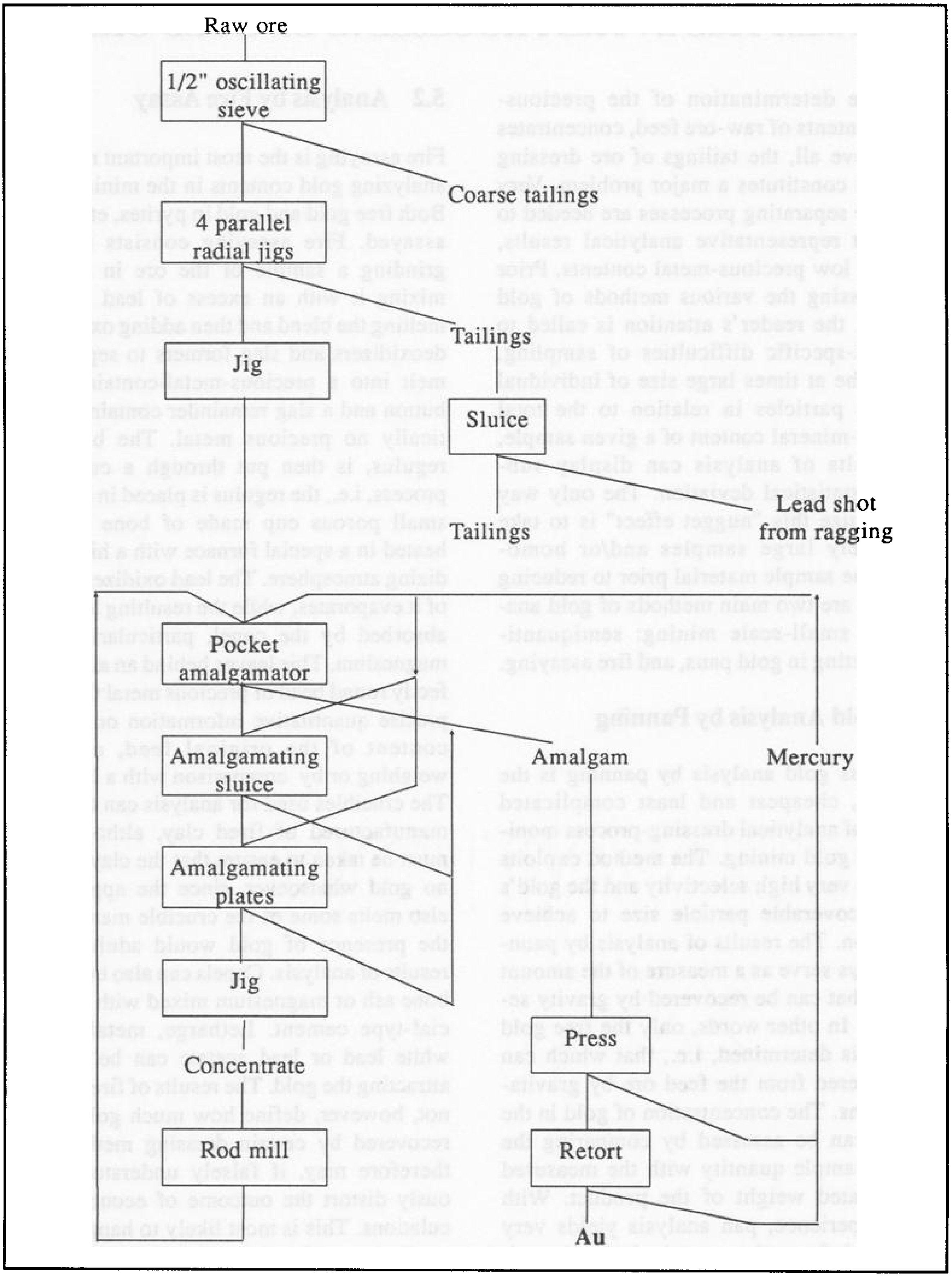
Flowsheet of a large, mechanized dressing plant for primary gold ore employing a combined process based on gravity separation and leaching. *The hydromechanically obtained preconcentrates are amalgamated, the fine material subjected to vat leaching and a subsequent Merrill-Crowe process and then refined by smelt separation. The numerous individual steps and, above all, the intricate process control make this approach too complicated for small-scale mining operation.*



Flowsheet of a small-scale manual dressing plant for primary gold ore in the Philippines. *This arrangement is suitable for processing high-grade ore obtained from formal veins. While the throughput at the size-reduction stage is comparatively low due to the manual work involved, prior hand picking minimizes the feed quantity for the grinding and sorting stages. Manual separation requires sharp boundaries between ore and neighboring rock, as is the case in vein-type ore bodies with no impregnation zone. The concentrates are enriched by manual magnetic separation, which eliminates relatively large amounts of black heavy-mineral sand.*



Flowsheet of a small, manual placer gold dressing plant in Brazil. *Apart from amalgamating the preconcentrates, amalgamation is often effected directly in the sluices, so that large amounts of metallic mercury escape with the water.*



Flowsheet of a mechanical placer gold dressing plant near Barbacoas, Nariño, Colombia

5. ANALYTICS IN THE PROCESSING OF GOLD ORE

Accurate determination of the precious-metal contents of raw-ore feed, concentrates and, above all, the tailings of ore dressing facilities constitutes a major problem. Very selective separating processes are needed to arrive at representative analytical results, even for low precious-metal contents. Prior to discussing the various methods of gold analysis, the reader's attention is called to the gold-specific difficulties of sampling. Due to the at times large size of individual valuable particles in relation to the total valuable-mineral content of a given sample, the results of analysis can display substantial statistical deviation. The only way to minimize this "nugget effect" is to take adequately large samples and/or homogenize the sample material prior to reducing it. There are two main methods of gold analysis in small-scale mining: semiquantitative sorting in gold pans, and fire assaying.

5.1 Gold Analysis by Panning

In-process gold analysis by panning is the quickest, cheapest and least complicated method of analytical dressing-process monitoring in gold mining. The method exploits the pan's very high selectivity and the gold's small recoverable particle size to achieve separation. The results of analysis by panning always serve as a measure of the amount of gold that can be recovered by gravity separation. In other words, only the free gold fraction is determined, i.e., that which can be recovered from the feed ore by gravitative means. The concentration of gold in the sample can be assessed by comparing the feed or sample quantity with the measured or estimated weight of the product. With some experience, pan analysis yields very accurate information, particularly since it often is a main concern of process control to find out whether or not the feed or tailings contain any gold at all.

5.2 Analysis by Fire Assay

Fire assaying is the most important method of analyzing gold contents in the mining sector. Both free gold and gold in pyrites, etc., can be assayed. Fire assaying consists of finely grinding a sample of the ore in question, mixing it with an excess of lead and flux, melting the blend and then adding oxidizers or deoxidizers and slag formers to separate the melt into a precious-metal-containing lead button and a slag remainder containing practically no precious metal. The button, or regulus, is then put through a cupellation process, i.e., the regulus is placed in a cupel (a small porous cup made of bone ash) and heated in a special furnace with a highly oxidizing atmosphere. The lead oxidizes and part of it evaporates, while the resulting letharge is absorbed by the cupel, particularly by the magnesium. This leaves behind an almost perfectly round bead of precious metal that yields precise quantitative information on the gold content of the original feed, either by weighing or by comparison with a line scale. The crucibles used for analysis can be locally manufactured of fired clay, although care must be taken to ensure that the clay contains no gold whatsoever, since the applied heat also melts some of the crucible material, and the presence of gold would adulterate the results of analysis. Cupels can also be made of bone ash or magnesium mixed with commercial-type cement. Letharge, metallic lead, white lead or lead acetate can be used for attracting the gold. The results of fire assay do not, however, define how much gold can be recovered by certain dressing methods and therefore may, if falsely understood, seriously distort the outcome of economic calculations. This is most likely to happen when analyzing refractory ores, where the ore contains a large amount of disseminated gold in the crystal lattice structure of pyrites and other sulfide minerals.

6. ENVIRONMENTAL AND HEALTH IMPACTS OF GOLD PROCESSING

Small-scale gold processing activities can pose a substantial hazard to the environment and to human health, particularly if mistakes are made in leaching and amalgamation. Those processes involve the use of cyanide and/or mercury, two highly toxic reagents that require very careful handling - something which is rarely the case in small-scale mining.

The processing of gold can give rise to the following nuisances and hazards:

Noise emission by comminuting equipment and their drive units.

Air pollution resulting from:

- engine exhaust
- dust emissions, mainly from dry classifying and sorting activities, e.g., winnowing (cf. ch. 3.4)
- mercury vapor escaping from open-cycle distillation of amalgam. Mercury contamination is a serious human health hazard. The symptoms of mercury poisoning are colics, loss of hair, ulceration and acute enteritis and can result in the death of the victim.

  Consequently, small-scale miners must always distill their amalgam in a closed-cycle arrangement. Retorts of the type described in chapter 3.3.5 are suitable for such operations. Mercury is prone to relatively rapid evaporation. Miners must always be sure to keep mercury under water in order to minimize the escape of mercury vapor to the atmosphere.

Water pollution by the effects of gold dressing can be both serious and hazardous, primarily as a result of the miners' using surface water for diverse processes. Particularly in semi-arid regions, rivers are often the "source of last resort". Basically, there are four fundamental uses for water:

1. as drinking water, in many cases directly from the river with no prior treatment,
2. as agricultural irrigating water,
3. as industrial process water, including mining and ore-processing,
4. as a habitat for potential food: particularly in remote tropical regions, fish are often the only source of animal protein.

Ore dressing can cause water pollution by release of the following substances:

- spent oil from equipment engines (1 liter of oil can contaminate 1 million liters of water)
- sludge, or slime, resulting from all wet-mechanical and flotative dressing processes. The resultant sedimentation fouling of the receiving water course is referred to as "siltation" and can seriously alter the environment of the affected waters. Recommended countermeasures include the construction and use of filtering dams, settling tanks, thickeners, tailings ponds - in addition to generally appropriate extraction planning.
- toxic flotation reagents such as sulfuric acid, diesel oil and long-chain hydrocarbons (frother and collector agents) like xanthates. Compared to large-scale flotation processes, small-scale mining involves extremely high reagent concentrations. The overdosing of reagents should be systematically avoided in order to prevent such contamination;
- cyanides used for leaching gold. Cyanides take approximately two years to break down, whereas intensive ultraviolet radiation in tropical regions accelerates the process. With a view to main-

taining the reactivity of the cyanide solution and/or to minimizing the cyanide alimentation requirement, the solution must always be protected against the effects of ultraviolet radiation throughout the process. Pertinent measures may consist of a simple corrugated metal or palm-frond roof over leaching vats and precipitation/recirculating tanks.

The tailings from leaching processes must be stockpiled or stored in settling tanks or tailings ponds. Such facilities should have full exposure to ultraviolet radiation in order to destroy the excess cyanide as quickly and efficiently as possible. The tailings can also be detoxified by mixing the tailings with water so as to lower the pH and drive out the HCN. Leaching tanks and vessels should be covered with wire netting to keep people and animals from coming into direct contact with the toxic solution.

mercury and amalgam. These two substances enter the environment via gold amalgamation in open reactors, e.g., in stamp mills, edge mills, sluices or tables. Metallic mercury escaping from amalgamating equipment is almost always in the form of the minute beads referred to as floured mercury. The beadlets are usually contaminated mechanically or chemically, and their high surface tension renders them inactive, i.e., they can neither amalgamate with gold nor fuse and are therefore washed out in case of sufficient pulp velocity. Such Hg losses can be avoided by separating the frequently simultaneous size-reduction and amalgamation stage into two separate steps. Chapter 3.5 describes the requisite amalgamating equipment. Normally, the process sequence will have to be changed. If that is not possible, at least some means of recovering the lost mercury and amalgam must be introduced, e.g., hydraulic traps (cf. ch. 3.3.10).

Excessive water pollution by mercury leads to severe environmental and health hazards, all the more so as it promotes methylization and resultant formation of highly toxic methyl-mercury in aquatic ecosystems. Mercury and mercury compounds enter the food chain and accumulate in fish, often reaching a concentration far in excess of that found in the natural surroundings.

All gold-dressing activities presuppose prior extraction of the ore from open-cast or underground workings. Such activities can also cause pollution and health hazards through:

- land consumption
- irreversible intervention in the landscape
- modification of the groundwater and surface-water regimens
- alteration of the soil and of the plant and animal life
- social problems resulting from conflicts of interest, resettlement, etc.
- dissemination of disease carriers (e.g., malaria) by formation of new stillwater bodies.

Table 1: Throughputs and initial costs of various gold dressing equipment

Device	Throughput	Cost of investment
Jaw crusher	> 350 kg/h	≅ US$ 1000 (350 kg/h) to US$ 4000 (1,5 t/h) for local manufacture (Bolivia)
Stamp mill	≅ 0.8-2.5 t/d per piston	steel stamp mills ~US$ 5500 incl. motor (Colombia), wooden 3-piston stamp mill ≅ US$ 700 without motor (Colombia)
Edge mill	≅ t/h for a power rating of 5-7 kW	≅ US$ 2800-3300 for local manufacture (Bolivia)
Ball mill	≅ 1 t/h for a power rating	US$ 4500 for 1 t/h, local
Rod mill	of 11-12 kW	manufacture (Bolivia) US$ 22000 for 1 t/h (US-production)
Quimbalete, Maray	≅ 1 kg/man x minute	very inexpensive, normally self-built
Washing sluice	≤ several 1 t/h	very inexpensive, normally self-built
Fixed screen	heavily particle-size-dependent, ≅ 100 kg/h (ladder-type) to several t/h (grizzly)	≅ US$ 170 for ladder-type system with 3 screens
Shaking screen	heavily particle-size dependent, but relatively high compared to fixed screens	> US$ 55 per screen for local manufacture
Sizing trommel	0.27 t/m^2 x h per mm mesh width for dry screening 0.45 t/m^2 x h per mm screen width for wet screening	≅ US$ 850 for local manufacture
Mechanized compact gold processors, e.g., Denver Gold Saver	2-3 m^3/h	≅ US$ 14000 ex factory or ≅ US$ 2200 for local manufacture (Colombia, Ecuador)
Hydrocyclone	≤ 100 t/h (or more), size-dependent	size-dependent, from US$ 55 to US$ 550
Spitzkasten	≅ 0.5-1 t/h	≅ US$ 110 for local manufacture
Hydroclassifier	≅ 0.5-1 t/h	≅ US$ 280 for local manufacture
Spiral classifier	≅ 2-4 t/h	US$ 550-2800 for local manufacture
Rake classifier	≅ 1 t/h	≅ US$ 5500 for local manufacture
Gold pan	≅ 1 t/d	≅ US$ 5-11
Jig	> 0.5 t/h	> US$ 1700 for local manufacture (Ecuador)
Sluice	20 kg/h to several t/h,dep. on size and operating mode	≅ US$ 5-11/m for local manufacture
Rocker	0.5-1 t/d	> US$ 110 for local manufacture
Dug-channel sluice	≤ several t/h	cost of labor for digging
Pinched sluice	> 0.5 t/h	> US$ 85 for local manufacture (Bolivia)
Spiral concentrator	> 2 t/h	> US$ 3300
Sweeping table	≅ 100 kg/h	≅ US$ 110 (self-built)
Blanket table	≅ 250 kg/h x m^2 deck area	≅ US$ 110 (self-built)
Tilting table	> 100 kg/h	≅ US$ 550 (self-built)
Concussion table	≅ 250 kg/h x m^2 deck area	≅ US$ 140 for local manufacture (Colombia)

Device	**Throughput**	**Cost of investment**
Centrifugal concentrator - Knelson 7.5" - Knelson 12" - Knudson - Falcon B12 - Falcon B6 - Vardax	 650 kg/h 5 m3/h 3-4 t/h 6 t/h 0.5 t/h 2 t/h	fob prices US$ 6850 US$ 12500 US$ 4500 US$ 34000 US$ 7000 US$ 2400
Hydraulic trap	several t/d	> US$ 55-110 for local manufacture
Air sluice	several 100 kg/h	> US$ 280 for local manufacture
Dry washer	several 100 kg/h	US$ 110-550 for local manufacture, dep. on power rating
Air table	> 1 t/h	> US$ 5500 for industrial manufacture
Winnower	≅ 100 kg/h	> US$ 280 for local manufacture
Pinched sluice with pneum. fluid. bed	several 100 kg/h	≅ US$ 550 for local manufacture
Amalgamating copper plates	≅ 3 t/d x m^2 plate area	≅ US$ 550 for local manufacture
Amalgamating barrel	> 50 kg preconcentrate/h, size-dependent	simple versions > US$ 55 for local manufacture
Pocket amalgamator/ Jackpot	0.5 t/h	≅ US$ 30 for local manufacture
Amalgam press	≤ several 100 kg amalgam-Hg mixture per day	≅ US$ 55 for local manufacture
Distillation retort	0.5-100 kg capacity, 20-30 min. process duration	US$ 55-280 for local manufacture, size-dependent

Table 2: Size of feed [µm] for different forms of processing equipment and techniques, with parenthesized top and bottom limits (extremes)

Wet classifier	(50) 75 - 5,000
Dry classifier	(40) 100 - 10,000
Sizing trommel	(250 - 50,000
Denver Gold Saver	100 - 100,000
Washing sluice	30 - 200,000
Conical hydrocyclone	(5) 10 - 200
Hydroclassifier, spitzkasten	(20) 50 - 1,000 (2,000)
Rake classifier, spiral classifier	200 - 5,000
Gold pan	(20) 50 - 30,000
Jig	50 - 2,500
Sluice	(60) 100 - 1,500 (3,000)
Rocker	100 - 2,000
Pinched sluice/conical separator	(30) 50 - 1,000 (3,000)
Spiral concentrator	(30) 50 - 1,000 (3,000)
Concussion table	(10) 30 - 2,000
Tilting table	100 - 2,000
Blanket table	50 - 2,000
Shaking table	(20) 50 - 1,000 (3,000)
Fluidized-bed centrifuge	30 - 2,000
Hydraulic trap	50 - 1,000 (3,000)
Pneumatic jig	30 (200) 500 - 2,000
Air table	50 - 600 (50,000)
Dry sluice	200 - 1,500
Pinched sluice with pneumatic fluidized bed	75 - 1,500
Amalgamation	(20) 50 - 2,000
Amalgamating copper plates	500 - 1,000
Gold leaching	0 - 750
Flotation	(5) 15 - 500
Froth flotation	(100) 150 - 1,500 (2,000)
Weak-field wet magnetic separator	(40) 50 - 2,000 (5,000)
Strong-field wet magnetic separator	(10) 20 - 500 (2,000)
Dry magnetic separator	
- weak-field	100 - 5,000
- strong-field	80 - 1,000

Table 3: Reference values for feed-pulp solids contents in classifying, sorting and clarifying equipment

	Feed-pulp solids content
Washing sluice	max. 10 vol. %
Shaking screen	20 - 40 vol. %
Hydrocyclone	max. 20 - vol. % (< 40 vol. % in extremes)
Hydroclassifier, spitzkasten	25 - 40 vol. %
Spiral classifier, rake classifier	30 - 50 vol. %
Jig	max. 10 vol. %
Sluice	10 - 20 vol. %
Pinched sluice	30 - 40 vol. %
Spiral concentrator	max. 15 - 20 vol. %
Sweeping table	max. 15 - 20 vol. %
Blanket table	max. 10 - 20 vol. %
Tilting table	max. 15 vol. %
Concussion table	20 - 50 wt. %
Shaking table	15 - 20 vol. %
Amalgamating copper plates	max. 20 vol. %

Table 4: Alternative methods of gold ore processing

Table 4a: Methods of obtaining gold ore preconcentrates

Type of separation	**Process**	**Steps**	**Equipment**	**Reagents**
mechanical	**GRAVIMETRIC DRESSING**	sorting	gold pan, jigs, sluices (wet and dry), tables (wet and dry), animal hides, centrifugal classifiers, spiral classifiers, CBC cyclone separators, hydraulic traps	
	HEAVY-MEDIA SEPARATION*	sink/swim concentration	heavy medium separator, glass-flask	dihydrogendodeca-wolframate rho = 3.1 g/cm^3
electric	**ELECTROSTATIC DRESSING**		electrostatic separator	
surface	**FLOTATION**			
	- indirect	conditioning sorting washing	conditioning tank flotation cell	frother agents, collector, depressant, activators, pH reactants
	- direct	conditioning sorting washing		
magnetic	**MAGNETIC SEPARATION**	sorting	magnetic separator	

* as an analytical method only

Techniques printed in bold are wholly or partly applicable to small-scale mining.

Table 4b: Methods of refining gold preconcentrates into high-quality marketable gold concentrates

Type of separation	Process	Steps	Equipment	Reagents
mechanical	**HAND SORTING**			
	GOLD COAL AGGLOMERATION	agglomeration, detachment stripping	reactor	oil, activated carbon
thermal	**ROASTING AND WINNOWING**	roasting, winnowing	roasting furnace	
	GOLD VOLATILIZATION	chlor. roasting, volatilization		common salt, chlorine gas
	SMELT SEPARATION	smelting, detachment of gold	furnace, crucible	borax, soda, potash
	FIRE ASSAY*	smelting with gold collector, detachment, cupellation	box furnace, crucible cupel	lead as collector, borax, soda, potash
chemical	**AMALGAMATION** with open or closed Hg cycle	alloying, detachment, destilling	in stamp mill, edge mill, amalgamating barrel, gold trap, sluice, amalgamating copper plates, gold pan, centrifuge, amalgam press, distillation retort	mercury, poss. caustic soda, sodium amalgam, ammonium chloride, cyanide or nitric acid for joining finest Hg spheres, surface-active agents
	CYANIDATION as heap leaching, vat leaching or agitation leaching with Merill-Crowe, CIP, CIC, CII process or zinc precipitation	chem. solution as complex, adsorption, stripping	leaching tanks, adsorbers	Na cyanide, CaO to adjust the pH value, Zn (poss. + $PbNO^3$) or activated carbon
	THIOUREA LEACHING	chem. solution as complex, adsorption, stripping	leaching tanks adsorbers	thiourea, pH react., Al or Fe powder, SO_2
	CHLORINATION BIO-D-LEACHANT PROCESS	halogen complexing (e.g., tetrachloride complex)	reactor tanks, leaching tanks	chlorine gas, organic bromide complexes
	BRINE LEACHING			salt solutions, manganese dioxide, sulfuric acid
	OTHER LEACHING			thiosulphate, thiocyanate, polysulfide or nitrile-containing solutions
biological	BACTERIAL LEACHING			bacteria, air as oxidizing agent

* as an analytical method only

Techniques printed in bold are wholly or partly applicable to small-scale mining.

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List of Abbreviations

"	inch, ~2.5 cm
$	U.S. dollar, ~1.7 deutschmarks
%	percent
<	less than
>	more than
A	Ampere
approx.	approximately
Au	gold
cm	centimeter
cm^3	cubic centimeter
CN^-	cyanide ion
Cu	copper
d	day
DM	deutschmark, ~0.6 $
e.g.	for example
Eh	redox potential
etc.	et cetera
µm	micron = 1/1000 mm
g	gram
GO	governmental organization
h	hour
HCN	hydrogen cyanide
Hg	mercury
i.e.	that is
incl.	inclusive
kg	kilogram
kW	kilowatt
kWh	kilowatt hour
l	liter
m	meter
m	square meter
m^3	cubic meter
mesh	number of screen apertures per inch
mg	milligram = 1/1000 g
min^{-1}	speed or frequency: per minute
mill.	million
mm	millimeter
NGO	nongovernmental organization
pH	negative decimal logarithm of effective hydrogen-ion concentration
ppm	parts per million grams per ton
t	ton = 1000 kg
UV	ultraviolet
V	Volt
vol.%	percent by volume
W	Watt
w/o	weight %
Zn	zinc
°	angle gauge
°C	degrees centigrade

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