DRAFT

PILOT PLANT

TREATMENT OF MINE DRAINAGE

WALKER MINE, PLUMAS COUNTY, CALIFORNIA

PREPARED FOR

CALIFORNIA REGIONAL
WATER QUALITY CONTROL BOARD,
CENTRAL VALLEY REGION

BY

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RICHMOND, CALIFORNIA

UNDER

REGIONAL BOARD PROJECT 1-043-150-0

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CONCLUSIONS

1. Removal of 97% of total copper from Walker mine drainage, to produce an effluent containing 0.5 mg Cu/L, was demonstrated in a 1.6 gpm chemical neutralization-sedimentation pilot plant operating at pH 10.2. The sludge produced thickened by gravity, and evaporated to dryness in summer air.
2. Batch studies, extending the scope of pilot plant work, indicate that at pH 11 a 2.2 acre sedimentation basin could settle the expected peak flow of 2 cfs, to produce an effluent containing approx 0.2 mg/L of total copper, to approximately meet the 0.01 mg Cu/L Basin Plan limit for receiving waters.
3. A 1.8 gpm pilot limestone barrier 500 ft long produced a stable effluent pH of 6.5 from pH 4.9 influent, and converted up to 90% of total copper in the mine drainage to solids that settled readily in batch mode.
4. Pilot facilities were set in unmanned operation for the current winter.
5. Given the required effluent copper concentration, and unit costs for items of construction of the prototype plant and for chemicals, and the economic present worth factor, procedures are presented (based on pilot plant and batch data), to identify, and to cost, the optimal prototype design for minimum total cost.
6. Available construction funds of $150,000 were found to be inadequate for a chemical neutralization-sedimentation plant to handle 2 cfs peak flow, even to produce an effluent containing 4.5 mg/L of total copper, that would cause the Basin Plan limit to be exceeded by a factor of 20. Presently, the Basin Plan limit for copper in the stream at the mine discharge is exceeded by a factor of approx 250. The item most demanding of construction funds is a sedimentation basin to resist destructive washout in the event of overtopping. In winter, 30% of copper in stream water is attributed to seepage from tailings.
7. The recommended plan with available funds is to construct a crushed limestone barrier and a sedimentation basin (or series of smaller basins), as large as safety and funds permit (to be determined in detail design), but probably not sufficiently large to provide effective settlement of brief, spring thaw, peak flows. Such an approach would provide an almost 'walk-away' interim partial solution, considered as the first step towards total abatement.

SUMMARY

The California Regional Water Quality Control Board, Central Valley Region, undertook a program to abate copper pollution that has destroyed aquatic life in approx 10 miles of stream in Plumas County, California, due to drainage from the inactive Walker Mine. As part of this program ditches were constructed in 1981 to divert surface waters from mine openings, and a recording flow measuring flume was also constructed in that year. In 1982 pilot mine drainage treatment facilities were constructed and operated, the subject of this report.

Pilot facilities included a neutralization-sedimentation-filtration unit that was operated for 6 days at a mean flow of 28 gpm, and for 57 days at a mean flow of 1.6 gpm. Also included were a 500 ft long crushed limestone barrier, graded at 12.6%, that was operated for 92 days at a mean flow of 1.8 gpm, and a copper cementation unit, sporadically operated, that removes copper from water by electrochemical replacement with iron from scrap steel. In addition, water levels in three evaporation ponds were observed, and the receiving stream, Dolly Creek was monitored above and below the mine.

The neutralization-sedimentation-filtration pilot plant included two limestone pre-neutralization processes, chemical neutralization to raise the pH to 9 to 10, a 15 ft fall spray decarbonation process, sedimentation in a 1500 sq ft basin with a mean depth of 1.0 ft to remove chemically precipitated copper, followed by filtration through straw bales.

The two limestone preneutralization processes were to raise the pH by up to 1 unit from the initial pH of 4 to 5, to reduce chemical consumption. Only one limestone process was in use at a time, the tumbling drum at high flow (28 gpm mean), and the autogenous mill at low flow (1.6 gpm mean).

In the tumbling drum, crushed limestone is loaded into the hub of a

water wheel that is turned by the flow of mine drainage. Rotation of the water wheel causes the stone to abrade, generating limestone fines that dissolve into and preneutralize the mine drainage.

The autogenous mill is also a drum containing crushed limestone, but for the mill the load of stone is larger than for the drum, and the speed of the mill is geared down from the water wheel so that the mill rotates typically at 1 rev/hr, compared with several rpm for the tumbling drum. This rotation also abrades the surface of stone in the mill, but less vigorously than in the drum, so that although the limestone surfaces are continually scoured clean of deposited metals, few fines are produced. Mine drainage is preneutralized while flowing through the mill before passing to the water wheel that rotates the mill.

As well as powering the tumbling drum and autogenous mill, the water wheel also powers the chemical feed pump. Consequently, the entire neutralization process is water-powered; in fact no outside power source is needed for any of the processes investigated, that ran themselves for up to three months continuously. The speed of the chemical feed pump automatically adjusts to the mine drainage flow being treated. Two reagents were dispensed to the flow by the chemical feed pump, first an approx 5.5% solution of soda ash for the duration of the high flow studies and for approx half the duration of the low flow studies, and second an approx equal alkalinity mixture of soda ash and caustic soda that was used for the remainder of the low flow studies. A peristaltic pump, delivering from 11 to 30 mL/rev, was used to dispense chemical, after a flexible impeller positive displacement pump was found to leak excessively between the sides of the impeller and the pump casing.

The pilot plant was monitored 5 times during the 4 days of operation at a mean flow of 28 gpm. Over this period, the mean increase in pH from raw mine drainage to final effluent was 4.9 units to pH 9.9. Removals of of total copper, zinc, manganese and iron during this period were 72%, 62%, 53% and 11% respectively, the low removal of iron being possibly related to dissolution of iron from the steel process units. The tumbling drum increased pH by an average of only 0.3 units, declining from an initial 0.9 pH units increase to zero pH increase at 11 days, shortly before the limestone was exhausted after 14 days. The straw bale filter did not remove further copper after sedimentation, so that the chemical neutralization-sedimentation processes achieved essentially the entire removal observed at high flow.

During 57 days of continuous operation at a mean flow of 1.6 gpm, the pilot plant was monitored 24 times. Over this period the mean pH increase was 4.7 units to pH 9.6 for the effluent, with a mean of 0.9 pH units increase in the autogenous mill and the remainder due to chemical neutralization. As for the tumbling drum, the degree of neutralization produced by the autogenous mill steadily declined over its operating period, from 3.3 pH units increase initially to 0.1 pH units at termination. Rectifiable problems were noted in both the tumbling drum and the autogenous mill, although no effort was made to implement improvements because it was desired to properly document their performance, and because the limestone barrier was performing so well that it was evident that the barrier would be the preferred preneutralization unit.

Removals of copper, zinc, manganese and iron during operation of the pilot plant at a mean flow of 1.6 gpm were (mean ± standard deviation): 93% ± 3%, 92% ± 4%, 84% ± 8% and 54% ± 35%, for total metals. Colorimetric analyses for copper, used for process control and interpreted as free copper, showed 98% ± 2% removal of free copper, the difference between removals of total and free copper being due to complexation of copper during neutralization. Complexed copper is biologically much less deleterious than free copper. Again, the straw bale filter had no significant effect on effluent quality.

In order to document the performance of the neutralization-sedimentation process train over a wider range of conditions than could be investigated in the pilot plant, batch neutralization and batch sedimentation studies were conducted. In batch neutralization tests, the pH increase and the removals of four metals from Walker mine drainage were determined for 21 dosages of each of three reagents, soda ash, equal alkalinity mixtures of cautic soda ash and caustic soda, and caustic soda. After approx 1 day of quiescent settlement, aliquots were removed from each reactor for determination of pH, copper, zinc, manganese and iron. For copper, no decrease in concentration was observed until the pH rose from 4.5 to 6, then a ten-fold reduction in concentration of copper in the supernatant was observed as the pH rose from 6 to 7 (paralleling and close to the theoretical solubility of copper), and a further ten-fold reduction in copper concentration with increasing pH from 7 to 11. Pilot plant data showed the same trend, but with copper concentrations 2 to 4 times higher at a given pH.

Sedimentation studies were conducted at the mine, in a 6 in. diam, 13 ft long column, the tests being monitored by colorimetric determination of free copper, from which total copper was estimated by a correlation. One 2 day batch test was conducted, and three continuous flow tests at different flows. Analyses of samples drawn from the column at one foot vertical intervals showed that copper removal was independent of depth from one foot above the column base, and depended only on sedimentation time. Removals of total copper increased from 75% for 15 min of settlement to 98% after two days according to a power law. Continuous flow column test results showed total copper residuals approx one-third of those in batch tests at the same surface overflow rate (computed as one foot of depth divided by detention time). Pilot sedimentation basin data showed residuals approx five times higher than predicted by the power law at the prevailing surface overflow rate, attributed to the inefficient outlet in the pilot basin.

Sludge from the sedimentation basin thickens readily by gravity, and thickened sludge evaporates at almost the same rate as water to a dry, dark green cake containing approx 25% copper. Sand bed dewatering of the sludge was unsuccessful; no solids were retained on a bed of rather coarse native sand when the sedimentation basin sludge drawoff mechanism was actuated.

The limestone barrier was operated for 94 days, excluding 3 days when the lower part of the barrier was frozen, and monitored during that period on 34 occasions. The frozen barrier thawed after a snow cover formed. From a mean influent pH of 4.9, the 500 ft long barrier increased the pH to a mean of 6.8. The effluent pH fell from over 8 initially to approx 7 in the first week of operation, then to pH 6.5 in the following month, remaining at pH 6.5 (with a standard deviation of 0.3 pH units) for the following seven weeks of the reporting period. The mean flow treated was 1.8 gpm.

Limestone barriers are designed by specifying the load factor, equal to the number of tons of crushed limestone in the barrier, divided by the stone size in inches and by the flow to be treated in cubic feet per second. For any barrier, effluent pH depends on load factor and influent pH. Designed for a load factor of 500 (by providing the appropriate tonnage of 1/4 - 1/2 in. crushed limestone), the geometric mean observed load factor corresponding to all observed influent and effluent pH data was 680, while for the final seven week period of stable operation the geometric mean observed load factor was 320. Thus, for stabilized operation at Walker mine, the reactivity of the limestone used is 320/500 = 64% of the reactivity for which the design load factor of 500 was calculated. This lower reactivity may be due to low temperatures during the period of stable operation, and/or to precipitation of metals in the barrier.

In particular early in operation of the limestone barrier, substantial removals of copper and iron were observed, but not zinc or manganese.

Copper removal increased to 90% in the first month of operation, where it remained for two weeks, before declining to approx zero at the end of the reporting period. Over this two week period the mean effluent pH was approx 7.0, the same as that producing 90% removal of copper in batch chemical neutralization. As the evaluation period progressed a green sediment was observed in the invert of the vee-section barrier flume, and occasionally in sunlit barrier effluent, though green solids were never observed in samples of barrier effluent. The limestone was tinged green.

When the lower section of the barrier was found to be frozen in mid-November a segment of the frozen barrier was removed, thawed, and the meltwater analyzed for metals. Meltwater contained 2500 mg/L of total copper (compared to 15 mg/L in raw mine drainage), 17 mg/L of zinc (cf 0.7 mg/L in mine drainage), 51 mg/L of manganese (cf 2.6 mg/L), and 160 mg/L of iron (cf 0.4 mg/L). Assuming that the distribution of metals along the barrier was uniform, metals stored in the barrier account for 40% to 60% of the amounts removed by the barrier up to the time of removing the frozen segment. Metal-rich sediment settled readily from the slurry-like meltwater, although the supernatant was not separately analyzed. Snow blocked vehicular access to the site on the next visit, precluding further work, but the barrier had thawed.

For design of limestone barriers as preneutralization units prior to chemical neutralization, it is necessary to interpret barrier performance in terms of acidity decrease through the barrier. Chemical consumption saved by the barrier per unit volume of water treated equals acidity decrease. For this purpose a number of alkalimetric titration curves on process waters from different points in the pilot facilities were determined on several occasions. All curves (corrected to zero standard acidity at an arbitarily selected reference pH of 7.5) exhibited the same pattern. The buffer intensity was 25 mg $CaCO_3$/L-pH from pH 4 to pH 11, except for pH 6.25 to pH 8.75 for which the buffer intensity was 5 mg $CaCO_3$/L-pH unit.

Thus the pH 4.9 to pH 6.5 rise in the barrier is equivalent to an acidity decrease of 25(6.25-4.9)+5(6.5-6.25) = 35 mg $CaCO_3$/L of acidity reduction, corresponding to annual chemical savings of 34 tons of $CaCO_3$ equivalent per cfs treated.

The copper cementation unit was a 55 gal drum containing approx 300 lb of detergent-washed steel turnings. Total metals were determined on only one pair of influent and effluent samples, when the influent pH was 5.1 and the flow through the unit was 1.0 gpm, corresponding to approx 50 min detention time. For these samples the influent and effluent concentrations of copper were 16.8 and 10.8 mg/L, and for iron 0.4 and 5.3 mg/L. The 94 uM decrease in copper concentration approximates the 88 uM increase in iron. In a previous attempt at copper cementation at Walker mine, sediment in mine water during the high flow spring thaw period fouled the process, so that a flume reactor rather than a tank reactor would be advisable.

Few evaporation data were obtained as a result of the late start with operation following a delay in gaining access to the site for construction, snowbound until late in the season from the previous severe winter. However, such data as were obtained were not inconsistent with published evaporation formulae.

The pilot limestone barrier, chemical neutralization plant and sedimentation basin are in unmanned operation for the current winter.

For a process train comprising a limestone barrier, chemical neutralization and sedimentation, economic optimization studies were carried out, to identify the optimal relative sizing of process units, and to provide a preliminary assessment of construction cost and annual chemical cost for economically optimized designs. Dependent variables considered were final effluent copper concentration, economic discount rate and economic life of the facility (combined into the present worth factor), and marginal cost for construction of the sedimentation basin per acre of area (for earthworks) or per lineal foot of basin peripheral length (for a walled basin).

Optimization and costing were based on two sets of economic parameter values, a discount rate of 10%/yr and an economic life of 10 yr, or a discount rate of 5%/yr and an economic life of 20 yr. The former assumption was less restrictive when considered with the financial limitation of $150,000 for construction, although operating costs were then higher than under the latter assumption. Prices paid for chemicals and for crushed limestone for pilot studies were assumed to be valid for prototype optimization, and $30,000 was allowed for construction of the chemical neutralization plant.

First, neutralization was optimally proportioned between limestone preneutralization and chemical neutralization, by identifying the barrier load factor at which the marginal cost of limestone treatment equalled the marginal cost of chemical neutralization. For the 10%/yr, 10 yr case the optimum barrier load factor was 120, corresponding to a barrier effluent pH of 5.8. For the 5%/yr, 20 yr case the optimum barrier load factor was 200, corresponding to a barrier effluent pH of 6.0. Thus, the 10%/yr, 10 yr assumption consumes less scarce capital, needed for sedimentation basin construction, but requires higher annual costs for neutralization chemical.

Next, the optimized marginal cost of neutralization was traded off against the marginal cost of the sedimentation basin, in order to determine the optimum process pH and basin size corresponding to given basin marginal costs and effluent quality requirements. This optimization used three experimentally determined relationships developed from pilot plant and batch treatment data: 1) the relationship between chemical dose and pH produced, from titration curve data; 2) the relationship between pH produced and the concentration of copper after quiescent settlement, from batch neutralization data; and 3) the relationship between the removal of precipitated copper in a sedimentation basin and the basin surface overflow rate, from settlement data.

Results of this optimization showed that for the more favorable economic assumption, 10% annual discount rate and 10 yr economic life, the least infeasible effluent copper concentration corresponding to a financial constraint on construction of $150,000 was 4.5 mg/L of copper in the final effluent, corresponding to a 70% reduction from the influent concentration of 15 mg/L. For the assumed peak flow of 2 cfs (i.e., 900 gpm, cf 830 gpm recorded over the severe 1981-82 winter), the optimal basin size was approx 1 acre, and the optimal process pH was 6.8, corresponding to an annual cost for neutralization chemical of $5,200. For a unit cost for basin construction of $15,000 per acre the plant construction cost was $150,000, equal to available finance. This $15,000 marginal cost per acre translates to $20 per lineal foot of basin peripheral length, which is quite inadequate for construction of a walled basin (such as by a ring of sheet piling), and sufficient only for the wave protection around the waterline of an earthworks basin. An attempt to design for minimum construction cost (rather than minimum total cost as in the preceding analysis) reduced construction cost by less than 5%.

The minimum technically feasible effluent copper concentration is thought to be 0.2 mg/L total copper at a basin pH of 11, based on extrapolation of pilot plant experience showing approx 0.5 mg/L of total copper with a basin pH of 10.2. The minimum construction cost for a plant to produce effluent containing 0.2 mg/L of total copper is thought to range from $600,000 to $1,300,000, for a 2.2 acre basin surrounded by a sheet piling wall costed at $200 to $500 per lineal foot. To meet the Basin Plan limit of 0.01 mg/L of total copper, this effluent would require 20-fold dilution; limited data suggest that 10-fold dilution is typically available directly at the point of discharge, and a greater ratio further downstream.

Wet season data suggest that seepage from the tailings near the mine may contribute 50% of the copper load discharged by the mine itself, although this would be reduced if the tailings were covered by a sedimentation basin.

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ACKNOWLEDGMENTS

Appreciation is expressed for constructive and supportive counsel by Larry Nash and by Pat Leary of Regional Board staff. Also acknowledged is crucial assistance by staff of Joe Nessler of the Beckworth office of the California Department of Water Resources, who provided Snocat transportation to the treatment plant site to rectify a critical pilot plant operating problem, after snow had blocked road vehicle access.

John Potter constructed the complete pilot facility as well as the Palmer-Bowlus flume, working with such energy and enthusiasm that others joined him at various stages of the work - Mr and Mrs Wold, George Wold, Tim Potter, Darren Potter and Stuart Foster.

Hugh McLean expertly performed atomic adsorption spectrophotometric analyses for metals, pointing out many pertinent features of the analyses.

Steve McBride, Project Engineer, of the Harrisburg, Pennsylvania firm of GEO-Technical Services, Inc., brought personal and corporate experience in mine drainage control to benefit design of the pilot plant and the diversion ditches.

Allan Pullinger brought professional engineering surveying skills to bear on design and layout for construction of the diversion ditches.

Vasil Diyamandoglu eased the time jam by working on as-built drawings of the pilot treatment facilities.

INTRODUCTION

Walker mine is located 22 miles north of the town of Portola at an elevation of 6,200 ft in the Sierra Nevada mountains, Plumas County, Calfornia. The mine is presently inactive; mining of copper peaked in the 1930s with an associated town of about 1,100 persons. Environmental effects of former mining activities persist however, notably sterile tailings of about 150 acres (not further addressed in this report), and stream pollution due to copper in drainage from the mine, that renders about 10 miles of stream devoid of aquatic life.

Water pollution abatement activities by the California Regional Water Quality Control Board, Central Valley Region, have proceeded from documentation of water quality degradation from 1957, through earlier attempts at abatement (limited for lack of effective technology), then a 1979 Engineering report by D'Appolonia, Inc., followed by the present program to design and implement abatement measures.

Diversion Ditches.- In July 1981 two ditches were constructed to divert surface runoff and intercept some groundwater from entering the mine at subsidence slumps created by collapse of overburden into mine workings (Fig. 1). Inspection of the ditches in September and October 1982 indicated that surface runoff during the preceeding winter of exceptionally high precipitation had been minimal from the mountainside as evidenced by scant erosion of that section of a ditch (Fig. 2) due to the porous nature of the volcanic ash soil. Accordingly, the mountain-side portion of the ditch in the Middle Branch of Ward Creek, that lies on U.S. Forest Service land, was not constructed. However, downstream of where each ditch intercepts the surface stream groundwater continues to be intercepted and diverted even through the dry summer. Considerable stream-flow must have been diverted during the 1982 spring thaw because a 5 ft deep channel was eroded in the South Branch ditch (Fig. 3), the Middle Branch

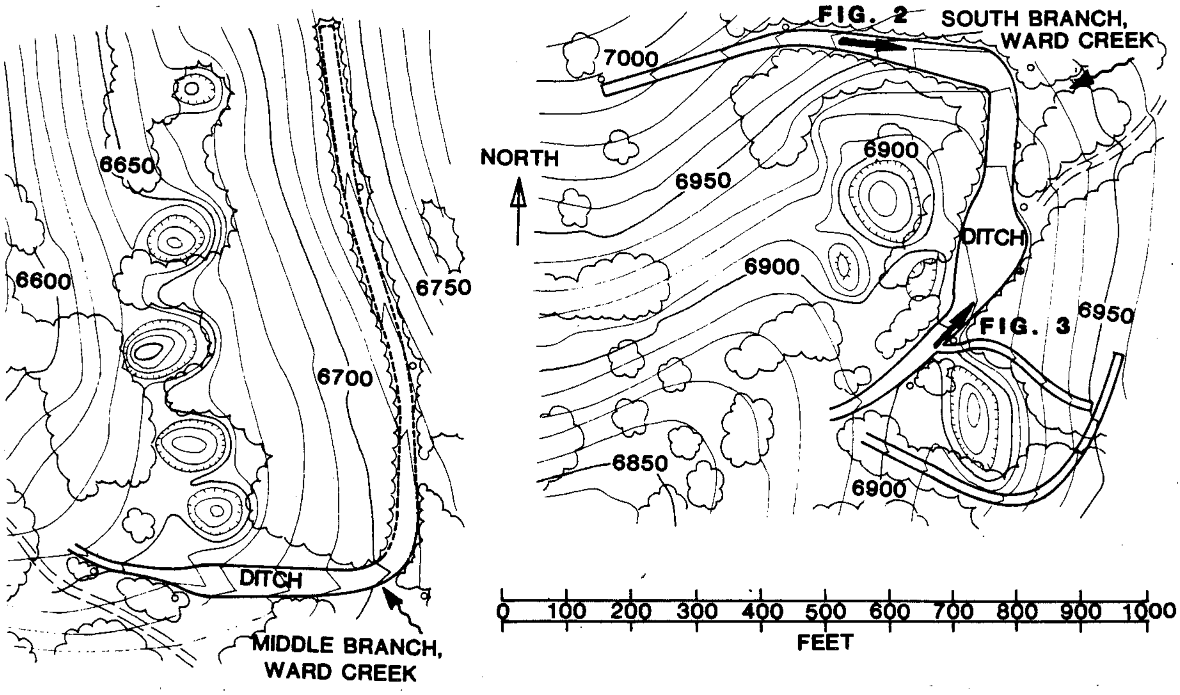


Fig. 1: Diversion Ditches Near Walker Mine. (Portion of Middle Branch ditch shown by broken line was not constructed) (1)

Fig. 2: Downslope on Uneroded Mountainside Section of Ditch, South Branch, Ward Creek.

Fig. 3: Upslope on Eroded Valley Section of Ditch, South Branch, Ward Creek.

being armored by boulders for its entire length (as is now the eroded section of the South Branch ditch). Soil eroded from the South Branch ditch must have been deposited on meadow beyond the end of the ditch, but was not conspicuous. The Regional Water Quality Control Board is considering grassing of the ditches.

Mine Drainage Flow Measurement.- In November 1981 a Palmer-Bowlus flume was completed in a 30 in. pipe carrying drainage from the mine, fitted with a cork dust peak level recorder to record the peak flow from the mine that occurs during the spring thaw, and equipped with electrodes set at 3 in. vertical intervals near the gauging section to provide for estimation of the flow-duration characteristics of the discharge by electrolytic integration of the time that each electrode is submerged (Fig. 4).

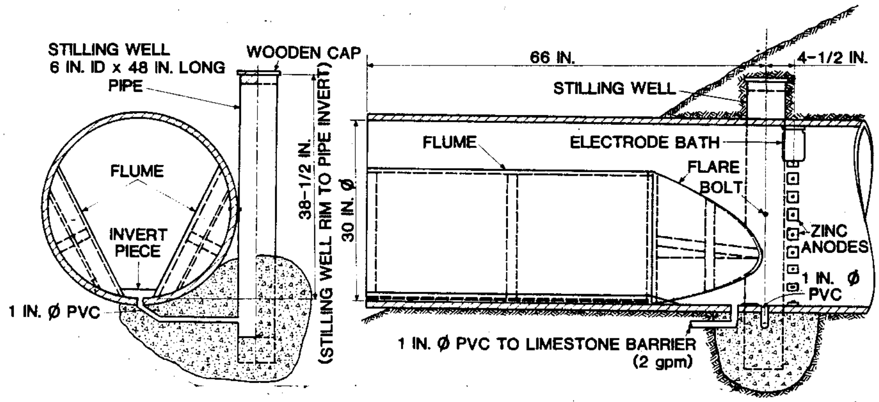


Fig. 4: Palmer-Bowlus Flume for Measuring Flow of Mine Drainage (2).

Over the 1981-82 winter a peak flow of 830 gpm was recorded. Due to fracturing of the glass electrode bath by freezing the flow-duration curve for this period could not be estimated, although an expected reduction of electrode weight change with increasing depth was noted. A plastic electrode bath with glass-fibre tape reinforcement was installed, and the meter set up to record flows for the 1982-83 winter.

Pilot Plant Treatment of Mine Drainage.- Walker mine drainage pollution abatement measures include treatment facilities to remove acidity and dissolved metals, notably copper. Available technology for removal of copper from mine drainage is limited to electrolytic stripping using sacrificial scrap steel anodes or an impressed current, and chemical precipitation by lime and/or sulfide for metal finishing industrial wastewaters (3).

To develop design and operating parameters for the Walker mine treatment facility pilot scale investigations were conducted, the pilot plant having been designed from results of bench scale tests and from available data and theory (4). Two constraints on design of both the pilot and prototype treatment facilities at Walker mine were the need to operate without electric power, and the need to operate unmanned from November to May when the final 16 miles of access road is blocked by snow.

The remainder of this report concerns construction and operation of the Walker mine pilot facilities during 1982; these facilities are at the time of writing operating unmanned over the winter and will be again reported on after reinspection in the spring of 1983. Facilities comprise a multi-process treatment plant, a static crushed limestone barrier, and a small sacrificial anodic copper stripping (cementation) unit. The treatment plant includes a water wheel-powered neutralization plant with two alternative mechanically agitated crushed limestone pre-neutralization units (tumbling drum and autogenous mill), followed by injection of soda ash and/or caustic soda to raise the pH to a level at which copper precipitates. Neutralized waste flocculates during passage through a pipe to a cascade (for stripping of any free carbon dioxide), followed by sedimentation in a basin followed by filtration through straw bales. Also, three evaporation ponds were installed, and the effect of Walker mine on the quality of the receiving water, Dolly Creek, was investigated.

DESIGN AND CONSTRUCTION OF WALKER PILOT MINE DRAINAGE TREATMENT FACILITY

Design Data.- Table 1 summarizes design data for the pilot facility (4).

TABLE 1: Pilot Plant Design Data

Neutralization/sedimentation/filtration plant	
Tumbling drum (limestone treatment unit) flow	60 gpm
Effluent pH (from pH 4 influent)	4.5 to 4.8
Autogenous mill (limestone treatment unit) flow	2 gpm
Effluent pH	6.3
Chemically neutralized waste flow	up to 60 gpm
Effluent pH (after limestone pre-neutralization)	up to pH 10
Length of pipe for flocculation after neutralization	240 ft
Height of cascade for stripping free carbon dioxide	15 ft
Sedimentation pond surface area (54 ft x 28 ft)	1500 sf
Surface overflow rate at 60 gpm (2 gpm)	60 (2) gsfd
Mean depth at 60 gpm (2 gpm)	1.2 (1.0) ft
Detention time at 60 gpm (2 gpm)	3.7 (94) hr
Weir loading at 60 gpm (2 gpm)	1630 (54) gfd
Straw filter area (at 2 in. water depth)	4 sf
Filter loading at 60 gpm (2 gpm)	22000(700)gsfd
Limestone barrier	
Flow treated	2 gpm
Length of barrier	500 ft
Effluent pH (from pH 4 influent)	6.3
Copper cementation unit	
Flow treated	up to 2 gpm
Sacrificial anode electrolytic current density	0.15 amp/m^2
Evaporation ponds	
Fresh water, raw AMD, neutralized AMD; area each	110 sf

Effort.- Table 2 summarizes the work schedule, person-days and expenses.

TABLE 2: Person-Days Expended and Expenses Incurred

Item	Person-days		Expenses	
	Amt	Total	Amt	Total
Design, drawings (Sept 1981 - May 1982)		32.9		1012
Construction (Mar - Sept 1982)				
Obtain crushed limestone	3.5		878	
Neutralization plant: concrete	3.0		121	
steel	15.4		1929	
chemical pumps	-		510	
Settlement pond and column	8.6		1667	
Straw filter and evaporation ponds	1.0		459	
Limestone barrier	7.0		175	
Copper cementation unit	0.5		0	
Pipework	5.1		1082	
Travel and subsistence	-	44.1	2064	8007
Operation and reporting (Sept 1982-Mar 1983)				
Operate, monitor, adjust, modify	39.0		846	
Travel and subsistence	-		3288	
Bench scale neutralization tests	2.0		0	
Metals analysis by AA (max)	22.0		580	
As-built drawings	5.5		92	
Data analysis, report preparation	27.0	95.5	700	5506
Totals		172.5		$14,525

The total cost of work described in this report approximates 95% of the budget for pilot plant investigations, with the spring 1983 reinspection and report remaining to be completed. Several factors favored effective and economical construction and operation of Walker mine pilot facilities:

- The mine owner, Mr. Robert Barry, facilitated access and permitted use of the mine for storing materials during construction and operation;
- Conoco, Inc. provided contour maps of the vicinity of Walker mine (200 ft to 1 in., 10 ft vertical interval) that expedited site investigations both at the mine and at the diversion ditches;
- Concrete foundations existing at the site provided a convenient base for construction of the neutralization plant, sedimentation basin and evaporation ponds, and for fastening pipework;
- Having mine drainage previously piped to the treatment site, and the means for control of the flow to the pilot facility with relative ease, minimized the expense of delivering mine waste to the treatment site;
- A streamflow gauging station in Dolly Creek that had been installed by Regional Board staff provided an unexpected opportunity for mass balancing of copper and other metal pollutants at point sources at the Walker mine, and for estimating the loadings of certain metals in Dolly Creek that appear to result from seepage from tailings in the wet season;
- No significant vandalism or theft of materials was experienced during construction or operation. Indeed, the interest of visitors to the mine in the work was most gratifying; and
- Staff of Mr. Joe Nessler of the California Department of Water Resources, Beckworth, provided indispensible support and transportation by Snocat to the mine in December 1982, for the purpose of rectifying an operating problem that would otherwise have precluded winter operation of the plant.

The major problem was heavy snowfall during the 1981-82 winter that blocked access to the mine until June 1982. Late construction of the evaporation ponds limited evaporation data obtained. However, the need to extend operation of the pilot facility into the fall 1982 wet season to obtain required operating data provided wet season operating experience invaluable in design of the prototype facility.

Construction.- Figures 5 - 12 are as built drawings of the pilot plant and limestone barrier. These drawings include modifications to the original design and construction that were implemented during operation as a result of operating experience.

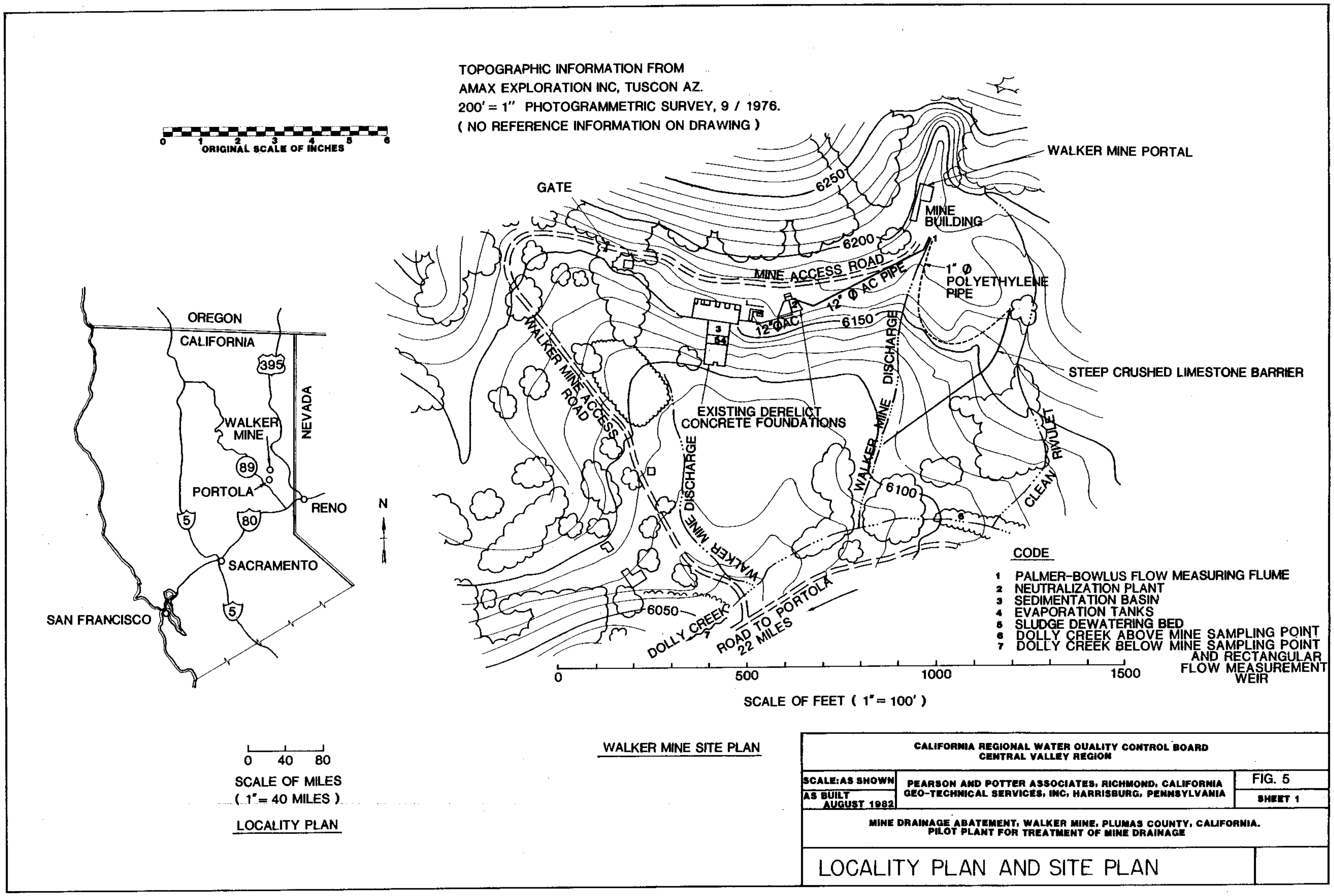


Fig. 5: Locality Plan and Site Plan (As-Built Construction Drawing Reduced to 25% Size).

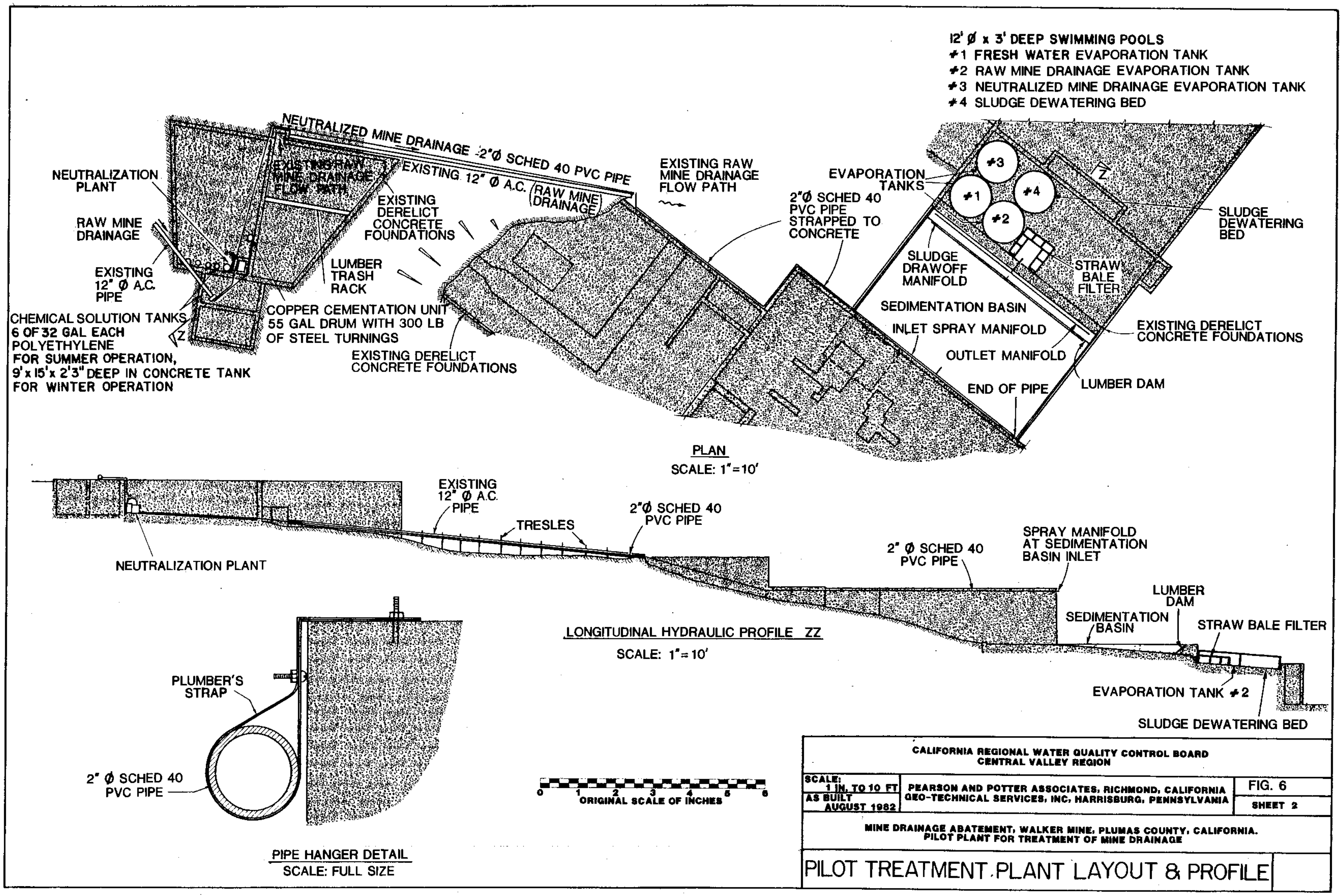


Fig. 6: Pilot Treatment Plant Layout and Profile (As-Built Construction Drawing Reduced to 25% Size).

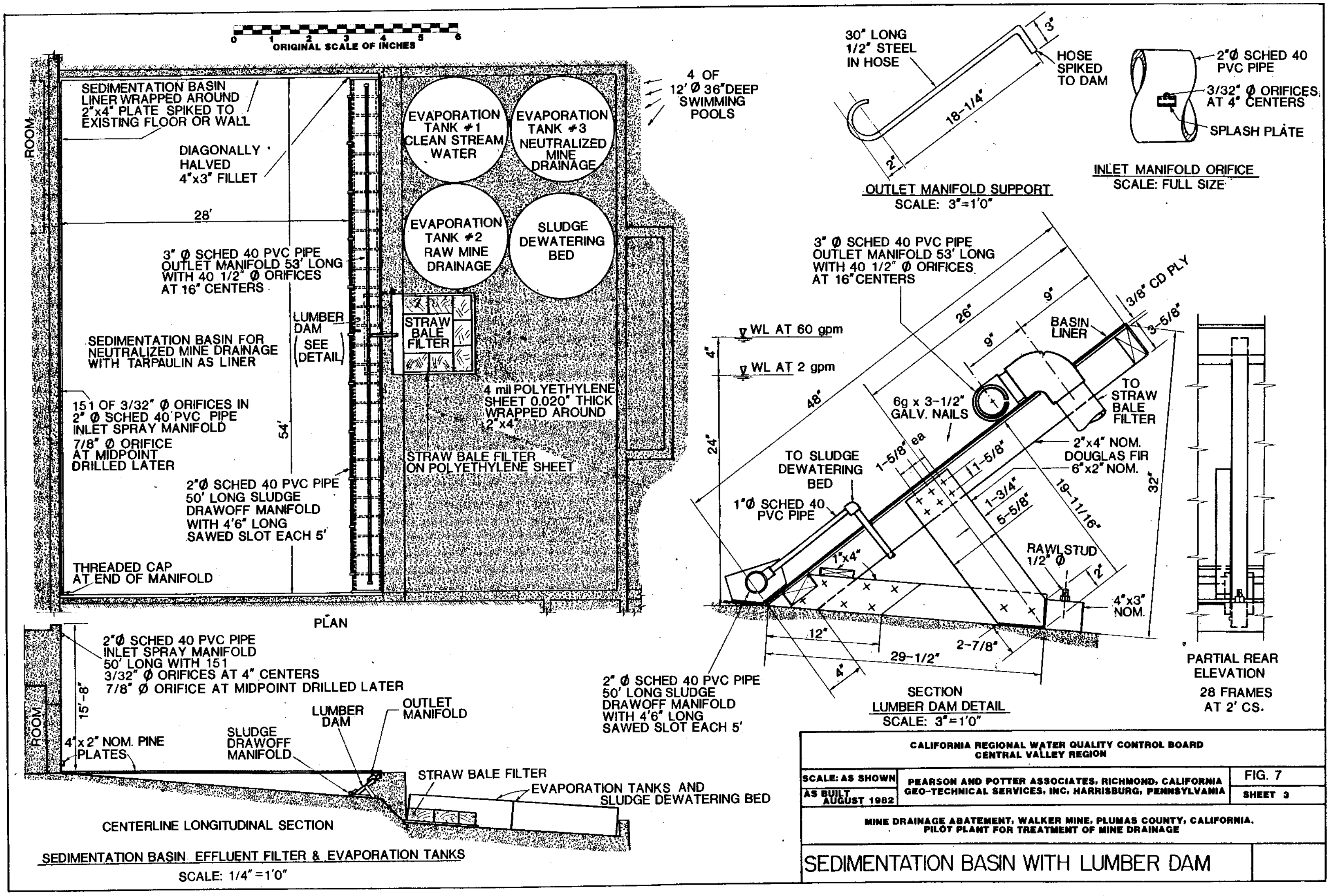


Fig. 7: Sedimentation Basin With Lumber Dam (As-Built Construction Drawing Reduced to 25% Size).

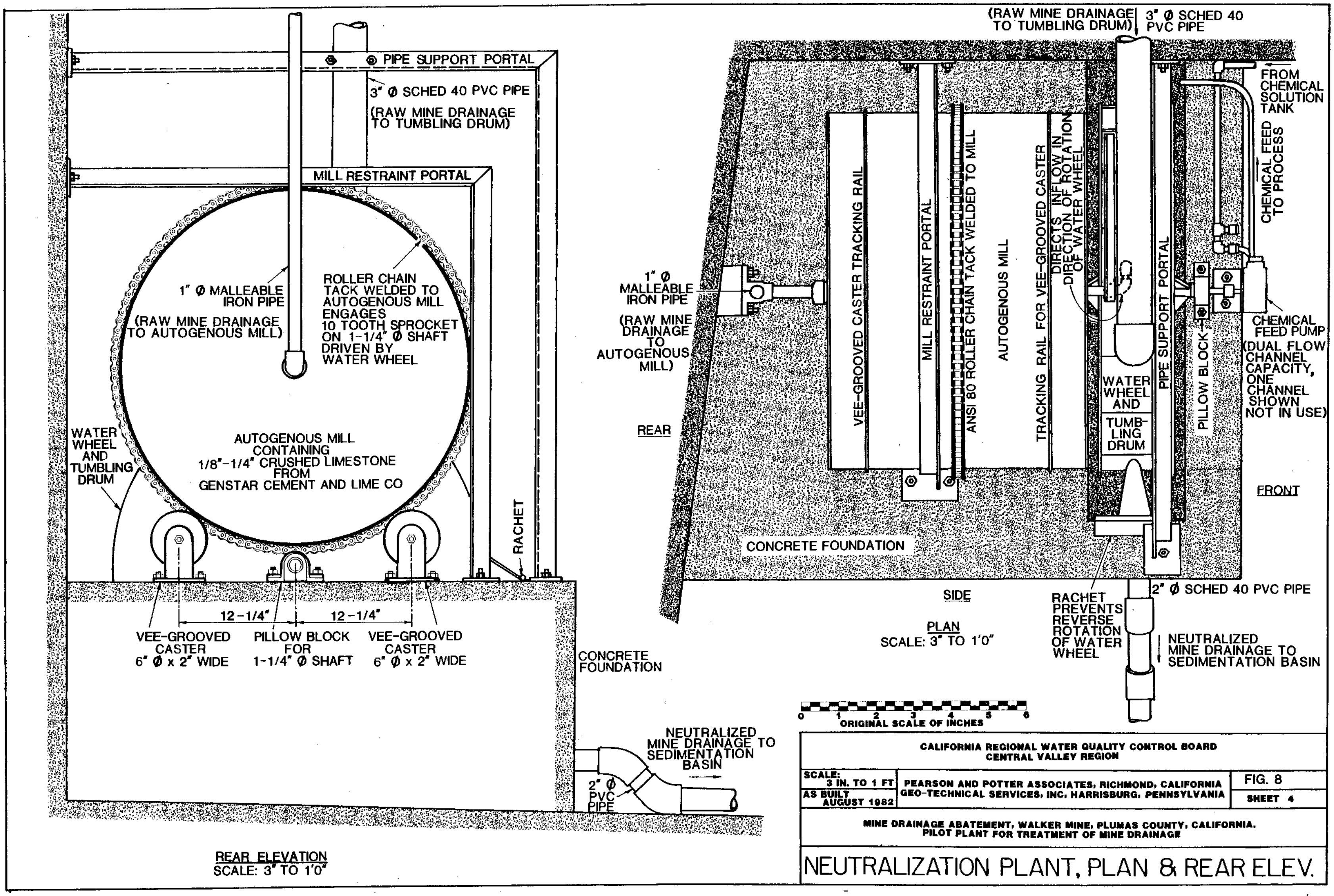


Fig. 8: Neutralization Plant, Plan and Rear Elevation (As-Built Construction Drawing Reduced to 25% Size).

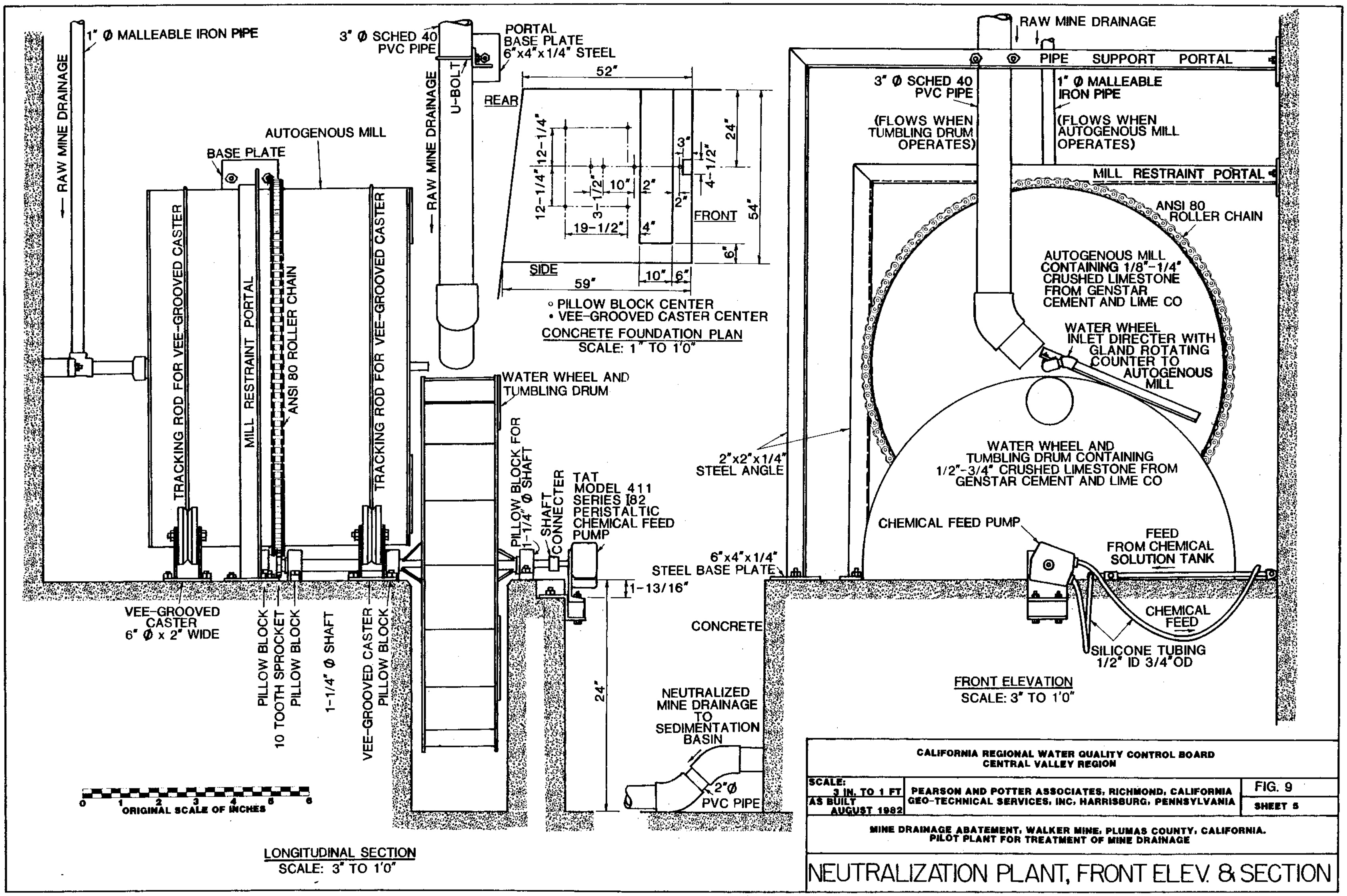


Fig. 9: Neutralization Plant, Front Elevation & Section (As-Built Construction Drawing Reduced to 25% Size).

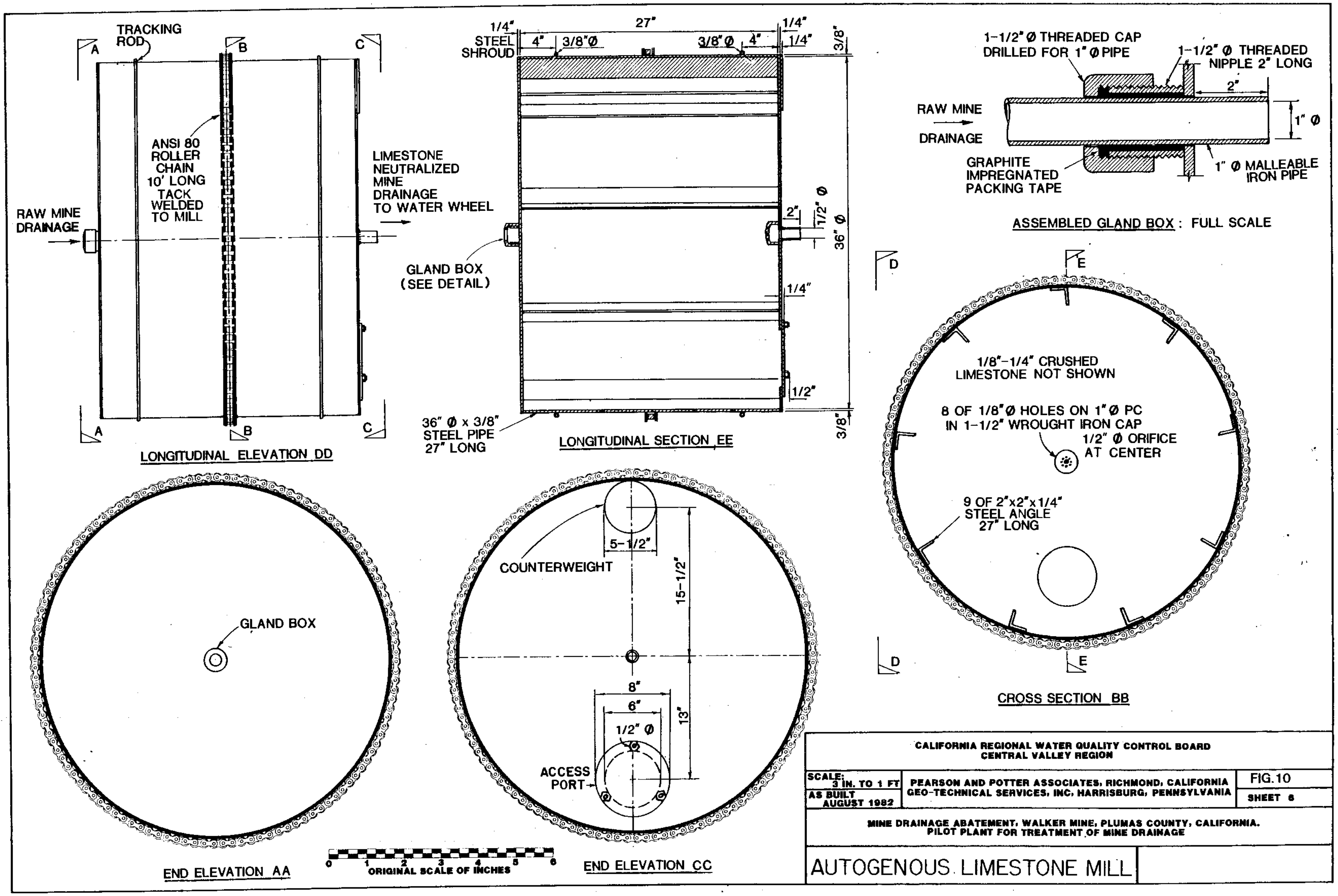


Fig. 10: Autogenous Limestone Mill (As-Built Construction Drawing Reduced to 25% Size).

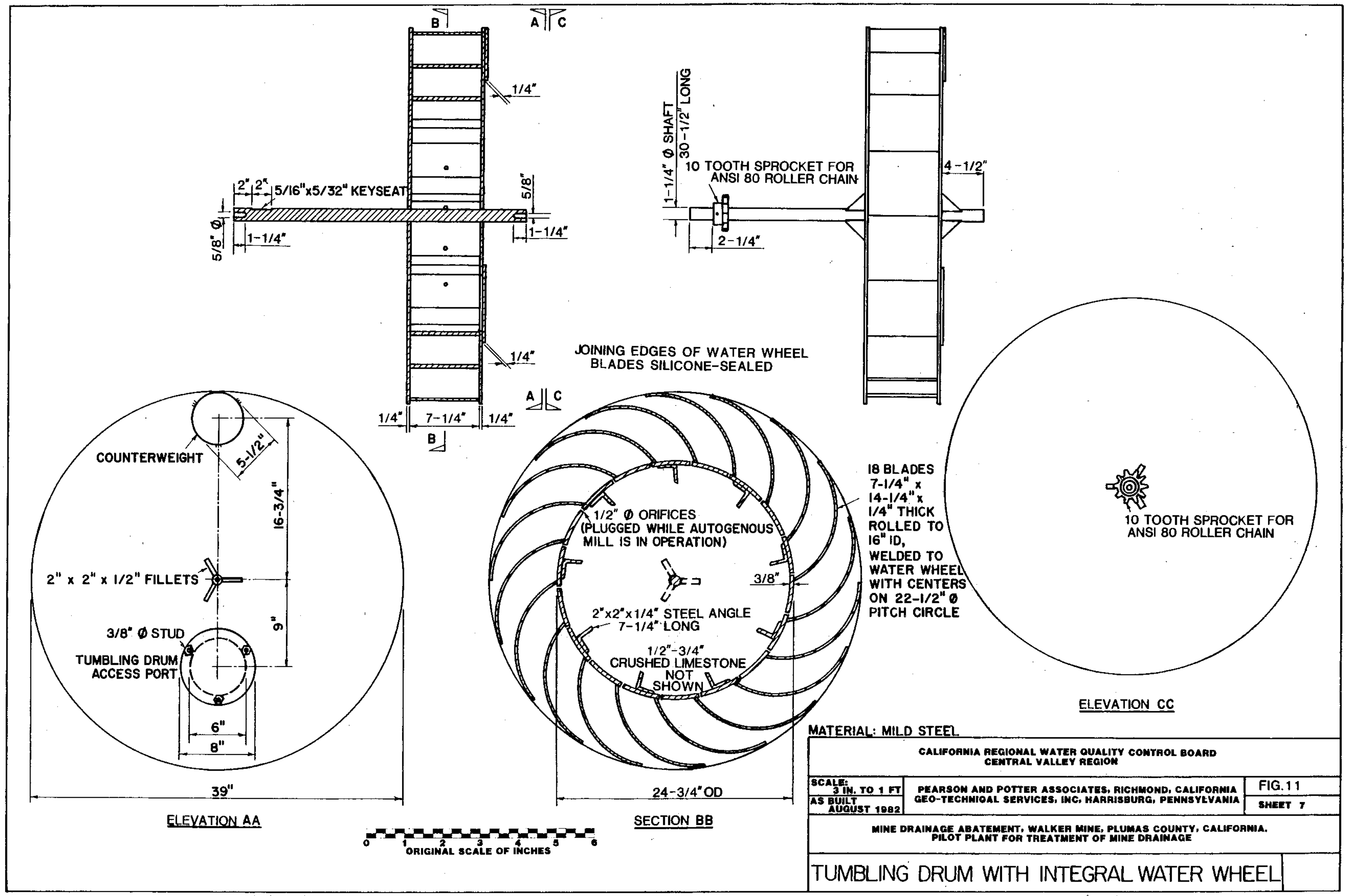


Fig. 11: Tumbling Drum With Integral Water Wheel (As-Built Construction Drawing Reduced to 25% Size).

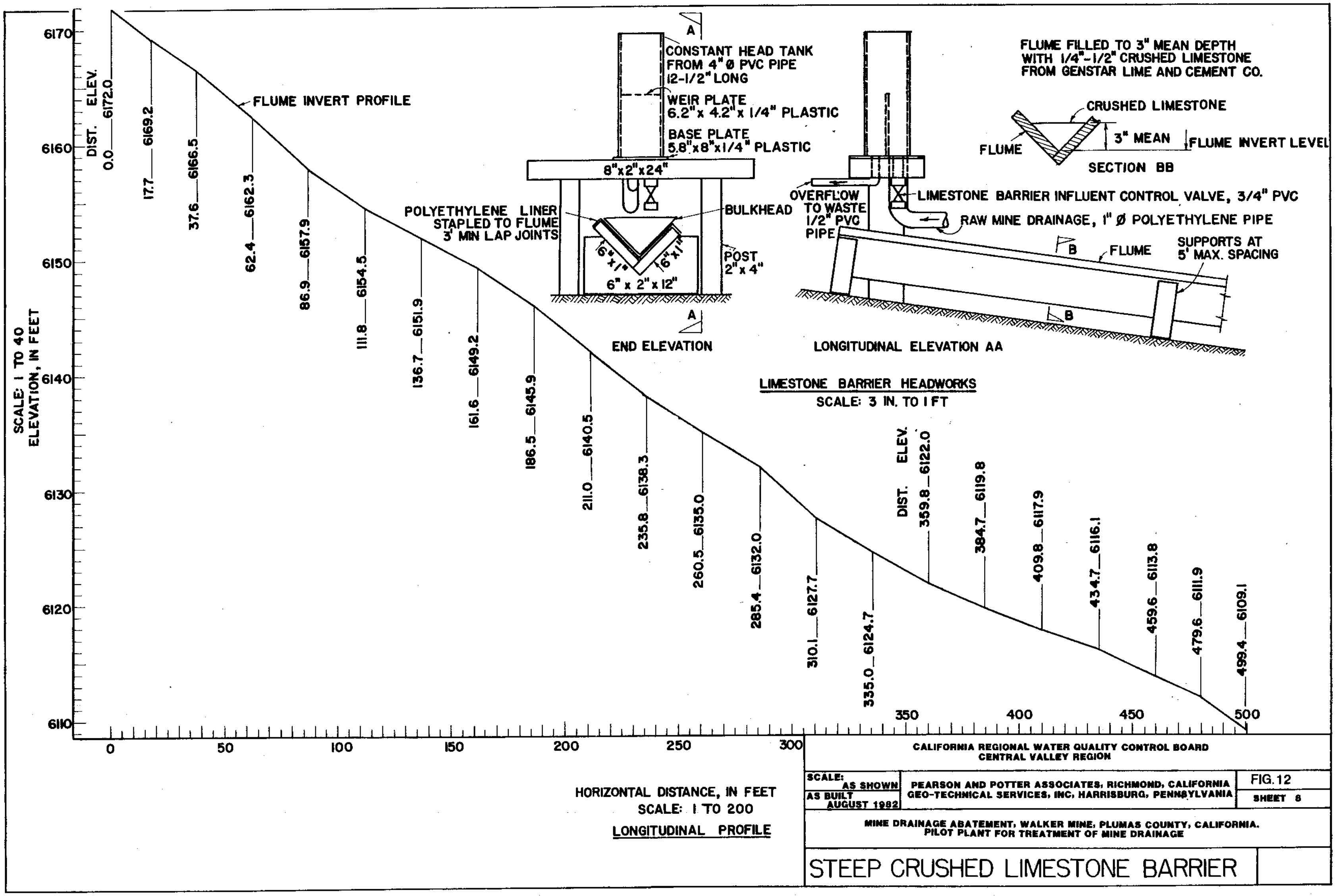


Fig. 12: Steep Crushed Limestone Barrier (As-Built Construction Drawing Reduced to 25% Size).

OPERATION OF PILOT TREATMENT FACILITY

Operating Period.- Over 97 operating days (9/2-12/7/82) the pilot facility monitored 42 times. Table 3 lists operating periods for each process unit.

TABLE 3: Process Unit Operating Periods and Monitoring Frequency

Process	Flow range, gpm	Days of continuous operation	Number of times monitored	In 1982-83 winter operation?
Pilot plant				
Tumbling drum	22-64	14[a]	11	No
Autogenous mill	0.17-2.8	44[b]	26	No
Soda ash neutralization	22-32	4	5	-
Soda ash neutralization	0.17-2.8	31	14	Yes
Caustic soda/soda ash neutlzn	0.74-6.9	24[c]	14	No
Sedimentation and filtration	0.17-64	97	42	Yes
Limestone barrier	0.32-5.1	93[d]	41	Yes
Copper cementation unit	0.024-1.5	47	8	No

[a]Drum operation terminated when first load of limestone was depleted.
[b]Mill operation pre-tested without limestone for an additional 13 days.
[c]Discontinuous chemical neutralization for an additional 23 days.
[d]Includes period of zero barrier outflow due to freezing, up to 21 days.
[e]Operation of copper cementation unit was discontinuous.

Operational Tasks.- Operational tasks may be grouped as follows:

- Plan program and make needed adjustments to flows and other conditions;
- Monitor conditions, and test performance of process units;
- Determine and implement needed changes in process units or pipework; and
- Prepare chemical solutions, load limestone, other housekeeping and repairs.

Table 4 lists the normal monitoring schedule for operating units.

TABLE 4: Normal Schedule of Monitoring and Sampling

Monitoring point	Flow	pH[a]	Copper[b]	Sett.[c]	Metals sample[d]
Raw mine drainage	X[e]	X	X		X
Tumbling drum effluent[f]		X	X		
Autogenous mill effluent		X	X		
Chemical-neutralized effluent		X	X	X	
Sedimentation basin effluent	X[g]	X	X	X	
Straw filter effluent		X	X	X	X
Limestone barrier, 23% length		X	X		
Limestone barrier effluent	X[g]	X	X		X
Copper cementation effluent	X[g]	X	X		
Dolly Creek above mine		X	X		X
Dolly Creek below mine	X[e,h]	X	X		X

[a]Using Lazar DPH-2 pH meter with glass electrode buffered at pH 4, 7 and 10.
[b]Using Standard Methods (5) cuprethol field colorimetric procedure.
[c]Settleable solids, measured in Imhoff cone at 60 min, by Std. Methods (5).
[d]Sample + 0.2 mL nitric acid/100 mL stored for metals analysis by AA.
[e]Flow measured in gauging flume.
[f]Drum effluent could not be monitored with chemical neutralization operating.
[g]Flow measured by bucket and stopwatch method.
[h]Flow in Dolly Creek measured only for latter third of program.

Where the operation of units was terminated reasons were as follows:

• Tumbling drum:	Performance deteriorated with time.
• Autogenous mill:	Performance deteriorated with time.
• Soda ash neutralization:	Not terminated; in winter operation.
• Caustic soda neutralization:	Freezing of 50% caustic soda in drum.
• Sedimentation basin:	Not terminated; in winter operation.
• Straw filter:	Ineffective, but retained in winter operation.
• Limestone barrier:	Not terminated; in winter operation.
• Copper cementation unit:	Effective only at low loading.

Methods for improving the performance of terminated units are presented later.

Operating Problems.- Problems experienced in operation of pilot units not terminated for inadequate performance require consideration in design of the prototype facility. These problems were as follows:

- Freezing of Caustic Soda.- A 55 gal drum containing about 500 lb of 50% caustic soda froze when stored in the mine in mid-November. Although a low night temperature of -10°C was recorded at this time, it appears unlikely that the mine was much cooler than 0°C. Heating of outdoors storage tanks and pipelines containing 50% caustic soda, is normally recommended (6), but would require solar panels at Walker, introducing complexity and vulnerability to chemical storage facilities.

  Less concentrated solutions of caustic soda freeze below 0°C, e.g., -17°C for a 14% solution (7), which appears sufficiently low to avoid freezing at Walker. Based on a winter mean daily sea level temperature of 5°C (41°F), and an adiabatic lapse rate for dry air of 3°C/1000 ft (8), the winter mean daily temperature at the 6,200 ft elevation at Walker is estimated as 5 - 3 x 6.2 = -14°C (8°F). For moist air the temperature drops slightly less with increasing altitude. A minimum recording thermometer is presently installed at Walker.

- Low Solubility of Soda Ash.- Soda ash dissolves to a maximum concentration of 7.1% by weight at 0°C (7), although various hydrates of sodium carbonate are listed as more soluble, e.g., 21.5% at 0°C for the decahydrate, washing soda. Soda ash powder (anhydrous) used at Walker could be dissolved to a maximum concentration of 5.5% in water at 6°C. Vigorous agitation was needed to attain this strength, obtained by trickling a fine stream of soda ash into a jet of water from a one inch hose discharging into a series of nested buckets on a bench in the solution tank, stopping frequently to break up and dissolve the cake that formed in the buckets, a task taking almost one hour per 100 lb of soda ash. If some form of sodium carbonate is to be considered for use in the prototype Walker plant alternative mixing procedures might be investigated, such as using a boat outboard motor in the solution tank, or testing the suitability of washing soda which may dissolve to a greater strength, requiring a smaller solution tank than for soda ash. The freezing point of a 5.5% solution of washing soda is -2°C (7).

- Corrosivity of Caustic Soda.- Even tiny splashes of caustic soda not immediately washed from protective clothing can produce skin (and eye) burns, making handling this material both tedious and hazardous, a particular consideration in a remote area where a "macho" attitude of an operator may lead to injury. Soda ash solution does not burn the skin.

- Low Flow Stalling of Water Wheel.- The water wheel driving the tumbling drum, autogenous mill and chemical feed pump rotated at a rate varying directly with the flow of water passing through the wheel over a range of perhaps 2-20 gpm under constant resistance torque. However, below about 2 gpm the wheel did not rotate at a constant rate but in sudden movements of about one-third revolutions, stalling between movements. At each burst of rotation, all water wheel buckets containing water emptied in such a short time that insufficient water entered buckets at the top of the wheel to sustain rotation. On the other hand, a steadily rotating wheel discharged water at the same rate that water entered the wheel.

  During low flow intermittent rotation buckets are filled in order from the top by spillage from the next higher bucket as the wheel remains stationary. Only those buckets for which the rim is above the axis of the water wheel can be filled by spillage, in contrast to the situation under steady rotation wherein all concave-upwards pools can fill and contribute to the driving torque (Fig. 13).

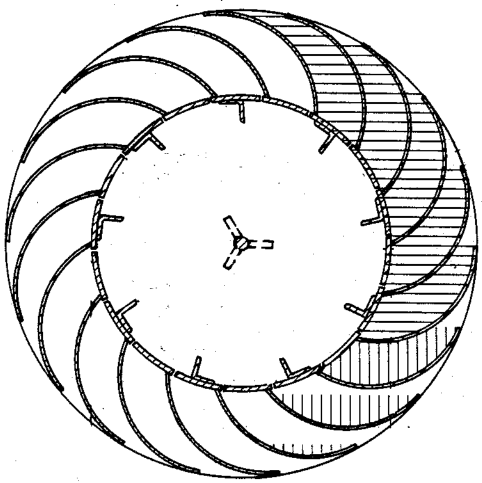


Fig. 13: Water Wheel Cross-Section Showing Water Pools for Intermittent Rotation (horizontal shading only) and Continuous Rotation (horizontal plus vertical shading).

  When an intermittently rotating water wheel is stationary, a rachet is needed to prevent reverse rotation under the reverse torque exerted by a tumbling drum or autogenous mill. Rotation occurs when sufficient buckets fill to overcome this reverse torque. The hydraulic efficiency of the water wheel is less for intermittent rotation than continuous rotation, as is the driving torque unless the wheel is fitted to direct spillage to the bucket below even when the rim of the receiving bucket is below the horizontal through the wheel axis. Water wheel hydraulic design procedures (9) have not taken into account the loss of efficiency and torque with intermittent rotation, and load had to be removed from the Walker pilot water wheel to remedy stalling and produce continuous rotation for winter operation.

- Life of Peristaltic Feed Pump Tubing.- In peristaltic pumps, flexible tubing containing the liquid to be pumped is locally closed by sqeezing under a roller that moves forward along the tubing, pressing the liquid ahead. One eighth inch wall thickness silicone tubing used in the pilot plant chemical feed pump has a rated life of three million revolutions, or one year at 5 rpm (10). At such slow speeds fine clearances are essential; a flexible impeller pump (Jabsco manufacture) was found to have too large a clearance between the side of the impeller and the pump body to move the liquid.

- Corrosion of Steel.- Corrosion was superficial over the three month reporting period. However, design of the water wheel will be modified to minimize splashing on the shaft and bearings.

- Occlusion of Orifices.- The 3/32 in. orifices in the sedimentation basin inlet manifold clogged fairly rapidly (about 10% per day after startup) with particles of rotted mine timbers, beetles, moss and leaves, sized up to 1/2 in. A 7/8 in. orifice was drilled, which also appears likely to alleviate icing up of the nozzles.

- Solids Separation.- Operation of the treatment plant showed solids separation to limit effluent quality, lower copper residuals being observed at lower sedimentation basin hydraulic loadings. Design of an adequate sedimentation basin, safe against washout and piping of sand, presents the major prototype design problem at Walker. Sand filtration appears infeasible at this site, and the straw filter did not significantly change the concentration of copper from that in sedimentation basin effluent.

- Sludge Handling and Disposal.- The 1:10 base slope of the pilot sedimentation basin was inadequate to gravitate sludge to the drawoff manifold, although sludge did not accumulate appreciably on the 1:1.4 vertical:horizontal face of the dam. Although an attempt to dewater sludge on native sand did not produce any cake of sludge on the sand surface, gravity thickening and evaporation were found effective. About 2 acre-feet of thickened sludge per year is likely to be produced (which requires consideration in design of the sedimentation basin), that evaporates at about the same rate as water to an estimated 30 tons of dry cake per year for disposal in slump depressions if not marketable (25% copper).

- Freezing of Limestone Barrier.- At a night temperature of -10°C, mine drainage initially at 5°C froze in the limestone barrier after flowing 350 ft (estimated 25 min flow time), resulting in zero discharge from the barrier. Two weeks later, with snow covering the barrier, flow resumed. Freezing is not expected to be a critical problem in a prototype barrier at Walker, because a large barrier cross-section necessary for treatment of the high spring flow provides at lower mid-winter flows a depth of dry stone above the water surface for supporting insulating snow.

- Seepage From Mine Tailings.- Wet season copper residuals in receiving streams below Walker mine may be governed largely by the load of copper that appears to originate in seepage from the tailings at the mine site. Mass balancing revealed significantly higher copper loads in Dolly Creek below the mine than contributed by the mine discharge plus that in Dolly Creek above the mine. Arrangements to intercept in the sedimentation basin seepage from tailings above the basin should aid in abatement. Alkaline waters seeping through the floor of the basin may fix copper in tailings downslope of the basin. Otherwise, restoration of the tailings may be needed to sufficiently reduce copper loads in receiving waters.

PILOT PLANT PERFORMANCE

Outline Description of Plant.- In an earlier effort at abatement of copper pollution from the Walker mine, mine drainage had been piped to a concrete tank in the mine ruins that contained scrap steel for sacrificial anodic stripping of copper from the water. Although this process was subsequently abandoned, its legacy of a supply of mine drainage to a suitable site for the pilot plant proved fortuitous for this work. Mine water was drawn from the plugged pipe, and the tank was used for chemical storage during unattended winter operation of the pilot plant.

Mine water enters a water wheel-powered neutralization plant that has two stages of treatment, first pre-neutralization by crushed limestone, then complete neutralization by chemical dispensed by feed pump. Plant performance was evaluated over two flow ranges, using the tumbling drum limestone process at high flows near 30 gpm, or the autogenous mill limestone process at low flows near 2 gpm. The mill is more intensive, and attains a higher effluent pH than the drum. Chemical feed is automatically proportioned to the rate of flow of mine drainage to be neutralized, within limits, because the water wheel powered chemical feed pump rotation rate depends on flow.

Neutralized mine drainage flocculates in a 240 ft pipe leading to a cascade that discharges to the sedimentation basin, the purpose of the cascade being to investigate the efficiency of stripping free carbon dioxide that otherwise contributes to the acidity of the water. Settling column studies were conducted on the site to supplement monitoring of the sedimentation basin. Settled effluent passes through a filter constructed of straw bales. Evaporation ponds and a sand bed for studying dewatering of sludge removed by settlement were also provided.

Figures 14 to 32 are photographs of the pilot plant.

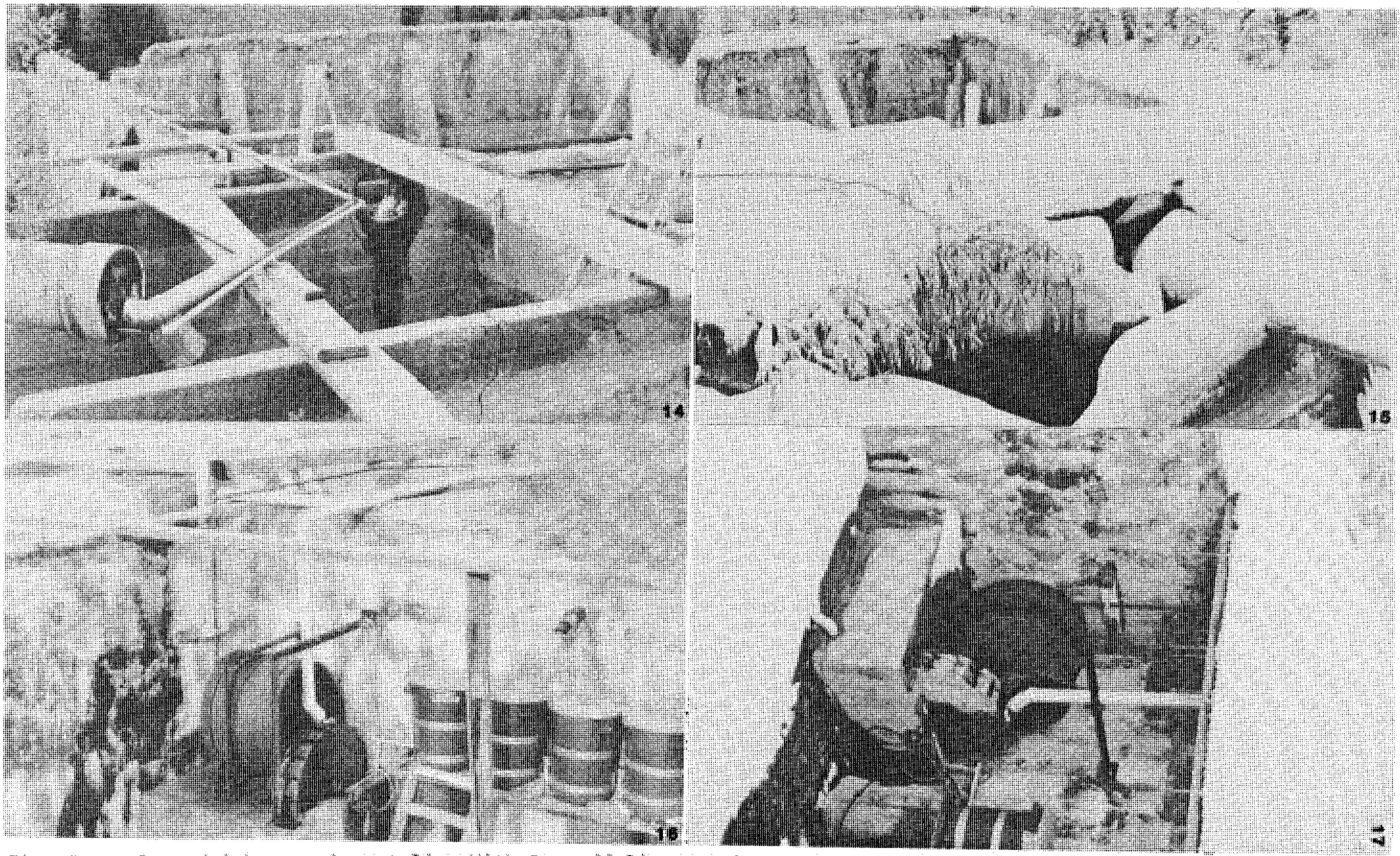
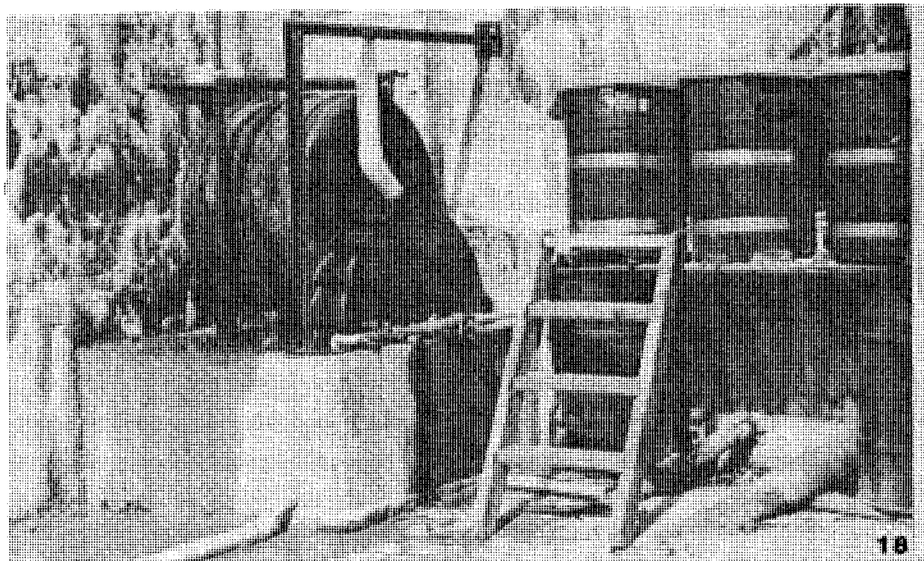
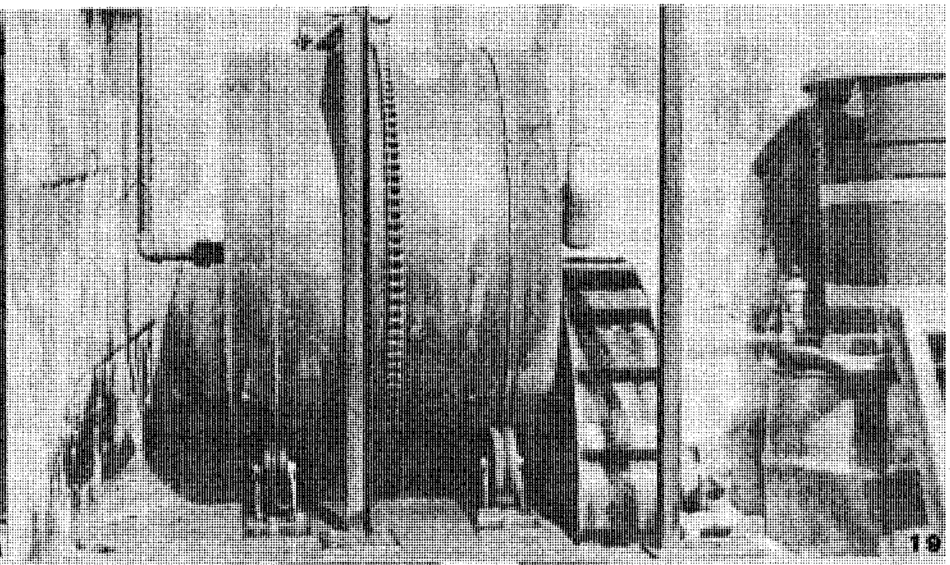
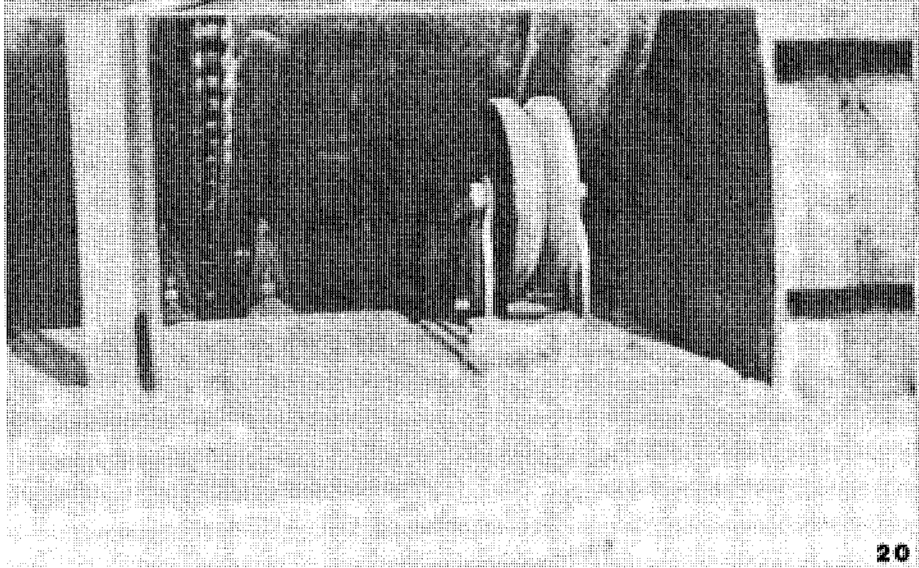


Fig. 14: Plugged Asbestos Cement Pipe With Drawoff Pipes to Process Units, Set Over Concrete Chemical Feed Tank.
Fig. 15: Partially Roofed Concrete Tank Containing Chemical Feed For Unmanned Winter Operation of Pilot Plant.
Fig. 16: Neutralization Plant With Chemical Feed Tanks for Summer Operation, Showing Roof Support Beam.
Fig. 17: Roofed Neutralization Plant Under Mid-November Snow Benefits From a Southern Aspect For Warmth.







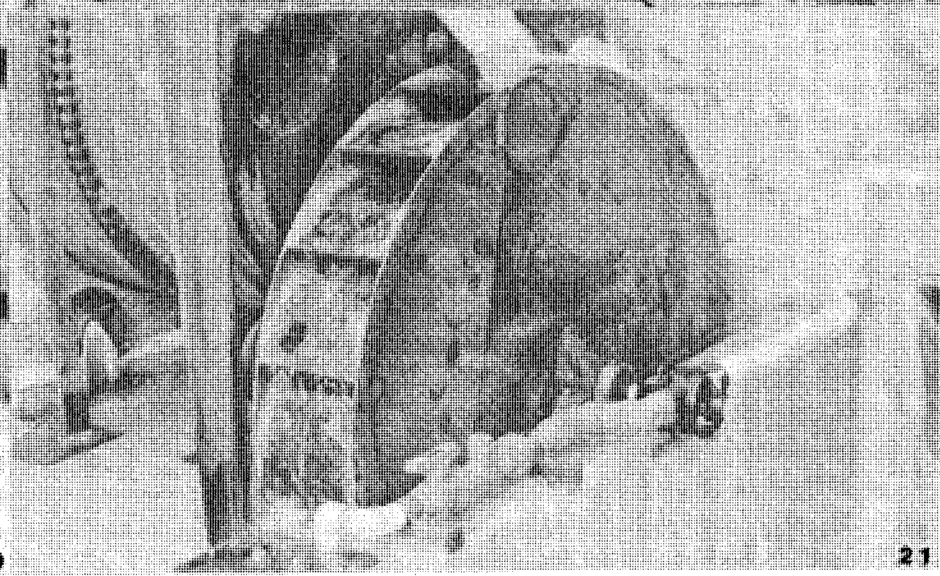


Fig. 18: Neutralization Plant, Oblique View.

Fig. 19: Neutralization Plant, Side View

Fig. 20: Autogenous Mill Support and Drive Detail.

Fig. 21: Water Wheel Drives Tumbling Drum At High Flow, Showing Jabsco Chemical Feed Pump In Service At This Stage.

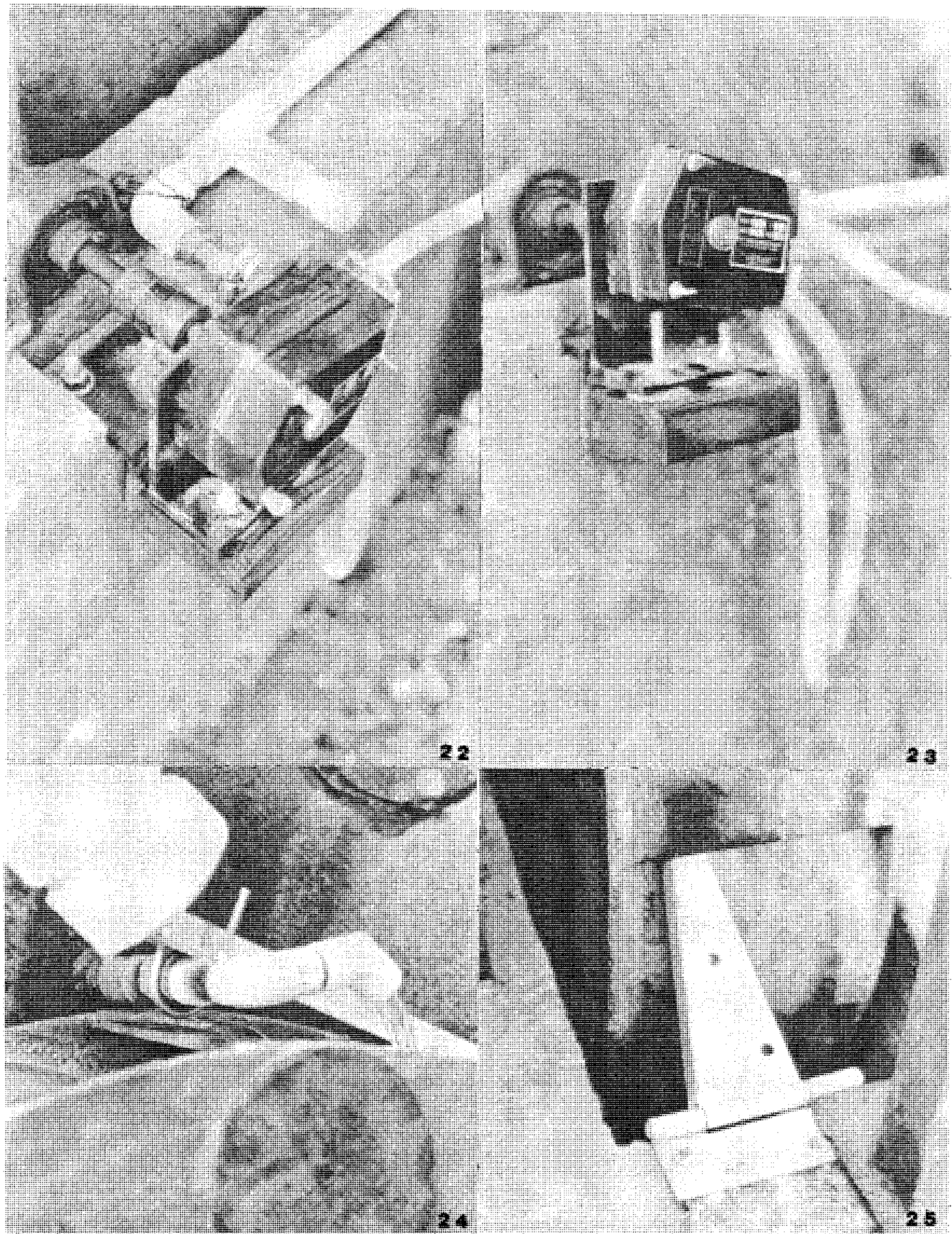


Fig. 22: Peristaltic Chemical Feed Pump Driven From Water Wheel, Top View.
Fig. 23: Peristaltic Chemical Feed Pump Fitted With Silicone Tubing, Side View.
Fig. 24: Autogenous Mill Effluent Drives Water Wheel That Rotates Mill.
Fig. 25: Rachet Prevents Reverse Rotation of Water Wheel at Low Flow.

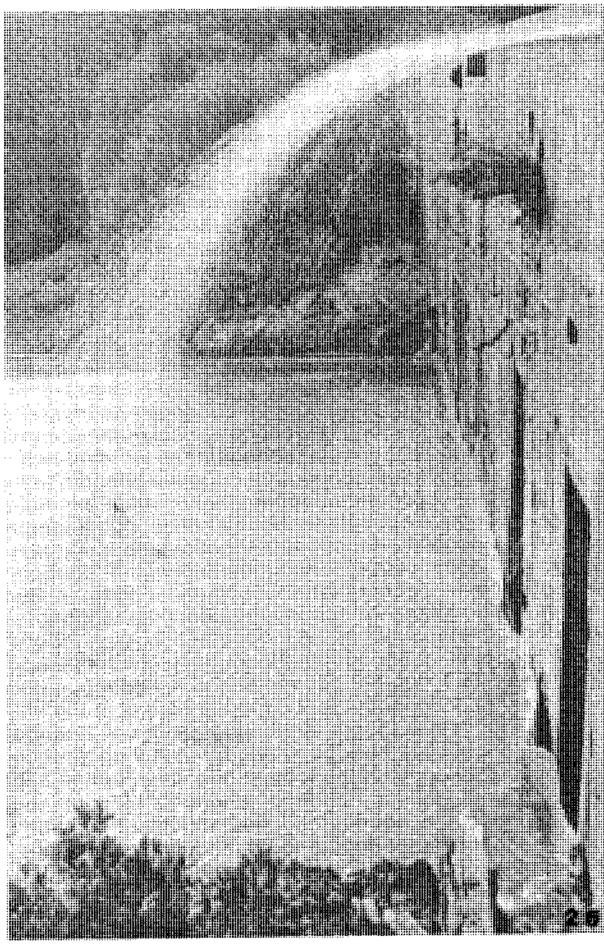


Fig. 26: Spray Manifold for Stripping Free Carbon Dioxide.

Fig. 27: Single 7/8 Inch Nozzle Discharges Into Sedimentation Basin, Flushing Sludge From An Area Of Floor.

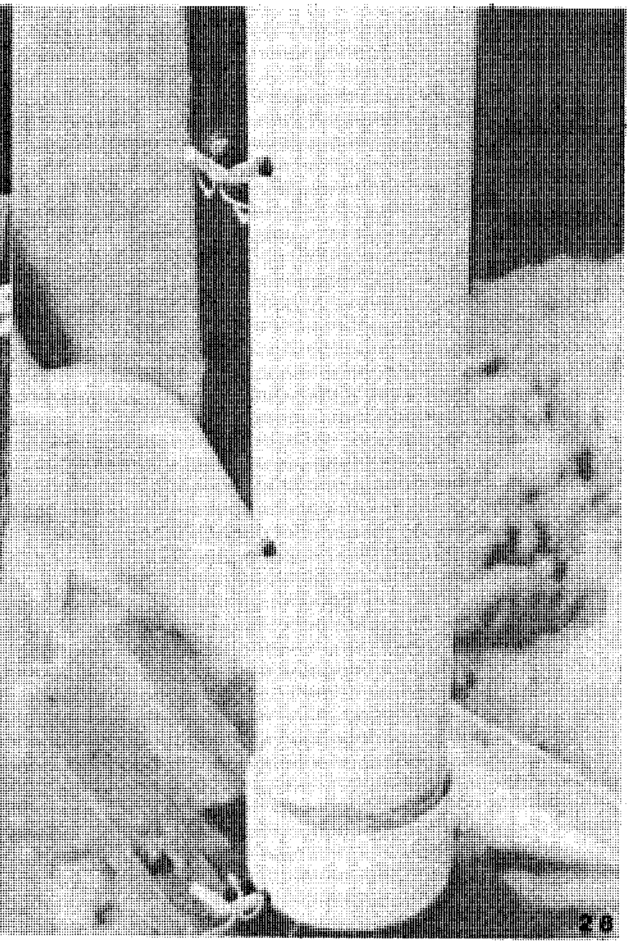
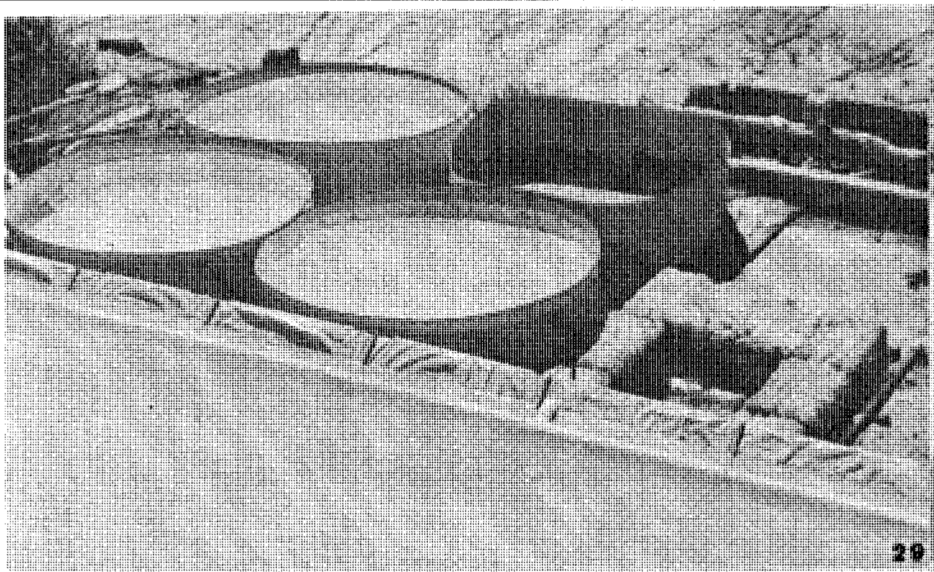
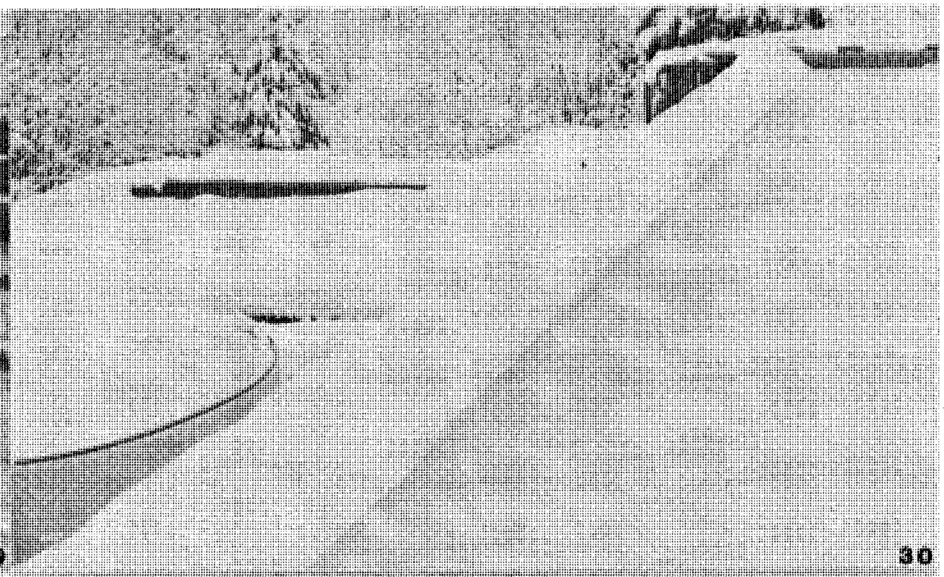
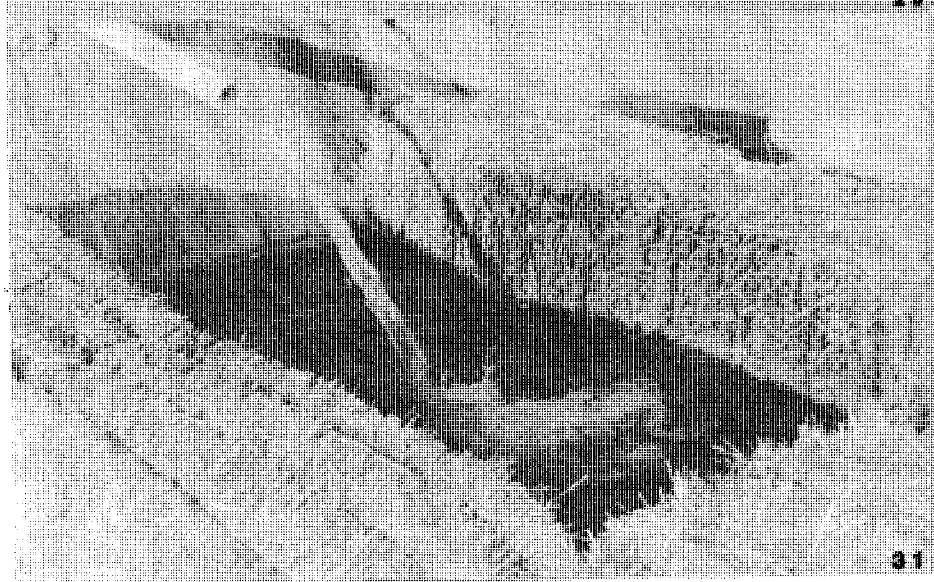


Fig. 28: Settling Column, Used in Static Batch and Continuous Flow Modes.







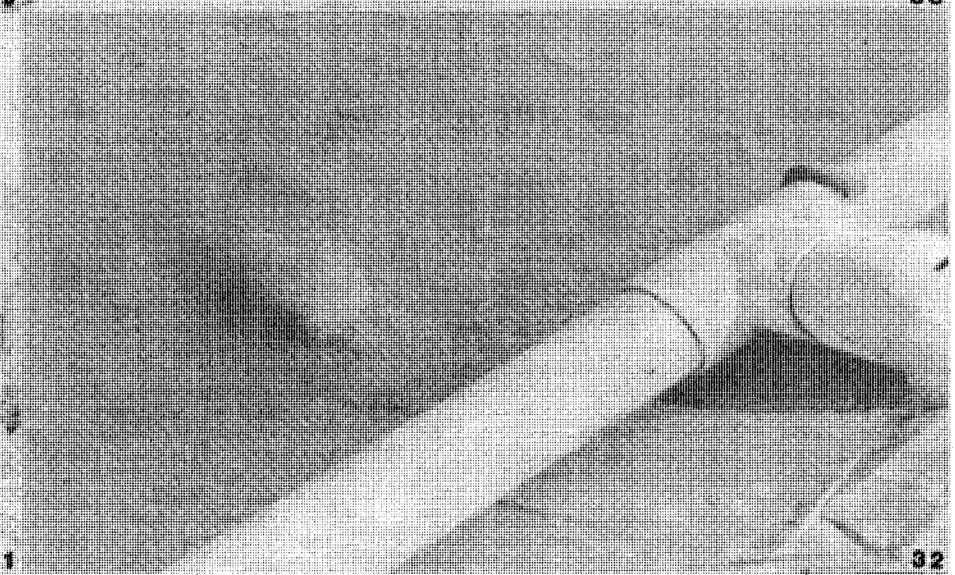


Fig. 29: Sedimentation Basin, Evaporation Ponds, Sludge Dewatering Bed and Straw Bale Filter.
Fig. 30: Sedimentation Basin Ice-Covered in Mid-November.
Fig. 31: Straw Bale Filter.
Fig. 32: Sedimentation Basin Outlet Manifold, Showing Sludge Drawoff Manifold Almost Inundated.

<u>Overall Performance of Pilot Plant</u>.- Table 5 summarizes the overall conditions of operation and removals of copper and other metals by the pilot plant in the two flow ranges investigated. At average flow rates of 28 gpm and 1.6 gpm, the overall removals of total copper were 72% and 93% (mean) respectively. Because the effluent pH was similar between high and low flow conditions, one can conclude that the efficiency of solids separation in the sedimentation basin limits overall process performance with respect to copper removal, rather than chemical precipitation resulting from neutralization.

TABLE 5: Summarized Overall Performance of Pilot Plant.

Parameter	High flow range				Low flow range			
	Mean	SD	Min.	Max.	Mean	SD	Min.	Max.
Operating period, days	4	-	-	-	57	-	-	-
No. of times monitored	5	-	-	-	24	-	-	-
Flow treated, gpm	28	4	22	32	1.6	0.8	0.2	3.0
pH increase, units	4.9	0.2	4.7	5.1	4.7	0.5	3.4	5.4
Effluent pH	9.9	0.3	9.4	10.2	9.6	0.6	8.3	10.3
Removal of metal, %								
Total copper	72	12	61	85	93	3	86	97
zinc	62	6	55	67	92	4	83	97
manganese	53	20	31	70	84	8	68	95
iron	11	74	-41	63	54	35	-23	76
Free copper	72	10	62	88	98	2	95	100

The term "free copper" in Table 5 expresses the results of colorimetric determinations by the cuprethol field procedure (5) that were performed for process control. Total metals were determined by atomic adsorption spectroscopy that responds to both free and combined forms of metals, in contrast to colorimetric procedures that may not detect strongly complexed species. An attempt to determine free copper by ion selective electrode was unsuccessful, due to erratic electrode response.

Table 6 presents complete records of operation and monitoring of the pilot plant. Figures 33 to 37 are profiles versus time of influent and effluent total copper, zinc, manganese and iron, and of pH.

TABLE 6 : Pilot Plant Operating Data

Sampling identification			Temperature, °C		Acid mine drainage total flow, gpm[a]	Treatment plant influent					Treatment system				Limestone neutralized effluent	Reagent neutralized effluent		Settlement basin effluent		Straw filter (final) effluent						"Free" copper by colorimetric analysis, mg/L				
Serial number (1)	Date (1982) (2)	Time (3)	Acid mine drainage (4)	Air (5)	(6)	pH (7)	Copper mg/L (8)	Zinc mg/L (9)	Manganese mg/L (10)	Iron mg/L (11)	Limestone treatment unit (12)	Neutralization reagent (13)	Flow treated, gpm (14)	Water wheel speed, rpm (15)	pH (16)	pH (17)	Settleable solids mL/L (18)	pH (19)	Settleable solids mL/L (20)	pH (21)	Settleable solids mL/L (22)	Copper mg/L (23)	Zinc mg/L (24)	Manganese mg/L (25)	Iron mg/L (26)	Treatment plant influent (27)	Limestone neutralized efflt. (28)	Reagent neutralized efflt. (29)	Settlement basin effluent (30)	Straw filter final effluent (31)
1	9/ 2	1400	-	-	-	4.4	-	-	-	-	drum[b]	-	-	-	5.3	-	-	7.1	-	-	-	-	-	-	-	12	12	-	3	-
2	9/ 3	1430	-	-	-	4.2	-	-	-	-	drum	-	-	-	4.6	-	-	4.8	-	-	-	-	-	-	-	12	12	-	12	-
3	9/ 4	1430	-	-	-	4.2	-	-	-	-	drum	-	24	-	4.5	-	-	4.6	-	-	-	-	-	-	-	12	12	-	12	-
4	9/ 5	1430	-	-	120	4.3	-	-	-	-	drum	-	64	-	4.4	-	-	4.4	-	4.4	-	-	-	-	-	12	12	-	12	12
5	9/ 6	1200	-	-	100	4.2	-	-	-	-	drum	-	48	2.8	4.6	-	-	4.6	-	4.6	-	-	-	-	-	12	12	-	12	12
6	9/11	1630	-	-	120	4.9	-	-	-	-	drum	-	48	3.0	4.9	-	-	5.0	-	5.0	-	-	-	-	-	13	13	-	-	13
7	9/12	1600	-	-	110	4.5	-	-	-	-	drum	-	44	2.7	4.7	-	-	4.7	-	4.7	-	-	-	-	-	13	13	-	12	12
8	9/13	1730	-	-	140	4.7	-	-	-	-	drum	ash[d]	32	1.8	-	10.0	33	9.4	5	9.4	1.3	-	-	-	-	13	-	12	5	5
9	9/14	1100	6	14	140	5.0	-	-	-	-	drum	ash	31	1.8	-	9.1	32	9.9	4	9.9	0.5	-	-	-	-	13	-	10	4	4.5
10	9/15	1000	6	3	110	5.1	16.8	0.67	2.58	0.41	drum	ash	22	1.7	-	10.5	-	10.1	-	10.0	-	2.5	0.24	0.77	0.15	10	-	8	2.5	2.5
11	9/15	1300	-	-	120	5.1	17.0	0.69	2.60	-	drum	ash	31	5.4	-	10.3	-	10.2	-	10.2	-	5.1	0.23	1.09	0.37	12	-	12	2.5	3.5
12	9/18	1800	6	6	120	5.1	16.6	0.69	2.55	-	-	ash	25	27[f]	-	-	-	5.1	-	5.2	-	14.0	0.68	2.58	0.63	12	-	-	12	12
13	9/19	1300	6	9	100	5.0	16.4	0.67	2.59	0.34	-	ash	25	9.3	-	10.3	26	9.8	0.2	9.8	0.3	6.4	0.30	1.79	0.48	10	-	10	1.2	1.2
14	9/20	0700	6	15	110	5.0	16.7	0.68	2.55	-	-	ash	1.7	-	-	11.1	11	9.4	0.0	9.0	0.0	1.1	0.06	0.76	0.19	10	-	4	0.25	0.15
15	9/21	0700	6	-	115	5.1	-	0.73	2.61	-	-	ash	1.6	-	-	11.2	11	10.4	0.0	10.3	0.0	0.3	0.02	0.15	0.11	10	-	4	0.03	0.03
16	10/ 2	1530	6	11	75	5.0	16.3	0.73	2.67	0.52	-	ash	0.52	0.104	-	11.3	0.0	10.2	0.0	10.2	0.0	0.7	0.03	0.23	0.22	8	-	0.2	0.03	0.04
17	10/ 3	1100	6	14	85	4.9	15.8	0.68	2.56	-	mill[c]	ash	0.95	0.067	7.0	11.1	0.1	10.4	0.0	10.3	0.0	0.6	0.03	0.23	0.26	9	1.4	0.2	0.03	0.03
18	10/ 9	1530	6	12	65	5.1	15.3	0.66	2.58	-	mill	ash	0.17	-	8.4	8.9	-	10.2	-	10.3	-	-	-	-	-	13	0.3	0.1	0.04	0.04
19	10/10	1430	6	14	70	5.1	15.3	0.68	2.62	0.45	mill	ash	0.71	0.100	6.5	9.7	80	10.2	0.0	10.2	0.0	0.4	0.02	0.13	0.11	10	3	130	0.05	0.04
20	10/11	1500	6	15	75	5.0	14.9	0.65	2.51	-	mill	ash	0.45	0.052	6.6	9.6	52	10.2	0.0	10.1	0.0	0.4	0.02	0.13	0.15	8	2.5	30	0.04	0.04
21	10/12	1630	6	15	75	4.9	14.9	0.67	2.55	-	mill	ash	1.00	0.24	6.3	9.7	65	10.2	0.0	10.1	0.0	0.4	0.02	0.13	0.15	9	3.5	30	0.05	0.05
22	10/13	1530	6	14	80	4.8	14.8	0.64	2.60	0.37	mill	ash	1.8	0.33	6.9	9.5	10	9.9	0.0	9.9	0.0	0.8	0.04	0.26	0.11	10	8	4	0.25	0.25
23	10/16	1630	6	15	70	4.7	14.5	0.65	2.57	-	mill	ash	2.8	0.31	5.5	9.6	-	9.8	-	9.7	-	1.0	0.05	0.32	0.11	8	8	5	0.2	0.25
24	10/17	1230	6	15	80	4.8	14.5	0.65	2.56	-	mill	ash	2.5	-	5.7	8.7	8	9.8	0.0	9.7	0.0	1.1	0.07	0.39	0.19	8	4	5	0.4	0.4
25	10/18	1530	6	13	60	4.8	14.5	0.64	2.59	0.34	mill	ash	1.6	0.22	5.9	8.7	7	9.6	0.0	9.7	0.0	0.9	0.03	0.33	0.08	8	4	4	0.1	0.1
26	10/19	1400	6	10	65	4.8	14.3	0.64	2.56	-	mill	ash	1.4	0.26	5.6	8.7	6	9.6	0.0	9.6	0.0	0.7	0.03	0.40	0.22	8	4	3	0.4	0.3
27	10/20	1000	6	7	60	4.9	14.3	0.64	2.53	-	mill	ash	1.4	0.25	5.5	8.6	6	9.4	0.0	9.5	0.0	1.8	0.11	0.66	0.19	8	4	4	0.4	0.4
28	10/23	1630	6	6	75	4.7	14.1	0.63	2.59	0.34	mill	mix[e]	2.3	0.24	5.8	10.8	16	8.9	0.0	8.6	0.0	-	-	-	-	10	4	14	0.2	0.4
29	10/24	1000	6	9	65	5.0	14.0	0.65	2.59	-	mill	mix	2.3	0.26	5.5	9.2	7	9.1	0.0	9.0	0.0	1.1	0.05	0.75	0.22	10	6	6	0.5	0.4
30	10/25	0830	6	4	60	4.9	14.0	0.66	2.61	-	mill	mix	0.76	0.21	5.5	9.7	11	8.9	0.0	8.3	0.0	0.7	0.06	0.45	0.63	14	12	12	0.1	0.1
31	10/26	1000	-	-	60	5.0	14.7	0.66	2.68	0.37	mill	mix	6.9	-[g]	5.1	6.1	0.0	6.9	0.0	7.0	0.0	0.9	0.29	1.04	0.81	11	8	4	0.4	0.2
32	10/30	1330	6	3	70	5.0	15.2	0.68	2.72	-	mill	mix	3.0	0.39	5.2	10.9	25	6.8	0.0	6.8	0.1	4.7	0.40	1.77	0.55	10	4	6	4	3
33	10/31	0930	6	0	70	5.1	15.2	0.69	2.74	-	mill	mix	2.2	0.30	5.3	12.0	45	9.9	0.0	9.8	0.3	1.5	0.08	0.43	0.26	10	6	4	0.4	0.5
34	11/ 6	0200	5	3	75	4.8	14.8	0.67	2.76	0.30	mill	mix	0.74	-	5.1	7.3	6	8.8	0.0	8.7	0.0	1.6	0.07	0.66	0.37	9	5	5	0.3	0.4
35	11/ 6	1545	6	8	75	4.8	14.6	0.68	2.71	-	mill	mix	1.5	0.28	5.3	9.8	19	9.8	0.0	9.8	0.0	1.3	0.07	0.57	0.19	9	7	6	0.1	0.1
36	11/ 7	0700	-	-2	80	5.2	14.6	0.68	2.71	-	mill	mix	2.3	0.30	5.5	10.2	22	10.2	0.0	10.2	0.1	0.8	0.04	0.31	0.11	9	7	7	0.2	0.2
37	11/ 8	0730	-	-4	75	5.2	14.4	0.66	2.69	0.45	mill	mix	0.85	0.21	6.5	-	-	9.8	-	9.8	0.4	1.4	0.07	0.38	0.19	11	2	-	0.03	0.6
38	11/13	0130	5	-10	65	5.1	13.8	0.66	2.68	-	mill	mix	1.2	0.16	5.4	7.8	6	9.5	0.0	9.5	0.0	0.9	0.06	0.50	0.04	9	6	1	0.3	0.1
39	11/13	1500	6	2	70	4.9	13.8	0.66	2.66	-	mill	mix	2.7	0.34	5.2	8.0	12	9.1	0.0	9.0	0.0	1.9	0.11	0.86	0.22	10	6	6	0.3	0.3
40	11/14	0800	6	-2	75	5.0	14.0	0.63	2.75	0.41	mill	mix	2.8	0.36	5.7	10.9	22	9.5	0.0	9.6	0.0	1.6	0.08	0.72	0.15	10	6	6	0.3	0.4
41	11/15	0715	6	-2	75	5.2	13.9	0.66	2.67	-	mill	mix	2.8	0.34	5.4	10.5	19	9.7	0.0	9.7	0.0	0.7	0.05	0.27	0.08	9	10	6	0.1	0.1
42	11/20	1300	6	-2	-	5.1	-	-	-	-	mill	mix	1.7	0.25[h]	5.2	7.6	-	6.9	-	6.9	-	-	-	-	-	-	-	-	-	-
43	12/ 7	1500	-	-	-	-	-	-	-	-	-	ash	2.0	2.2[h]	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

[a]treated flow plus bypassed flow metered by a Palmer-Bowlus flume. [b]tumbling drum. [c]autogenous mill. [d]soda ash. [e]mixed soda ash and caustic soda. [f]water wheel running free, with reagent and limestone both exhausted. [g]at high inflow jet overshot water wheel stalling wheel; mine drainage flooded settlement basin. [h]after restarting stalled water wheel; stalling remedied on 12/7 by disconnecting autogenous mill for winter operation.

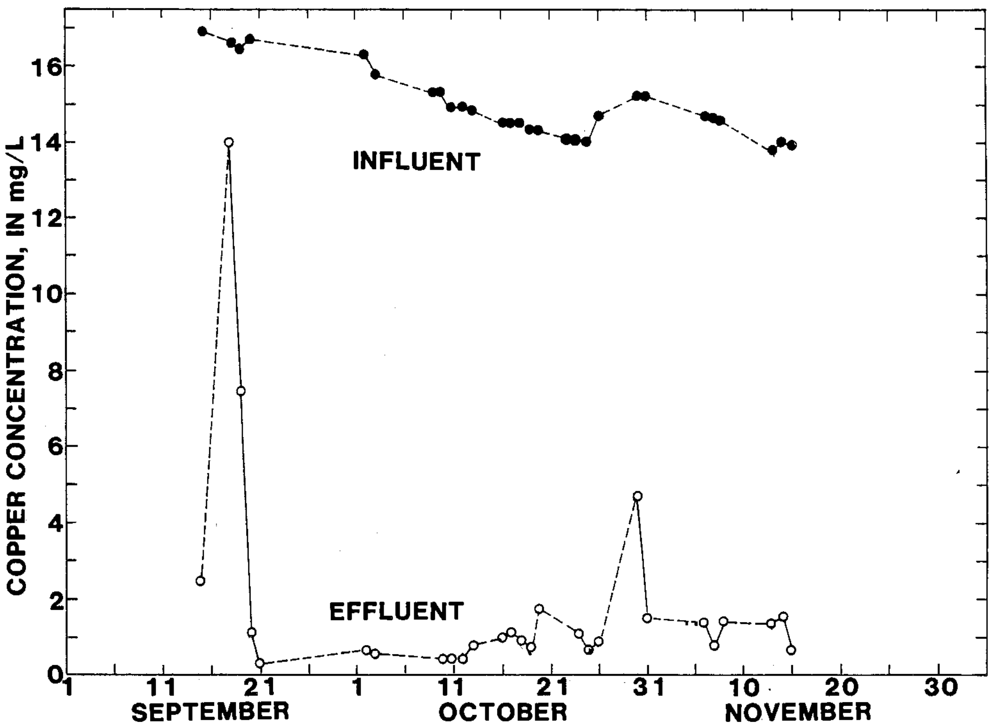


Fig. 33: Pilot Plant Influent and Effluent Total Copper Versus Time.

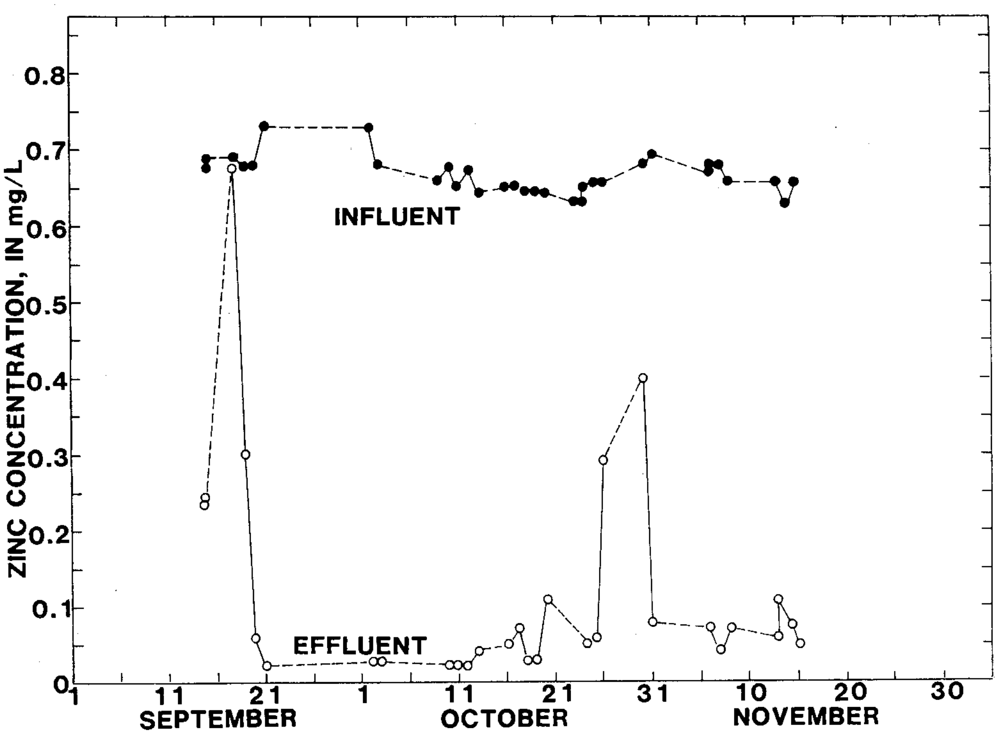


Fig. 34: Pilot Plant Influent and Effluent Total Zinc Versus Time.

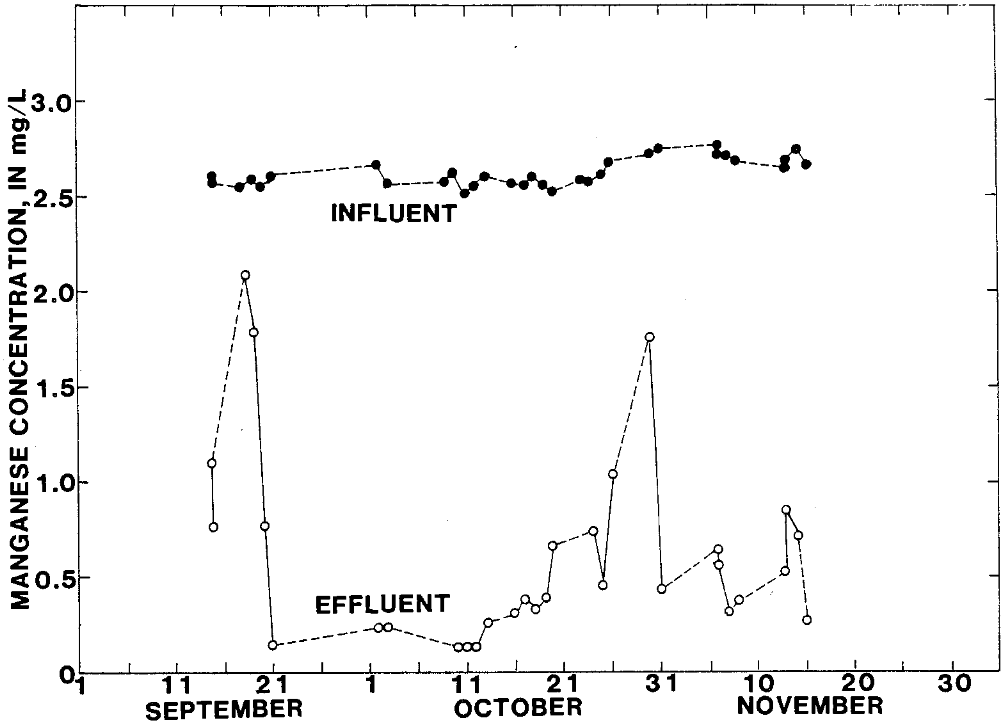


Fig. 35: Pilot Plant Influent and Effluent Total Manganese Versus Time.

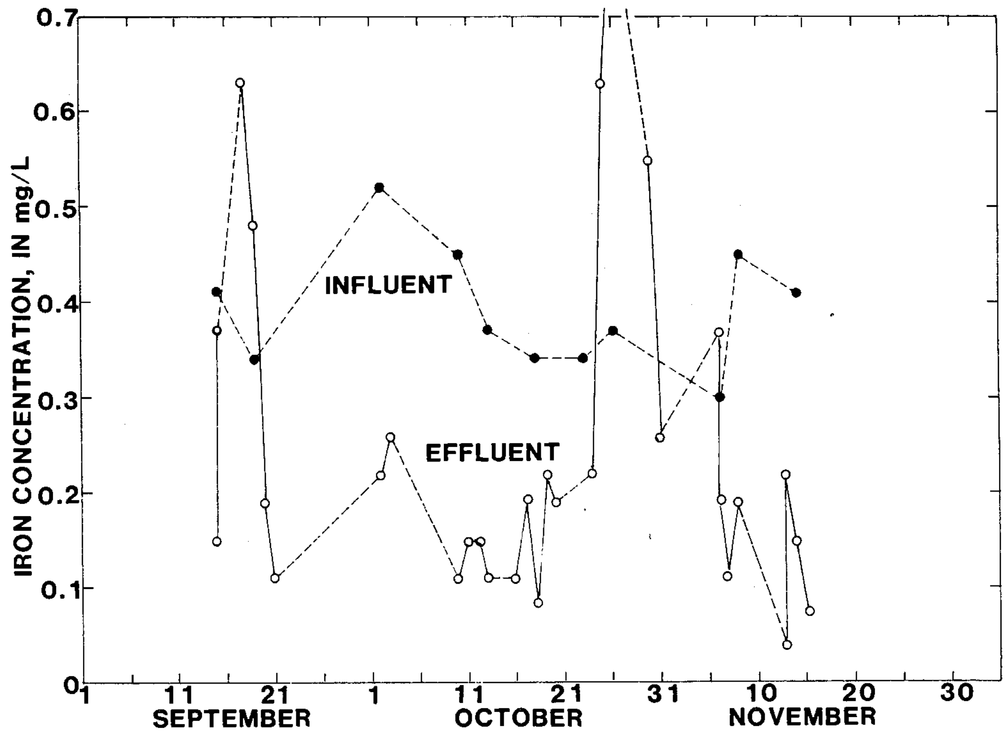


Fig. 36: Pilot Plant Influent and Effluent Total Iron Versus Time.

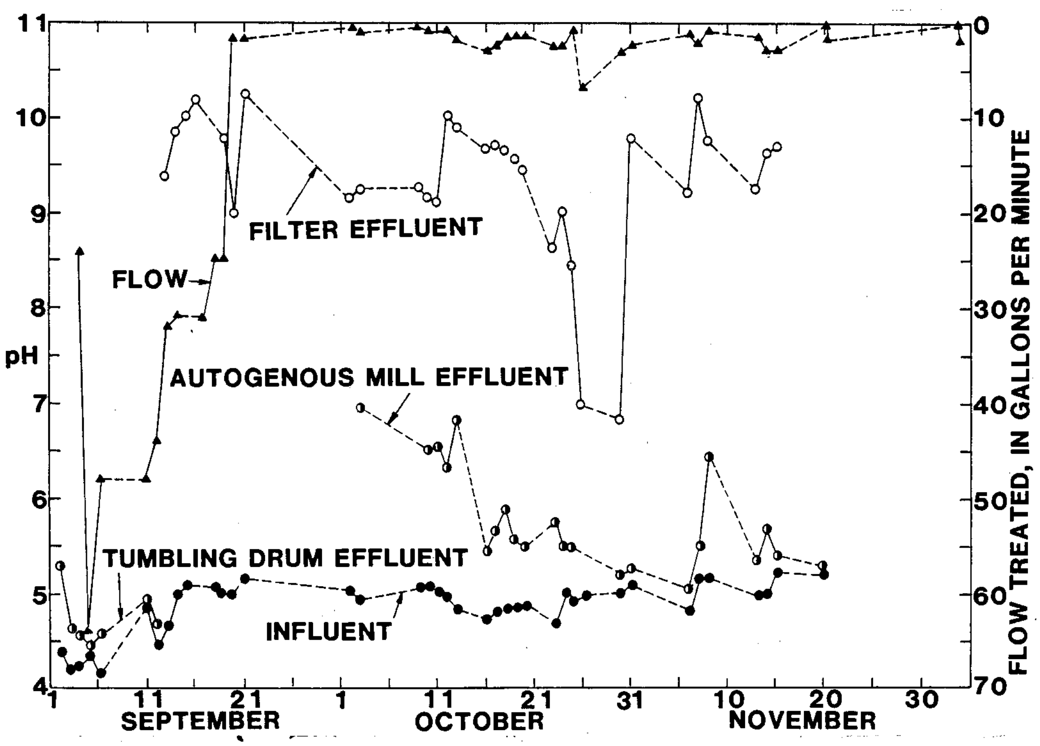


Fig. 37: Flow Treated By Pilot Plant, and Influent and Effluent pH Versus Time.

Table 5 (and subsequent performance evaluations of individual processes) exclude process failures indicated by peaks in metals and low effluent pH on September 18 and October 26 and 30. The September 18 planned failure resulted from insufficient chemical storage capacity at that time to continue chemical neutralization over a prior three day absence. Failure on the evening of October 25 resulted from flow to the pilot plant so high as a result of rain that the jet from the autogenous mill overshot the water wheel, stalling the wheel and the chemical feed pump. Mine water also flooded into the sedimentation basin.

Performance of Individual Process Units.- For each process in the pilot plant, Table 7 presents summary data on operation and performance at both low and high flow. These data are compared to nominal design values selected in design of the pilot plant, although the determination of suitable parameter values rather than the validation of design values was the objective of pilot testing.

Free Copper Versus Total Copper.- Although only free copper was generally determined on influent to and effluent from individual processes, process performance with respect to total copper can be estimated from a correlation of observed pairs of free and total copper determinations on pilot plant influent and effluent. Figure 38 plots these data, the trend line being described by the equation

$$\text{Ratio} = \frac{\text{free copper}}{\text{total copper}} = 0.8 \left[1 - \exp(-0.3 \times \text{total copper}) \right] \qquad (1)$$

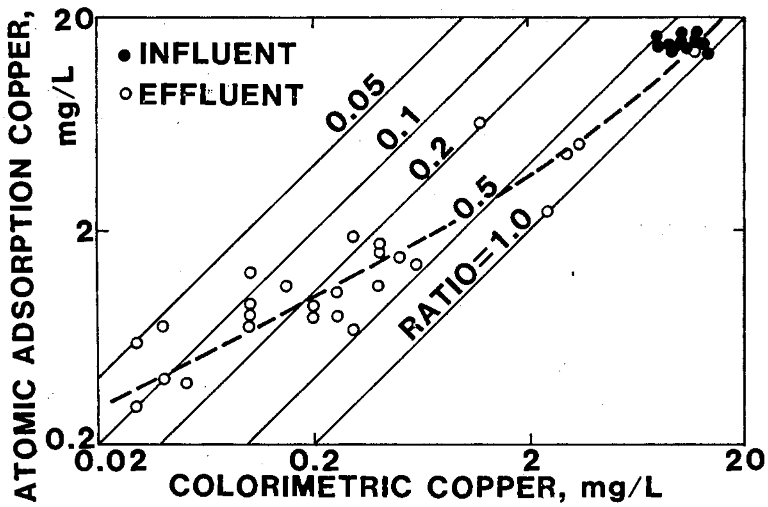


Fig. 39: Total Copper By Atomic Adsorption Spectroscopy Versus Free (Colorimetric) Copper On Pilot Plant Influent and Effluent.

Limestone Pre-Neutralization.- Initially the tumbling drum (at 46 gpm mean flow) and the autogenous mill (at 1.7 gpm mean flow) produced pH increases of 0.9 and 3.3 units respectively, compared to design values of 0.7 and 2.3 units respectively. However, the pH increases attained steadily declined with time, as shown in Fig. 37, to become insignificant after 11 days of operation for the tumbling drum, and 44 days for the autogenous mill.

Because the pilot plant was operated to simulate the conditions of low maintenance or unmanned operation of the prototype in winter, no attempt was made to remedy the declining preformance of the limestone preneutralization units. For the tumbling drum, this decline resulted from depletion of the load of limestone in the drum, and from rounding of the stone, both factors leading to a decreasing rate of abrasion of neutralizing fines from the limestone surfaces. After 14 days of operation the entire load of 1/2-3/4 in. stone had been sufficiently abraded to pass through the 1/2 in. weep holes in the water wheel. Table 8 shows the grading of the three sizes of crushed limestone used in limestone neutralization units at the prototype facility, that were produced by crushing raw rock graded as shown by Fig. 39. In Fig. 40, the fresh limestone charged into the tumbling drum is shown by Frame c, and the rounded stone that passed out of the drum is shown by Frame d. It would appear necessary to develop a mechanism for automatically charging stone to a tumbling drum to obtain the stamina necessary for unmanned winter operation at Walker.

For the autogenous mill, two design problems led to the declining performance. First, the loss of mechanical efficiency of the water wheel when rotating intermittently at low flow reduced the power developed by the wheel to rotate the mill, thereby reducing the ability of the mill to abrade the stone surfaces. Intermittent rotation of the water wheel at low flow was unexpected and a mechanical rationalization is lacking, but to avoid this condition one should evidently design a smaller water wheel to rotate at 1.7 rpm or faster (as used for operation at high flow) rather that below 0.4 rpm (as used for low flow studies). It would then be necessary to correspondingly adjust the gear-down ratio from the water wheel from its value of 11.6, equal to the ratio of the 37.5 in. diam. pitch of the drive chain around the mill to the 3.236 in. pitch of the drive sprocket.

TABLE 8 : Size Grading of Crushed Limestones.

Screen size	Fraction by weight passing stated screen		
	1/8-1/4 in.	1/4-1/2 in.	1/2-3/4 in.
#12 square mesh (0.060 in.)	0.182	0.001	0.000
#4 square mesh (0.187 in.)	0.454	0.009	0.000
1/4 in. square mesh	0.987	0.828	0.103
1/2 in. square mesh	0.997	0.934	0.172
5/8 in. circular mesh	1.000	1.000	0.960
7/8 in. circular mesh	1.000	1.000	1.000
1 inch circular mesh	1.000	1.000	1.000

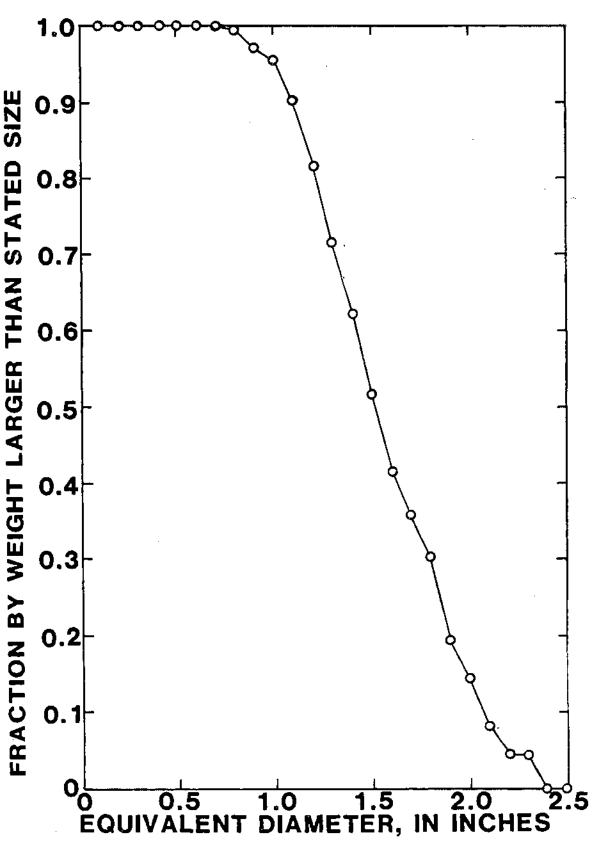


Fig. 39: Size Grading of Limestone Before Crushing

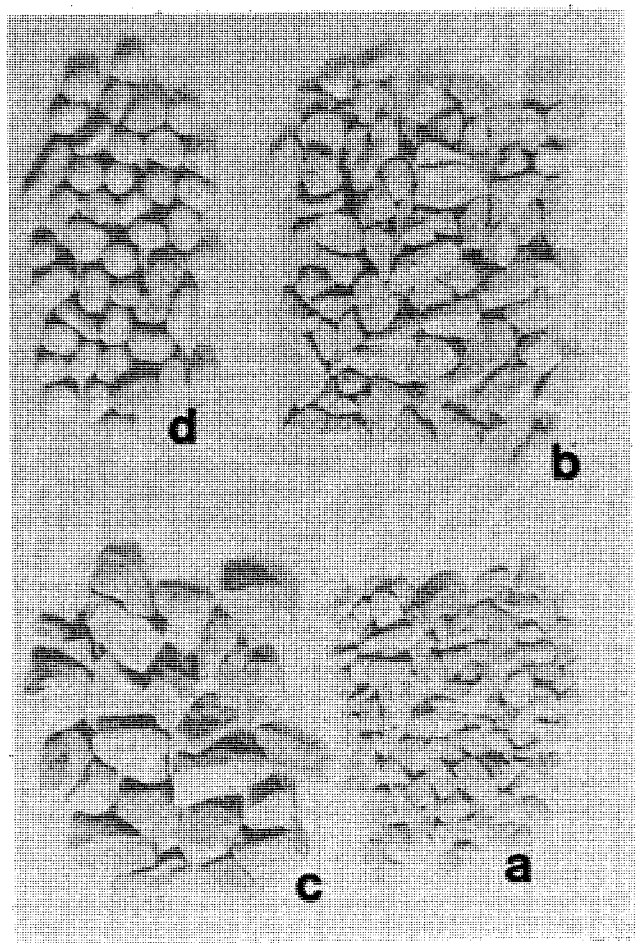


Fig. 40: Photo of Crushed Limestone on Millimeter Graph Paper (40% size).
a: 1/8-1/4 in. stone; b: 1/4-1/2 in. stone; c: 1/2-3/4 in. stone; d: 1/2-3/4 in. stone, after abrasive rounding in tumbling drum

TABLE 7: Operating Conditions and Performance of Process Units.

Pilot plant process unit	Conditions			Performance		
	Evaluation period, days	No. of times monitored	Flow treated gallons per min	pH increase units	Effluent pH, units	Free copper removal percent
Tumbling drum, design	-	-	60	0.7	4.7	0
actual, mean	11	7	46	0.3	4.7	0
SD	-	-	14	0.3	0.3	0
min	-	-	24	0.0	4.4	0
max	-	-	64	0.9	5.3	0
Autogenous mill, design	-	-	2.0	2.3	6.3	0
actual, mean	44	25	1.7	0.9	5.7	45
SD	-	-	0.9	0.8	0.6	26
min	-	-	0.2	0.1	5.1	-11
max	-	-	3.0	3.3	7.0	98
Chemical neutralization						
High flow, design	-	-	60	4 - 6	8-10	0
actual, mean	4	5	28	5.1	10.0	10
SD	-	-	4	0.5	0.6	11
min	-	-	22	4.1	9.1	0
max	-	-	32	5.3	10.5	23
Low flow, design	-	-	2.0	4 - 6	8-10	0
actual, mean	57	25	1.8	3.7	9.8	24
SD	-	-	0.8	1.4	1.2	35
min	-	-	0.2	2.2	7.3	-50
max	-	-	3.0	6.7	12.0	86
Sedimentation basin						
High flow, design	-	-	60	0.0	8-10	-
actual, mean	4	5	28	-0.2	9.9	71
SD	-	-	4	0.6	0.3	13
min	-	-	22	-0.6	9.4	58
max	-	-	32	0.8	10.2	88
Low flow, design	-	-	2.0	0.0	8-10	-
actual, mean	57	25	1.8	0.0	9.7	90
SD	-	-	0.8	1.1	0.5	12
min	-	-	0.2	-2.1	8.8	60
max	-	-	3.0	1.7	10.4	99
Straw bale filter						
High flow, design	-	-	60	0.0	8-10	-
actual, mean	4	5	28	0.0	9.9	-11
SD	-	-	4	0.0	0.3	17
min	-	-	22	-0.1	9.4	-40
max	-	-	32	0.0	10.2	0
Low flow, design	-	-	2.0	0.0	8-10	-
actual, mean	57	25	1.8	-0.1	9.6	-3
SD	-	-	0.8	0.2	0.6	30
min	-	-	0.2	-0.6	8.3	-100
max	-	-	3.0	o.1	10.3	67

A second improvement to the design of the autogenous mill would be to install baffles to reduce hydraulic short-circuting through the unit. Three baffles would be needed at one-third points along the length of the mill, each baffle with a port against the wall of the mill, the ports being separated by 120^{o}. For circular ports, the diameter of the port should not exceed four-thirds the minimum depth of stone in the mill minus one-third the diameter of the mill.

Table 7 also documents an average reduction of 45% of free copper through the autogenous mill, though quite variable with a standard deviation of 26%. Initially pronounced with up to 98% removal, this effect declined with time to average about 30% removal near the end of autogenous mill studies; in one case an 11% increase in copper through the mill was noted. On opening the mill at the end of its period of operation the limestone was found to be tinged green.

Two effects may cause a reduction of copper through the mill. Galvanic replacement of iron from the mill body would plate copper from the water on the mill; a bronzed coating was at times observed on the water wheel. This effect may help explain the sporadic removal of iron by the treatment plant, although increases in the concentration of iron are much less than reductions in the concentrations of free copper. Second, carbonate from the limestone is likely to complex part of the copper to a chemical state undetected by the colorimetric analysis for free copper. Although complexation would not reduce the concentration of total copper, the lesser biological significance of complexed copper (10) relative to free copper merits consideration. In view of the decline in removal of free copper concurrently with the declining degree of neutralization by the autogenous mill, complexation appears likely to be the major factor responsible for the reduction of free copper through the autogenous mill.

However, if one attempts with the aid of Eq. 1 to adjust the observed 45% removal of free copper in the autogenous mill to estimate the removal of total copper, a scarcely changed value of 40% results, which is inconsistent with postulating that removal of free copper in the autogenous mill is largely a result of complexation. Although we later see that limestone is capable of removing an even larger percentage of total copper (in the limestone barrier), it is prudent to consider that because complexation is primarily associated with neutralization and copper removal with sedimentation, free copper determinations per se cannot reliably be interpreted in terms of copper speciation or total copper removal.

Chemical Neutralization.- Limestone treated mine drainage (or raw mine drainage) was chemically treated to a mean pH of 10.0 during 6 days of continuous operation at high flow (28 gpm mean), and to a mean pH of 9.8 during 57 days of continuous operation at low flow (1.6 gpm), as Table 7 shows. Soda ash dissolved in mine water to an approx. 5% solution was used as the neutralizing agent for high flow operation, and for the first 31 days of low flow operation, at which time 50% caustic soda was incrementally added to the chemical feed tanks for the remaining 26 days of low flow operation.

As the chemical feed became stronger with daily 2 gal additions of 50% caustic soda to the 30 gal of soda ash solution, so the required feed rate of chemical solution was decreased by adjustment of the chemical feed pump. The rated discharge of the peristaltic feed pump (Tat model 411) is 11 mL per revolution for a 3/8 in. diam. feed pipe, 19 mL/rev for a 1/2 in. diam. feed pipe, and 30 mL/rev with both feed pipes installed. The peristaltic pump replaced a flexible impeller positive displacement pump (Jabsco model 12860-0001) after the end of high flow operation because leakage between the sides of the impeller and the body of the Jabsco pump (10 gal/hr at zero rotation under a head of approx. 3 ft) was excessive for adequate control of feed rate.

Because a drum of 50% caustic soda had frozen in the mine in mid-November, and a subsequent attempt to haul in a thawed drum was unsuccessful due to loss of road access to the mine, plans to provide winter feed of mixed soda ash and caustic soda were thwarted. To attain the higher water wheel-driven pump speed needed for winter operation on 5.5% soda ash solution, resistance was removed from the water wheel by disengaging the autogenous mill (by jacking it clear of the drive sprocket). The resulting 2.2 rpm speed of the water wheel and feed pump, together with the 19 mL/rev delivery of the pump with a 1/2 in. feed pipe, resulted in dosing 2.2 x 19 x 1440 / 3785 = 15.9 gal/day of 5.5% soda ash solution to the measured 2.0 gpm flow treated. The resulting concentration of soda ash in chemically neutralized water is 2.2 x 19 x 5.5% x 10^6/(3785 x 2) = 300 mg/L, equivalent to 300 x 100 / 106 = 280 mg $CaCO_3$/L of alkalinity, that is later in this report shown to approximate the soda ash dosage needed to attain pH 10 in summer operation. The 2,300 gal of chemical solution in the feed tank should last up to 2,300/15.9 = 145 days, or December 7, 1982 to May 9, 1983, although inspection earlier than the latter date will be attempted.

A nominal removal of free copper due to chemical neutralization is noted in Table 7, 10% mean at high flow and 24% mean at low flow, apparently due to complexation of the metal to a form undetected by the test for free copper. Batch tests, described later, were performed to evaluate dose-response characteristics of neutralizing reagents. Pilot plant characteristics are discussed together with batch test results.

Sedimentation Basin.- Solids removal in the sedimentation basin proved to limit the overall efficiency of removal of copper in the pilot plant. At high flow, with a surface overflow rate of 27 gallons per day per square foot of basin surface area and a basin detention time of 6.7 hr at the mean flow rate of 28 gpm, Table 7 shows a mean free copper removal of 71%. At the low flow of 1.6 gpm, for the surface overflow rate of 1.5 gpd/ft^2 and detention of 4.9 days, the removal of free copper was 90%. These percentage removals are likely to also represent removals of total copper as may be judged by comparing the overall pilot plant removal of total copper in Table 5 (72% at high flow and 93% at low flow) with the above removals of free copper, since effects of complexation on free copper analyses have already been accounted for in the apparent removals observed in neutralization processes. Nevertheless, residual copper is present in sedimentation basin effluent at considerably higher concentration than the theoretical solubility of copper at the pH in the sedimentation basin, as is discussed under batch neutralization.

To characterize the settling characteristics of copper chemically precipitated from Walker mine drainage, batch settling column tests and continuous flow settling column tests were conducted. For this purpose a capped 15 ft length of 6 in. diam PVC pipe was fitted with 14 valved sampling nozzles at the base and at even footages for 13 ft up the column, and a 3/4 in. drain valve at the base (Fig. 28). The column was installed against the wall at the inlet to the sedimentation basin adjacent to the screw-capped end of the inlet manifold, and a port was cut in the column at the level of the manifold to provide for rapid filling of the column for batch settling tests. For continuous flow settling tests the required flow of neutralized mine drainage was drawn from an orifice in the capped manifold, and delivered by 1/2 in. hose to the base of the column. In this case, flow upwards through the column was measured by timing a measured volume leaving the column at the open sampling orifice 13 ft above the column base.

To monitor the performance of the settling column free copper was determined on 50 mL samples drawn from each sampling nozzle, with sets of samples at intervals of time in the case of batch tests. For evaluation of total copper removal in the sedimentation basin and in settling tests, free copper could be adjusted to total copper using Eq. 1, because complexation of copper appears to be largely completed before the neutralized waste enters the sedimentation basin or settling column. This is indicated by the correspondence between overall removals of total copper in the pilot plant listed in Table 5 (72% at high flow or 93% at low flow) and removals of free copper in the sedimentation basin listed in Table 7 (71% at high flow or 90% at low flow). Table 9 lists the resulting estimated removals of total copper (Part A), together with total copper concentrations calculated by Eq. 1 (Part B) from determined concentrations of free copper (Part C).

Table 9A data, plotted in Fig. 41, reveal that for batch tests at any given time of settlement as well as for continuous flow tests the removal of copper is generally independent of depth, except of course at the base of the column where settled sludge accumulates. The only test displaying a variation of removal with depth statistically significant at the 90% level was the lowest flow continuous flow test, run at a surface overflow rate of 365 gsfd. Because the removals observed in sets of samples from batch tests at particular times were statistically indistinguishable within sets above one foot from the base of the column, these sets were treated as replicate readings from a one foot settling depth. This permits comparison of settling column data with data from the sedimentation basin, that also has a mean depth of 1.0 ft. Conseqently, all data can be considered together, as in Fig. 42.

TABLE 9: Batch and Continuous Flow Settling Column Test Data.

A: Percentage Removal of Total Copper, Calculated From Table 9B Data.

Overflow rate, gsfd	Settling time, min.	Height of sampling point above floor of settling column, in feet													
		0	1	2	3	4	5	6	7	8	9	10	11	12	13
0	15	19	56	87	80	87	87	87	80	74	74	38	68	74	<0
0	30	<0	88	74	68	68	68	95	87	74	80	87	74	87	74
0	45	<0	80	87	80	74	74	74	74	87	87	87	80	80	80
0	60	<0	80	87	89	80	90	88	89	90	90	90	87	80	87
0	90	<0	89	91	90	90	91	92	89	91	92	87	90	88	89
0	120	<0	94	92	90	92	91	92	90	92	94	91	94	90	87
0	180	<0	92	94	91	91	92	92	92	94	94	94	90	92	80
0	360	<0	94	95	94	94	95	96	96	96	95	95	92	95	87
0	1440	80	97	97	97	97	97	97	98	97	97	97	97	98	92
0	2880	95	98	98	97	98	97	98	98	98	98	98	98	98	98
365	-	18	89	90	93	93	93	95	95	95	95	95	95	95	96
400	-	<0	94	93	93	92	92	92	92	92	92	92	93	93	93
670	-	<0	<0	<0	<0	91	93	93	96	93	93	93	87	92	93

B: Total Copper, Calculated From Free Copper By Equation 1, mg/L.

Overflow rate, gsfd	Settling time, min.	Height of sampling point above floor of settling column, in feet													
		0	1	2	3	4	5	6	7	8	9	10	11	12	13
0	15	15[a]	8	2	4	2	2	2	4	5	5	10	6	5	25
0	30	50	2	5	6	6	6	1	2	5	4	2	5	2	5
0	45	200	4	2	4	5	5	5	5	2	2	2	4	4	4
0	60	500	4	2	2	4	2	2	2	2	2	2	2	4	2
0	90	1100	2	2	2	2	2	1	2	2	1	2	2	2	2
0	120	900	1	1	2	1	2	1	2	1	1	2	1	2	2
0	180	500	1	1	2	2	1	1	1	1	1	1	2	1	4
0	360	200	1	1	1	1	1	0.7	0.7	0.7	1	1	1	1	2
0	1440	4	0.5	0.5	0.6	0.5	0.5	0.6	0.4	0.5	0.5	0.5	0.6	0.4	1
0	2880	1	0.4	0.4	0.5	0.4	0.5	0.4	0.4	0.4	0.4	0.3	0.4	0.4	0.3
365	-	10[b]	1	1	0.8	0.8	0.8	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.5
400	-	400[c]	0.7	0.8	0.8	1	1	1	1	1	1	1	0.8	0.8	0.8
670	-	300[d]	200	200	200	2	1	1	0.7	1	1	1	2	1	1

Influent concentrations: 19[a], 13[b], 12[c] and 18[d].

C: Free Copper, Determined By Colorimetric Field Method (5), mg/L

Overflow rate, gsfd	Settling time, min.	Height of sampling point above floor of settling column, in feet													
		0	1	2	3	4	5	6	7	8	9	10	11	12	13
0	15	10[a]	6	1	2	1	1	1	2	3	3	9	4	3	20
0	30	40	0.9	3	4	4	4	0.2	1	3	2	1	3	1	3
0	45	200	2	1	2	3	3	3	3	1	1	1	2	2	2
0	60	400	2	1	0.8	2	0.7	0.9	0.8	0.7	0.6	0.6	1	2	1
0	90	900	0.8	0.5	0.6	0.6	0.5	0.4	0.8	0.5	1	1	0.6	0.9	0.8
0	120	700	0.3	0.4	0.7	0.4	0.5	0.4	0.6	0.4	0.5	0.5	0.3	0.6	1
0	180	400	0.4	0.3	0.5	0.5	0.4	0.4	0.4	0.3	0.3	0.3	0.6	0.4	2
0	360	100	0.3	0.2	0.3	0.3	0.2	0.1	0.1	0.1	0.2	0.2	0.4	0.2	1
0	1440	2	.06	.06	.08	.06	.06	.08	.04	.06	.05	.05	.08	.04	0.4
0	2880	0.2	.04	.04	.05	.04	.05	.04	.04	.04	.02	.02	.04	.04	.02
365	-	8[b]	0.3	0.3	.15	.15	.15	0.1	0.1	0.1	0.1	0.1	0.1	0.1	.05
400	-	350[c]	0.1	.15	.15	0.2	0.2	0.2	0.2	0.2	0.2	0.2	.15	.15	.15
670	-	200[d]	150	120	120	0.5	0.3	0.3	0.1	0.3	0.3	0.3	0.9	0.4	0.3

Influent concentrations: 15[a], 10[b], 9[c] and 14[d]. pH values: 11.1[a], 9.7[b], 11.1[c], 9.7[d].

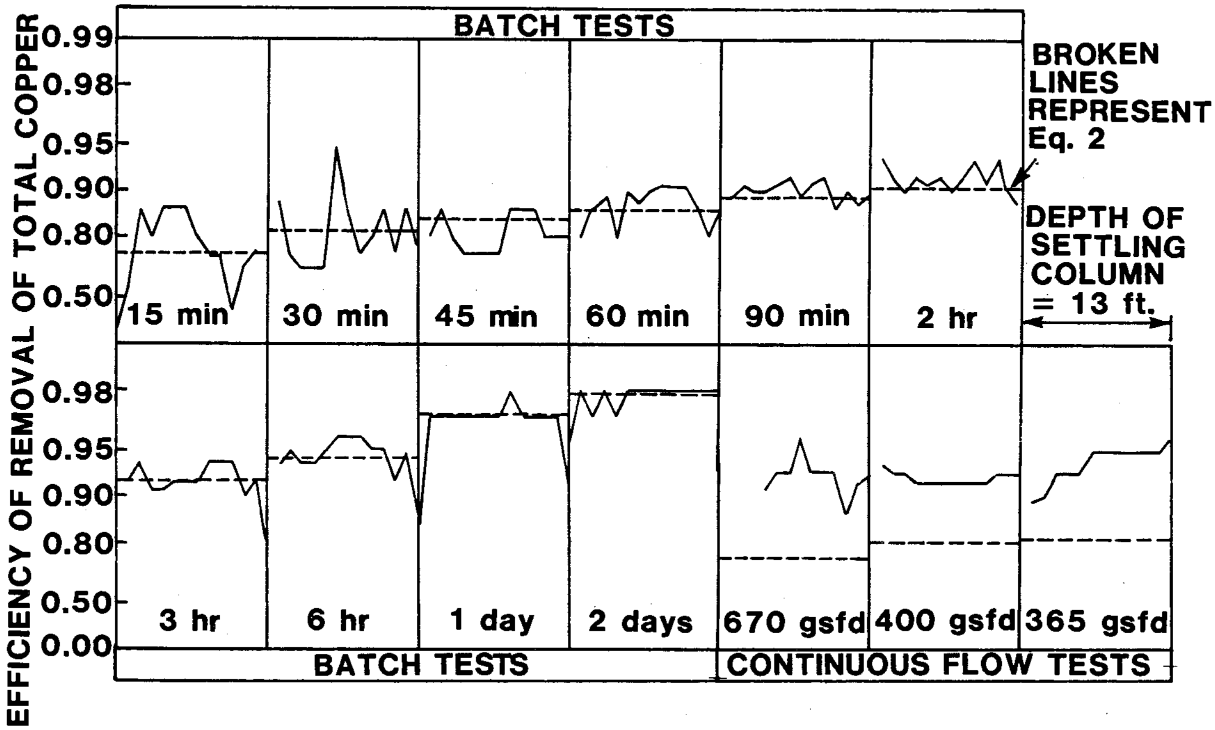


Fig. 41: Total Copper Removal vs. Depth for Settling Column.

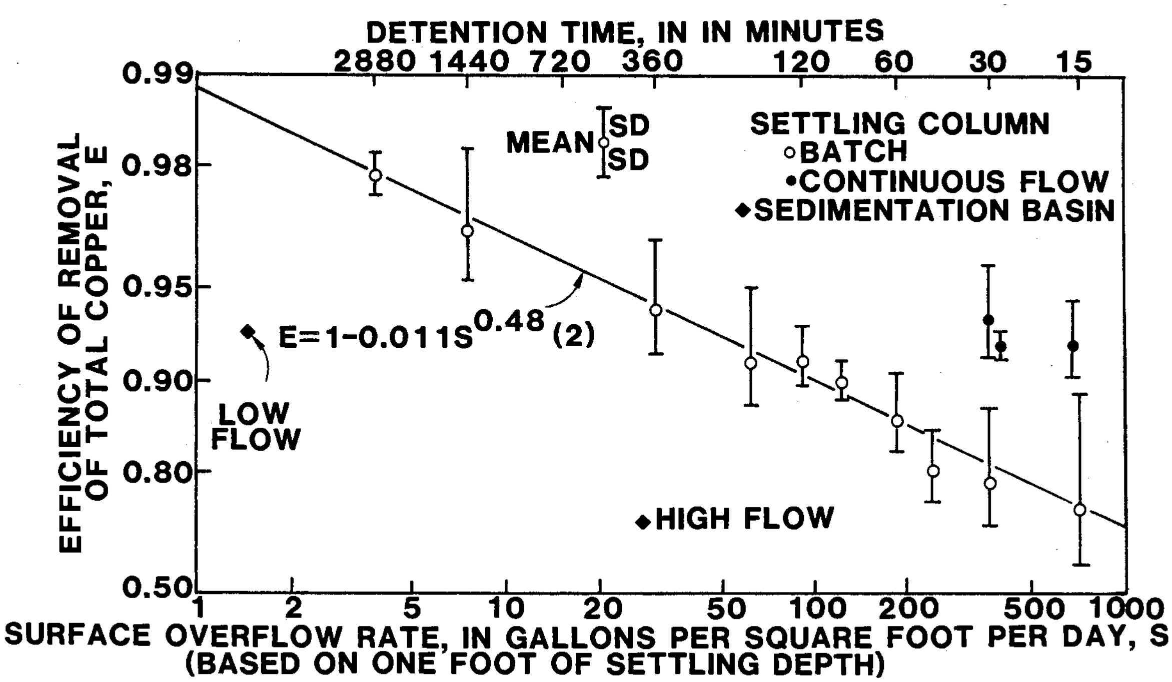


Fig. 42: Copper Removal vs. Surface Overflow Rate for Settling Column.

In Fig. 42, batch test detention times are converted to equivalent surface overflow rates, calculated for a one foot settling depth as 10,800 gsfd divided by detention time, in minutes. Batch test total copper removal is correlated with surface overflow rate according to

$$E = 1 - 0.011\ S^{0.48} \qquad (2)$$

where E =efficiency of removal of total copper; and S = surface overflow rate, in gallons per square foot per day. Residual total copper in continuous flow column tests is approx one-third that in batch tests at the same surface overflow rate, possibly due to sludge blanketing. Sedimentation basin residuals are 5.2 and 5.4 times higher than indicated by Eq. 2 at overflow rates corresponding to high and low flow respectively.

Sludge blanketing could not occur in the pilot sedimentation basin because upflow through a sludge layer could not occur to help trap solids. Use of this device in a prototype by introducing neutralized waste near the lowest point in the basin may introduce a risk of washing out sludge at high flow. More critical considerations in design of a prototype basin are to provide an adequate depth of water at the outlet weir to minimize scour and upwelling of solids (10 ft is common in water treatment clarifiers, compared to 2 ft in the pilot sedimentation basin), and to avoid conditions conducive to short-circuting.

Sedimentation basin performance can be characterized by the ratio of the surface overflow rate in a settling column providing the same removal as the basin to the basin surface overflow rate. At high flow, 72% removal of total copper at 27 gsfd in the basin could have been achieved in the column at 850 gsfd, 31 times higher. At low flow, 93% removal at 1.4 gsfd in the basin would occur at 47 gsfd in the column, 34 times higher. One might expect these multiples to be lower in a basin deeper at the outlet than the pilot basin, although theory (12) suggests that the ratio may not fall below that associated with normally occuring wind induced mixing in lagoon-type basins. To remove the same fraction of solids as a quiescent settling column, a mixed basin must operate at a surface loading reduced from that in the column by a factor 1 - E[1]. On this basis, Eq. 2 takes the form

$$E = 1 - 0.011 \left[S / (1 - E) \right]^{0.48}$$

$$= 1 - 0.047 S^{0.32} \qquad (3)$$

For example, at the peak 1981-82 spring flow of 830 gpm, Eq. 3 implies a 0.3 acre pond for 80% removal of total copper, or 2.6 acres for 90% removal; a 2.6 acre pond appears infeasible at the Walker site.

Straw Bale Filter.- Table 7 shows that no statistically significant change in free copper concentration occurred through the straw bale filter. Copper not removed in the sedimentation basin also passes through this filter. The most cost-effective way to enhance sedimentation basin effluent appears to be to improve the basin, by maximizing its size, and installing baffles and inlet and outlet devices to inhibit short-circuting. Other improvements to the basin outlet, such as inclined clarifier tubes, a suspended gravel bad, or a wedge-wire screen would further enhance basin performance, although the question of robustness of these improvements at the largely unattended site requires consideration.

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[1]In a quiescent batch column the fractional removal of solids is given by settling velocity of the suspended particles divided by surface overflow rate, $E_c = v/S_c$, where v = settling velocity. In a completely mixed basin the fractional removal of solids is $E_b = 1 - 1/(1 + v/S_b)$. Subscripts c and b refer to column and basin respectively. If the basin is to be designed to remove the same fraction of solids as observed in column tests at a measured surface overflow rate, S_c, then

$$\frac{v}{S_c} = E_c = E_b = 1 - \frac{1}{1 + v/S_b}$$

which reduces to

$$\frac{S_b}{S_c} = 1 - \frac{v}{S_c} = 1 - E_c$$

Spray Decarbonation.- Neutralized mine drainage was sprayed from a height of 15 ft into the inlet side of the sedimentation basin during earlier phases of operation of the pilot plant, as Fig. 26 shows. Only at pH values below about 8 might any appreciable increase in pH be expected as a result of expulsion of carbon dioxide from the water, i.e. when the waste is not strongly chemically neutralized. With the tumbling drum alone in operation, the mean of six readings of pH increase due to the spray was 0.1 units, from a mean pH of 4.7.

Sludge Handling.- An olive-colored sludge blanketed the sedimentation basin floor soon after the start of chemical neutralization, attaining a maximum depth of approx 3 in. (Fig. 32). Sludge minimally tended to gravitate down the 1 in 10 sloped floor of the sedimentation basin, but did not accumulate on the 1 in 1.4 sloped lumber dam. On one occasion sludge was drawn on to the dewatering sand bed; only 60 gal could be drawn before the sludge became excessively dilute. On the sand bed the sludge immediately passed through the sand, without leaving any discernable sludge solids on the sand surface.

Characteristics of the sludge were studied. Samples from the sedimentation basin rapidly thickened, leaving a clear supernatant (Fig. 43). After approx 5 days gravity thickening the total residue was assayed at 13,000 mg/L and free copper at 3,000 mg/L. To study the feasibility of dewatering sludge on dewatering beds during summer, 100 mL samples of raw sludge (pH 10), neutralized sludge (pH 6.3) and deionized water were exposed to the outdoors during September, 1982 in identical jars weighed daily. Raw and neutralized sludge evaporated at the same rate, 15% slower than the rate for water, to produce a dry green-black cake. Gravity dewatering and air evaporation are effective for dewatering sludge produced by chemical neutralization of Walker mine drainage.

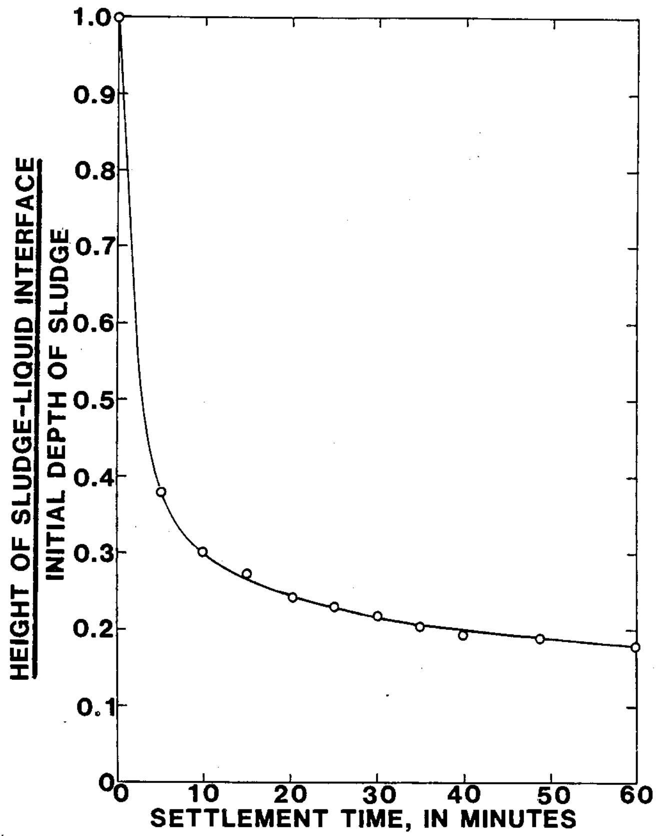
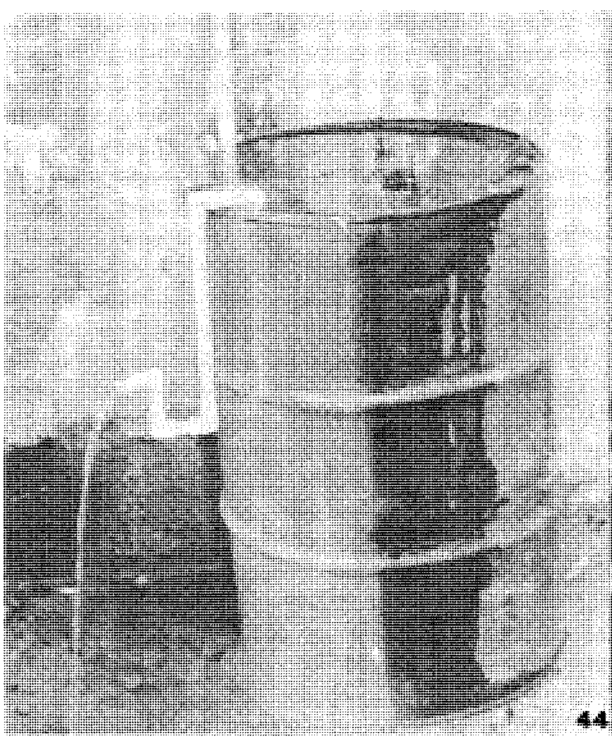


Fig. 43: Gravity Thickening Test Results For Neutralization Sludge.

Copper Cementation.- Mine drainage was piped to the base of a 55 gal drum containing approx 300 lb of detergent-washed steel turnings, as shown in Figs. 44 and 45.



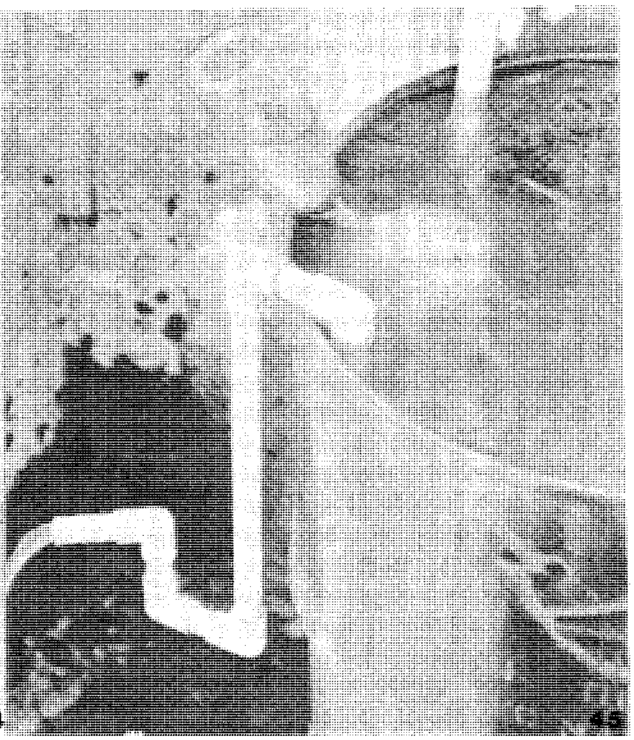


Fig. 44: Copper Cementation Unit.

Fig. 45: Gas Collection From Copper Cementation Unit.

The cementation unit was operated rather sporadically on account of difficulty with flow control. Performance is best characterized by total metals determinations on a single pair of influent and effluent samples (sampling 10) with the following results: copper 16.8 and 10.8 mg/L, zinc 0.7 and 0.5 mg/L, manganese 2.8 and 2.6 mg/L, iron 0.4 and 5.3 mg/L, flow 1.0 gpm and influent pH 5.1. Note the expected approx balance between the molar decrease in copper (94 uM) and the increase in iron (88 uM), and that zinc and manganese were scarcely if at all affected. Otherwise, the unit was monitored by field determinations of flow and influent and effluent pH and free copper, with results listed in Table 10; copper data should be considered cautiously on account of the possible interference by iron in the analysis.

TABLE 10: Copper Cementation Unit Field Monitoring Data.

Sampling serial number	Flow, gpm	Influent pH, units	Effluent pH, units	Influent free copper,mg/L	Effluent free copper,mg/L
10	1.0	5.1	-	12	5
14	0.48	5.0	5.7	10	5
15	1.1	5.1	-	10	6
18	0.029	5.1	6.7	13	0.5
20	0.024	5.0	6.3	8	0.5
32	1.4	5.0	5.6	10	7
33	1.5	5.1	5.2	10	7

The exponential removal rate for free copper calculated from the five data sets with approx 50% free copper removal is approx 0.6 per hour, compared with 0.1 per hour for the two sets indicating approx 95% removal. The slight increase in pH observed may be due to soda ash initially used to wash oil from the steel turnings. An unidentified non-combustible gas was evolved from the unit during later stages of operation.

A local source mentioned that when copper cementation was previously operated at the Walker mine, sediment deposited from high silty spring flows deposited in the unit, inactivating the process.

Evaporation Tanks.- Three swimming pools, nominally 12 ft diam by 3 ft deep, were filled with water and water levels occasionally measured as the distance from a fixed point on the rim of each pool to the water level. Figures 7 and 29 show these evaporation tanks. The first tank was filled on September 2, 1982 with fresh stream water, the second on September 3 with raw mine drainage, and the third on September 4 with mine drainage neutralized to pH 10.1 with soda ash (neutralized mine drainage). Profiles of water level versus time are plotted in Fig. 46.

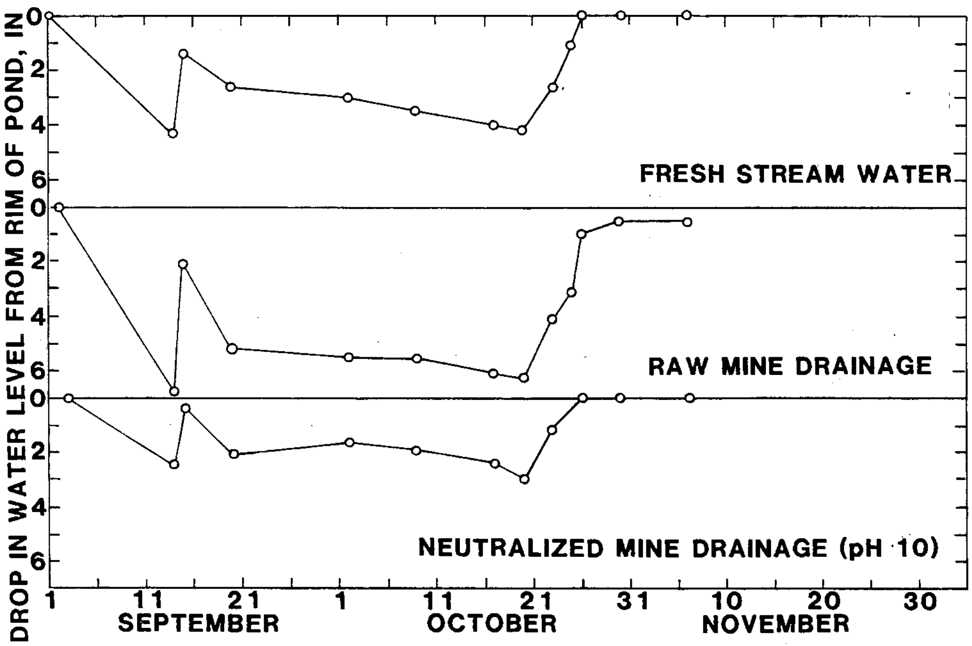


Fig. 46: Water Level vs. Time in Evaporation Tanks.

By September 14 water levels in the three tanks had fallen 4.3 in., 6.8 in. and 2.5 in. respectively; the raw mine drainage tank with the 6.8 in. fall may have leaked. In the other two tanks, the rate of fall was 0.33 in. per day for fresh stream water, and 0.28 in. per day for neutralized mine drainage. Salts in the latter tank would reduce the evaporation rate. Based on a pan factor of 0.7, the estimated daily lake evaporation would be 0.23 in. per day for fresh stream water, or 0.20 in. per day for neutralized mine drainage. Frequent precipitation after September 14 precluded obtaining further useful evaporation data.

A host of evaporation formulae exist, the Rohwer formula being (12)

$$e = 0.497 (1 - 0.0132 p) (1 + 0.268 w) (1 - r / 100) v \quad (4)$$

where e = evaporation, in. per day; p = barometric pressure, in. of mercury; w = wind speed, miles per hour, r = relative humidity, percent; and v = saturation vapor pressure of water at the ambient temperature, in. of mercury. Using tabulated vapor pressure data (7) and a standard (8) atmosphere pressure at the 6000 ft altitude of the Walker mine of 24 in. of mercury, Table 11 expressing evaporation in terms of temperature, wind speed and humidity was calculated using Eq. 4.

TABLE 11: Theoretical Evaporation at Walker Mine, Inches per Day.

Temperature, ^oF	Wind speed, in miles per hour											
	0			5			10			15		
	Relative humidity, percent											
	30	60	90	30	60	90	30	60	90	30	60	90
30	0.055	0.032	0.008	0.129	0.074	0.018	0.203	0.116	0.029	0.277	0.158	0.040
40	0.059	0.034	0.008	0.137	0.078	0.020	0.216	0.123	0.031	0.295	0.168	0.042
50	0.080	0.046	0.011	0.187	0.107	0.027	0.295	0.168	0.042	0.402	0.230	0.057
60	0.119	0.068	0.017	0.279	0.159	0.040	0.439	0.251	0.063	0.599	0.342	0.086
70	0.176	0.101	0.025	0.413	0.236	0.059	0.649	0.371	0.093	0.885	0.506	0.126
80	0.251	0.144	0.036	0.588	0.336	0.084	0.924	0.528	0.132	1.261	0.720	0.180
90	0.344	0.196	0.049	0.805	0.460	0.115	1.265	0.723	0.181	1.726	0.986	0.247
100	0.454	0.260	0.065	1.063	0.608	0.152	1.672	0.955	0.239	2.281	1.303	0.326

For example, evaporation of 0.23 in. per day would be indicated by Table 11 at a temperature of 54^oF (12^oC), a relative humidity of 30%, and a mean wind speed of 5 mph, conditions possibly representative of those at Walker mine in early September. Note that because the vapor pressure of water varies clsely with the square of temperature above freezing, the root mean square temperature, rather than mean temperature, is pertinent to estimating rates of evaporation. Particularly if the diurnal and annual temperature ranges are large, as at mountain sites, root mean square temperature is rather higher than mean temperature. For sinusoidal variation root mean square = $(mean^2 + range^2/8)^{0.5}$. Chemical integration of temperature, for example by observing the rate of hydrolysis of sucrose by polarimeter, is a simple method to obtain temperature data (13).

Batch Neutralization.- To supplement chemical dose vs. performance data obtained from operation of the pilot plant, batch neutralization tests were conducted to study the change of pH and the removal of total metals resulting from adding various doses of different reagents to Walker mine drainage. Three reagents were used, soda ash alone, caustic soda alone, and mixed soda ash and caustic soda proportioned such that each contributed equally to the alkalinity of the reagent. A range of 21 doses of each of these reagents was studied, from 9 to 362 mg $CaCO_3$/L expressed as alkalinity. Reagents were spiked into 125 mL aliquots of Walker mine drainage collected on November 15, 1982. After approx 24 hr of quiescent settlement pH was measured in each test aliquot, and an approx 25 mL sample was pipetted from each, acidified, and analyzed for total copper, zinc and other metals by atomic adsorption spectrophotometer. For three doses of each reagent the remainder of the aliquot was filtered through Watman No. 4 filter paper before acidification and total metals determination.

Figure 47 plots resulting pH values vs. reagent dose expressed as added alkalinity, and Figs. 48-50 plot for copper, zinc and manganese respectively, residual total metal as a percentage of initial total metal vs. reagent dose. Figure 48 shows that neutralization to given pH levels below about 8.5 required a smaller dose of soda ash than for mixed reagent, that in turn was smaller than for caustic soda alone; possibly small doses of caustic soda are partially inactivated by absorbing carbon dioxide from the air to form almost neutral sodium bicarbonate. Above about pH 8.5, any further rise in pH of soda ash-neutralized waste was inhibited by the rising buffer intensity associated with the second dissociation of carbonic acid, theoretically most intense at pH 10.3. For example, reagent doses of 127 mg $CaCO_3$/L of alkalinity produced a pH of 10.0 with caustic soda alone, a pH of 9.2 with mixed reagent, and a pH of 8.8 with soda ash alone.

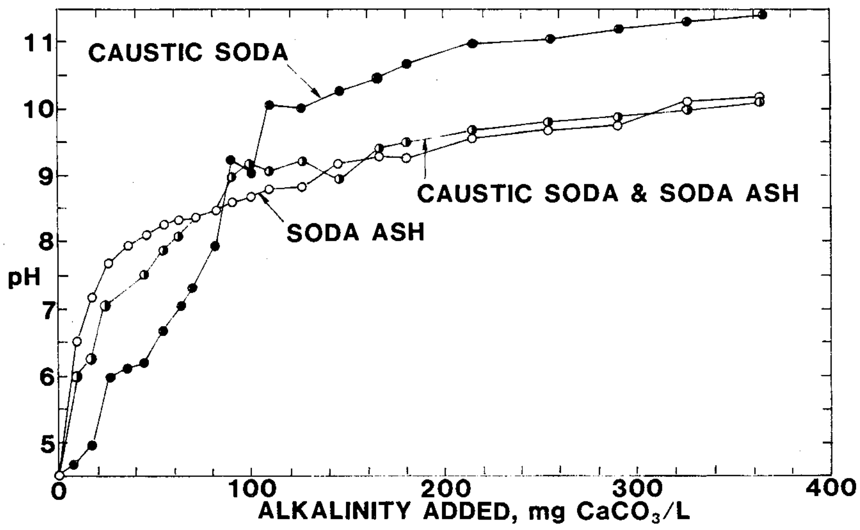


Fig. 47: pH vs. Added Alkalinity for Batch Neutralization.

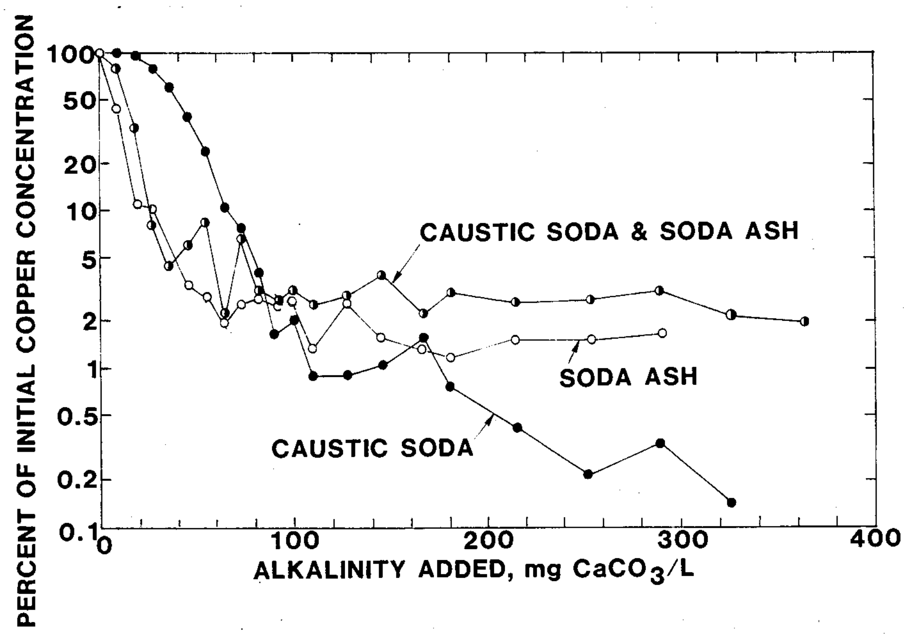


Fig. 48: Residual Total Copper as a Percentage of Initial Total Copper vs. Added Alkalinity for Batch Neutralization.

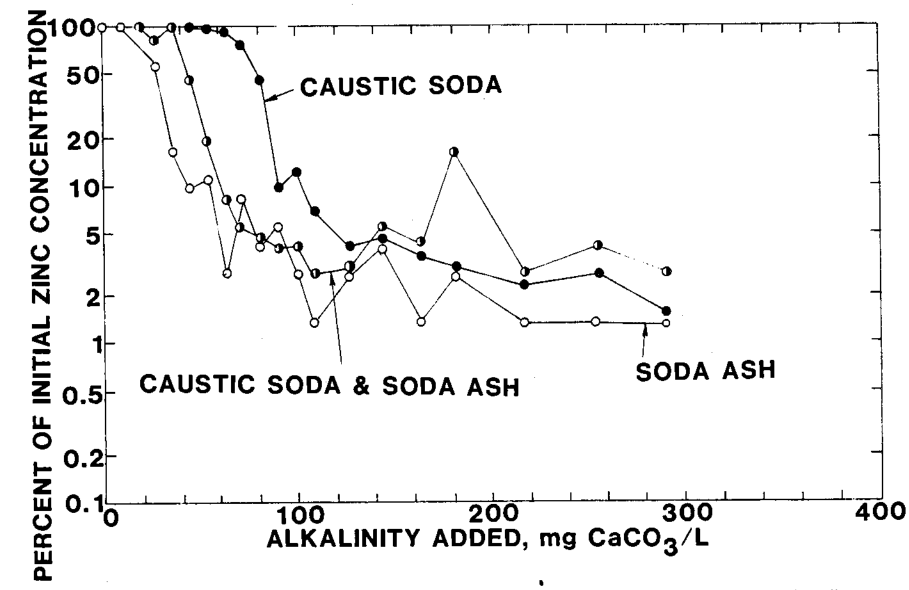


Fig. 49: Residual Total Zinc as a Percentage of Initial Total Zinc vs. Added Alkalinity for Batch Neutralization.

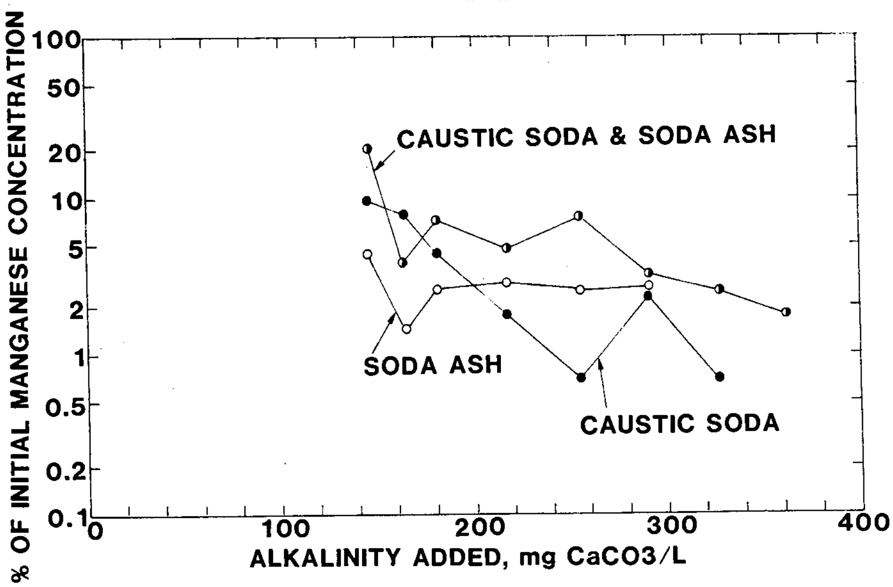


Fig. 50: Residual Total Manganese as a Percentage of Initial Total Manganese vs. Added Alkalinity for Batch Neutralization.

In batch neutralization tests copper removal increased with reagent dose to approx 97% at a dose of 100 mg $CaCO_3$/L as alkalinity for all reagents, thereafter producing a lower marginal removal with further increased dose. For zinc, both soda ash reagent and mixed reagents at 100 mg $CaCO_3$/L as alkalinity produced approx 95% removal, while caustic soda at the same dose produced approx 90% removal of zinc. Again for zinc, marginal removals were lower for reagent doses above 100 mg $CaCO_3$/L as alkalinity. Theory predicts that copper removal is unaffected by the choice between these reagents (at given pH levels), but that zinc removal is higher with soda ash on account of the formation of insoluble carbonate species. Figure 51 compares the theoretical pH-dependent solubility of copper with residual copper vs. pH data from both batch neutralization tests and from operation of the pilot plant in the low flow range.

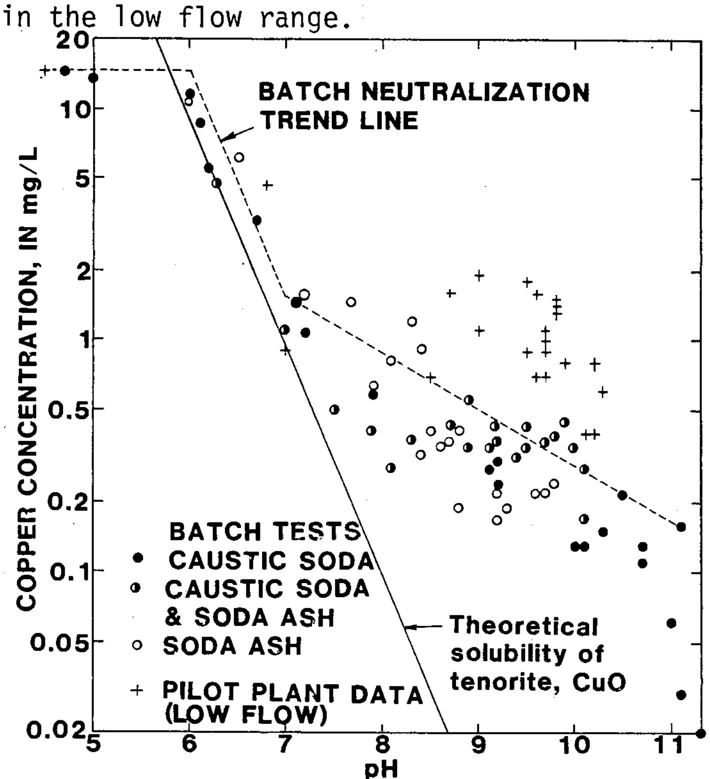


Fig. 51: Residual Copper vs. pH for Batch Tests and Pilot Plant Operation.

Figure 52 plots the theoretical solubility of various metals vs. pH, carbonate concentration and sulfide concentration.

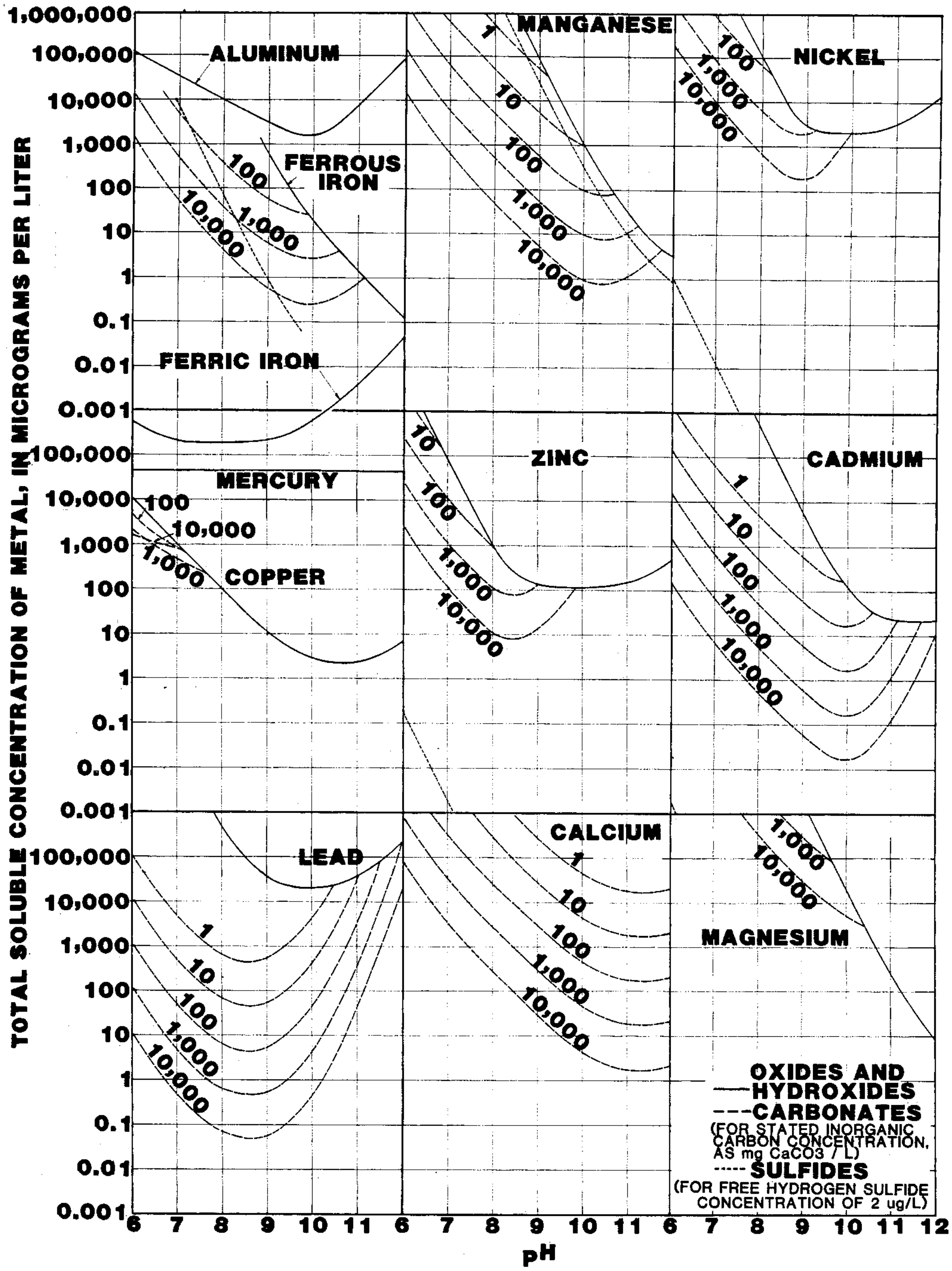


Fig. 52: Theoretical Solubility of Various Metals vs. pH, Carbonate Concentration, and Sulfide Concentration (4).

As Fig. 51 shows, the concentration of total residual copper is unaffected by pH below pH 6, in which range the initial concentration (15 mg/L mean for Walker mine drainage) is less than or only slightly exceeds the theoretical solubility of copper. With increasing pH from 6 to 7 residual total copper in batch test effluent decreases approx ten-fold from the initial concentration, evidently limited by the solubility of copper. Above pH 7 the marginal reduction in total copper with increasing pH diminishes, such that a four-unit pH increase, from 7 to 11, is needed for a further ten-fold reduction to 1% of the initial concentration, apparently limited by kinetics of nucleation and crystal growth. The trend line for batch neutralization in Fig. 51 is

$$\log\left(\frac{c_m}{c_i}\right) = \min\ \{\ 0,\ \max\ [\ 6 - \mathrm{pH},\ 0.25\ (\ 3 - \mathrm{pH}\ )\ ]\ \} \qquad (5)$$

where c_m = minimum attainable residual total copper concentration after neutralization and quiescent settling, mg/L; and c_i = initial total copper concentration, mg/L. The term pH is the pH after neutralization.

Now reconsider Eq. 3, that was based on batch settling tests at pH 11.1, a pH at which Eq. 5 indicates that removal of total copper is constrained to 0.99. Then Eq. 3 expressed in terms of the removal of settleable copper (i.e. copper in excess of the minimum attainable concentration after quiescent settling following neutralization) becomes

$$E = 1 - 0.047\ S^{\,0.32} = 0.99\left(1 - \frac{c_e - c_m}{c_i - c_m}\right) \qquad (6)$$

where c_e = concentration of total copper in sedimentation basin effluent, mg/L.

Figure 53 plots Eq. 6 for influent with 15 mg Cu/L. Because Eq. 6 was developed solely from batch neutralization and batch settling data, it may be verified by comparing its predictions of effluent copper under pilot plant operating conditions to measurements of copper in pilot plant effluent, as in Fig. 54. The mean ratio of actual to predicted copper concentration for pilot plant effluent is 1.04. Table 12 lists batch neutralization data.

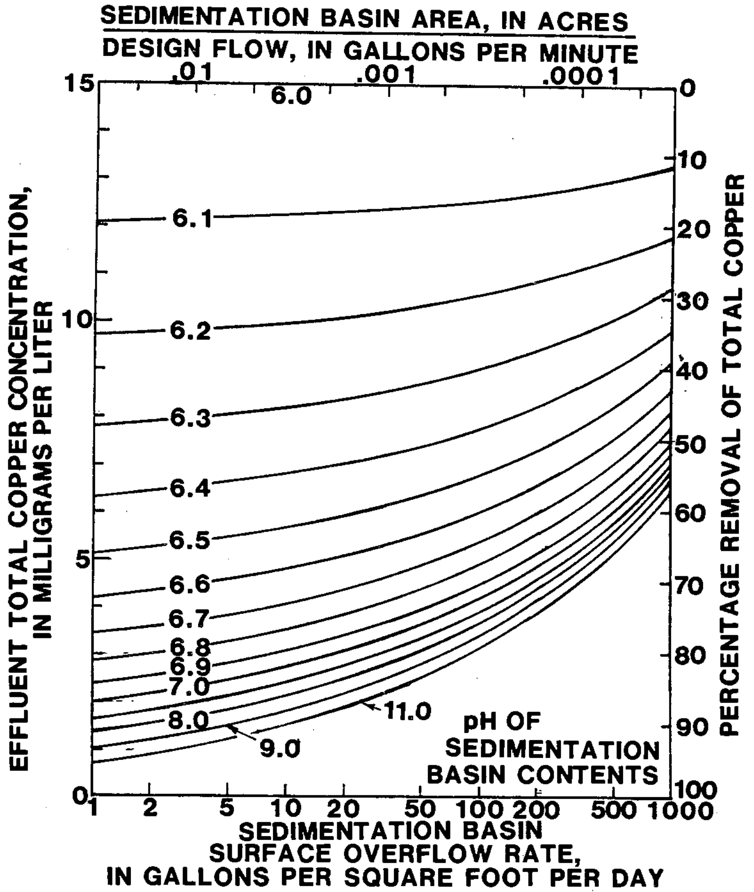


Fig. 53: Sedimentation Basin Effluent Total Copper Versus pH and Surface Overflow Rate.

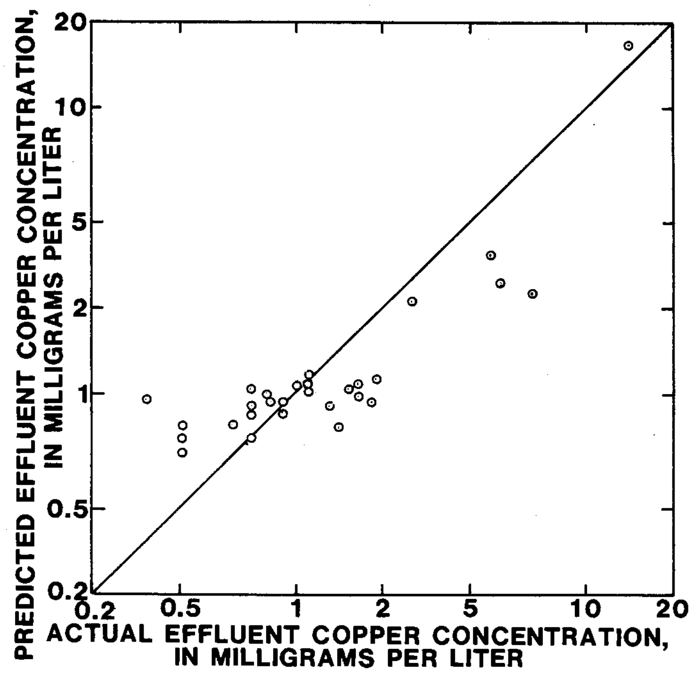


Fig. 54: Predicted Sedimentation Basin Effluent Total Copper Versus Actual Sedimentation Basin Effluent Total Copper.

TABLE 12: Batch Neutralization Test Data

Alkalinity added, in mg $CaCO_3$/L	Neutralization reagent and parameter concentration, in pH units or milligrams per liter of stated metal											
	Soda ash				50% soda ash plus 50% caustic soda				Caustic soda			
	pH	Copper	Zinc	Manganese	pH	Copper	Zinc	Manganese	pH	Copper	Zinc	Manganese
0	4.5	14.3	0.73	2.77	4.5	14.3	0.73	2.77	4.5	14.3	0.73	2.77
9	6.5	6.1	0.74	-	6.0	11.1	0.73	-	4.7	14.3	0.80	-
18	7.1	1.56	0.65	-	6.3	4.7	0.74	-	5.0	13.6	0.75	-
27	7.7	1.48	0.42	-	7.0	1.10	0.61	-	6.0	11.1	0.74	-
36	7.9	0.64	0.12	-	-	-	-	-	6.1	8.6	0.73	-
45	8.1	0.84	0.07	-	7.5	0.50	0.34	-	6.2	5.6	0.72	-
54	8.3	1.23	0.08	-	7.9	0.41	0.14	-	6.7	3.3	0.70	-
63	8.4	0.32	0.02	-	8.1	0.28	0.06	-	7.1	1.4	0.68	-
72	8.4	0.93	0.06	-	8.4	0.37	0.04	-	7.3	1.08	0.58	-
82	8.5	0.41	0.03	-	8.7	0.43	0.03	-	7.9	0.58	0.33	-
91	8.6	0.35	0.04	-	8.9	0.35	0.03	-	9.2	0.24	0.07	-
91[f]	8.6	1.00	0.10	-	8.9	0.45	0.17	-	9.2	0.30	0.20	-
100	8.7	0.37	0.02	-	9.2	0.43	0.03	-	9.1	0.28	0.09	-
109	8.8	0.19	0.01	-	9.1	0.35	0.02	-	10.1	0.13	0.05	-
127	8.8	0.41	0.02	-	9.2	0.37	0.02	-	10.0	0.13	0.03	-
145	9.2	0.22	0.03	0.12	8.9	0.56	0.04	0.54	10.3	0.15	0.04	0.26
163	9.3	0.19	0.01	0.04	9.4	0.32	0.03	0.14	10.5	0.22	0.03	0.21
181	9.2	0.17	0.02	0.07	9.5	0.43	0.12	0.19	10.7	0.11	0.02	0.12
181[f]	9.2	0.19	0.04	0.08	9.5	0.35	0.24	0.11	10.7	0.13	0.04	0.11
217	9.6	0.22	0.01	0.08	9.7	0.37	0.02	0.13	11.0	0.06	0.02	0.05
254	9.7	0.22	0.01	0.07	9.8	0.39	0.03	0.21	11.1	0.03	0.02	0.02
290	9.8	0.24	0.01	0.07	9.9	0.45	0.02	0.09	11.2	0.19	0.01	0.07
326	10.1	-	-	0.41	10.0	0.35	0.01	0.07	11.3	0.02	0.00	0.02
362	10.2	-	-	0.14	10.1	0.28	0.05	0.05	11.4	-	-	-
362[f]	10.2	-	-	0.07	10.1	0.17	0.02	0.05	11.4	-	-	-

STEEP CRUSHED LIMESTONE BARRIER PERFORMANCE

General Description of Barrier.- The 500 ft long vee-section flume is located in the mine area as shown in Fig. 5, set on a mean gradient of 12.6%, and containing an approx 3 in. depth of 1/4-1/2 in. crushed limestone shown in Fig. 41, graded as listed in Table 8. Figure 12 is a reduced engineering drawing of the barrier, and Figs. 55-58 show details of the unit, and of the bucket and stopwatch method to measure the flow treated by the unit.

The 500 ft length of the barrier was selected as the greatest the site could accomodate, while maintaining a flow of 5 gpm in the 1 in. diam polyethylene feed pipe to provide a self cleansing velocity of 2 ft per sec, and to avoid encroaching on flatter country below where the barrier terminated. It is crucial to maintain an adequate velocity of flow within the stone in a barrier to avoid excessive deposition of solids that would inactivate the stone, as operating experience with crushed limestone barriers on various slopes has revealed (14).

Raw mine drainage was delivered to a constant head tank at the inlet end of the barrier where a control valve was used to adjust the flow into the barrier, the remainder of the flow being wasted. The control valve was adjusted such that the water level in the barrier was approximately at the level of the stone surface, thereby maximizing the flow treated, while largely avoiding short-circuting of treatment by flow over the surface of the barrier. Initially the flow treated varied about approx 2 gpm, but as such deposition of solids that did occur within the stone progressed, the flow was reduced to approx 1.1 gpm. Over the three month period of continuous operation of the barrier reported on herein both the hydraulics and the neutralization capability of the barrier stabilized, except for briefly freezing before snow bridged over the barrier, when flow resumed for the winter.

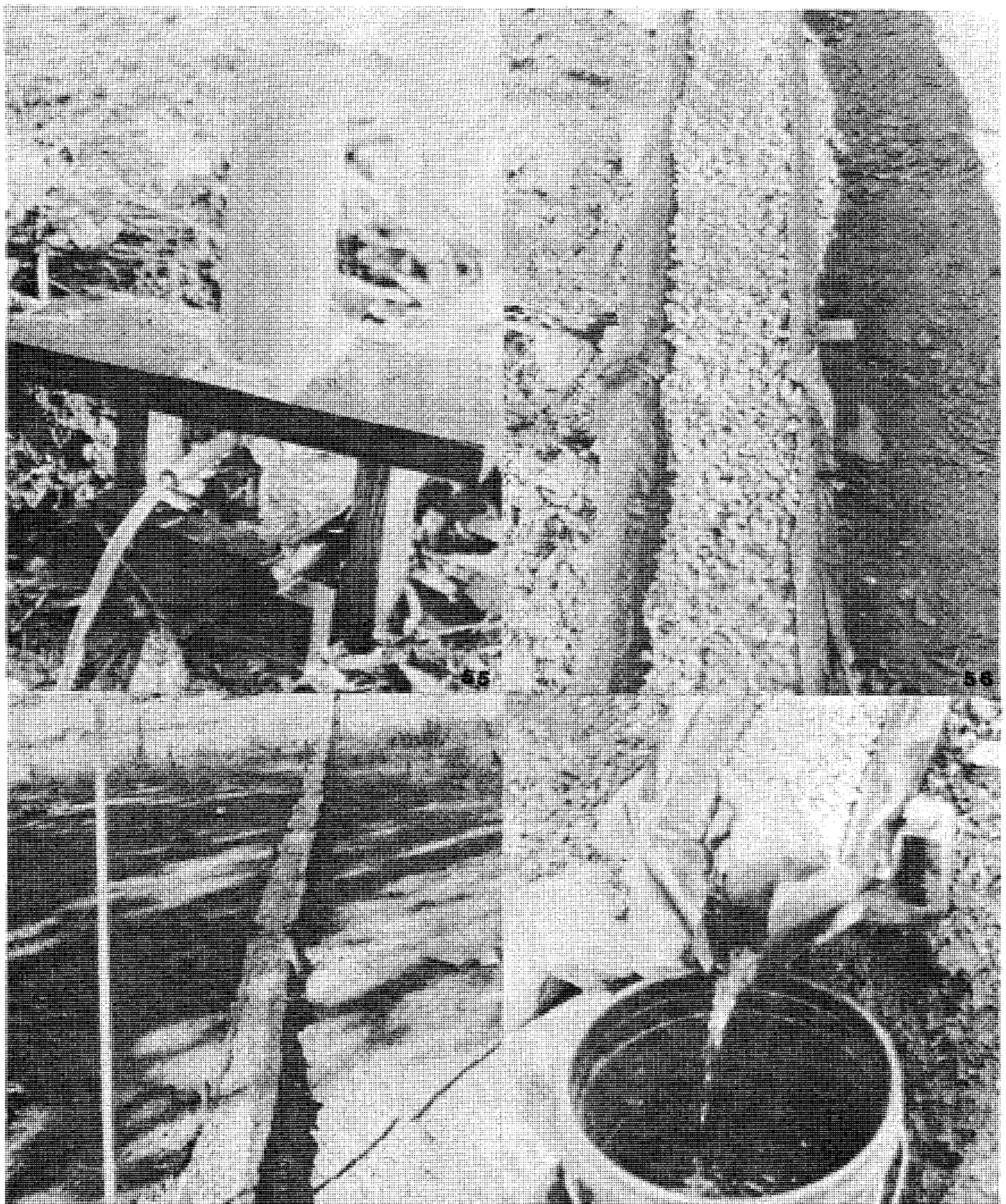


Fig. 55: Constant Head Tank at Inlet of Barrier.
Fig. 56: Typical Detail of Barrier.
Fig. 57: View Upstream From Outlet End of Barrier.
Fig. 58: Bucket and Stopwatch Method of Flow Measurement.

Barrier Hydraulics.- The progress of the initial front of water down the barrier was timed over consecutive 5 ft segments for most of its length, with cumulative time data listed together with the longitudinal profile of the barrier in Table 13. These data may be interpreted in terms of the Carman-Kozeny equation for the flow of fluids through beds of solids (15), that can be reduced to the following form

$$h = \frac{21\ (1 - e)\ V}{s\ d\ e\ g} \left(V + \frac{1030\ \nu\ (1 - e)}{s\ d\ e} \right) \qquad (7)$$

where: h = hydraulic gradient of flow through stone; e = stone porosity; V = actual velocity of flow through stone, ft per sec; s = stone shape factor; d = stone equivalent diameter, in., g = acceleration due to gravity, ft per sec^2; and ν = kinematic viscosity of water, ft^2 per sec. Equivalent diameter equals the mean diameter of a sphere with the same volume as a stone particle, and was here approximated as the geometric mean of the 1/4 in. and 1/2 in. screens used to separate the stone, i.e., 0.35 in. Shape factor is the ratio of the surface area of a sphere of equal volume to the particle to the surface area of the particle, and was taken to be 0.75. Acceleration due to gravity is 32.2 ft per sec^2, and at 10^{o}C (50^{o}F) the kinematic viscosity of water is 1.41 x 10^{-5} ft^2 per sec. Figure 59 shows measured velocities plotted vs. hydraulic gradient, together with lines calculated by Eq. 7 for various stone porosities.

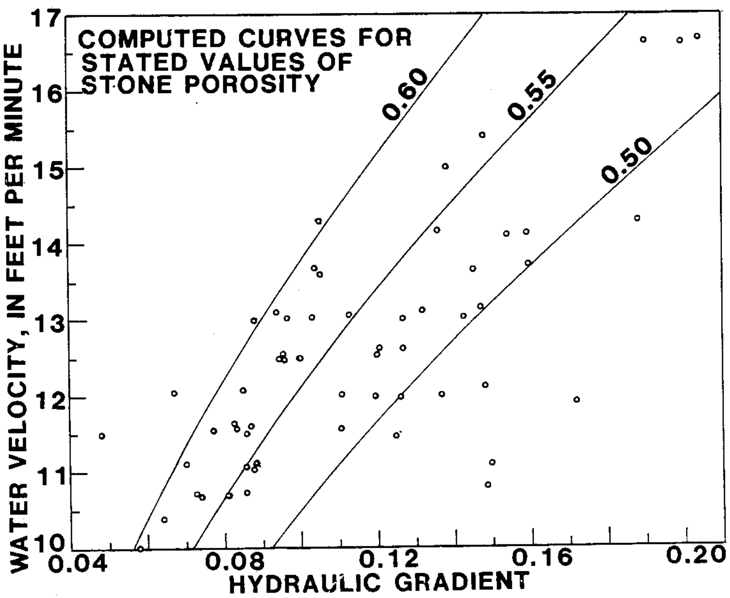


Fig. 59: Computed and Observed Water Velocity In Barrier vs. Hydraulic Gradient.

TABLE 13 Longitudinal Profile and Flow Time for Limestone Barrier.

Distance[a], in feet	Elevation, in feet	Flow time, in minutes	Distance, in feet	Elevation, in feet	Flow time, in minutes
0.0	6172.0	0.00	250.6	6136.2	17.23
2.8	6171.6	-	255.6	6135.7	17.60
7.8	6170.7	-	260.5	6135.0	17.95
12.7	6170.0	-	265.5	6134.5	18.33
17.7	6169.2	-	270.5	6133.9	18.72
22.7	6168.5	-	275.5	6133.3	19.13
27.6	6167.8	-	280.4	6132.6	19.53
32.6	6167.2	-	285.4	6132.0	19.95
37.6	6166.5	-	290.4	6131.3	20.42
42.6	6165.7	-	295.3	6130.3	20.72
47.5	6164.9	-	300.2	6129.4	21.07
52.4	6163.9	-	305.1	6128.4	21.37
57.4	6163.1	-	310.1	6127.7	21.75
62.4	6162.3	-	315.0	6127.0	22.13
67.2	6161.4	-	320.0	6126.3	22.52
72.2	6160.5	-	325.0	6125.8	22.90
77.1	6159.6	-	330.0	6125.3	23.30
82.0	6158.7	-	335.0	6124.7	23.70
86.9	6157.9	-	339.9	6124.2	24.10
91.9	6157.1	-	344.9	6123.7	24.53
96.9	6156.4	-	349.9	6123.0	24.92
101.9	6155.8	-	354.8	6122.6	25.30
106.9	6155.2	-	359.8	6122.0	25.73
111.8	6154.5	-	364.8	6121.4	26.15
116.8	6153.9	-	369.8	6120.9	26.62
121.7	6153.4	-	374.8	6120.6	27.05
126.7	6152.8	-	379.7	6120.2	27.52
131.7	6152.3	-	384.7	6119.8	27.98
136.7	6151.9	-	389.7	6119.4	28.43
141.7	6151.3	-	394.7	6118.9	28.87
146.6	6150.8	-	399.7	6118.6	29.37
151.7	6150.3	-	404.7	6118.3	29.78
156.6	6149.7	-	409.8	6117.9	30.20
161.6	6149.2	-	414.7	6117.4	30.63
166.6	6148.7	-	419.7	6117.1	31.10
171.6	6148.2	-	424.7	6116.6	31.55
176.5	6147.5	-	429.7	6116.4	31.98
181.5	6146.7	-	434.7	6116.1	32.47
186.5	6145.9	-	439.7	6115.7	32.92
191.4	6145.1	-	444.7	6115.2	33.35
196.4	6144.3	-	449.7	6114.8	33.75
201.2	6143.5	-	454.6	6114.1	34.15
206.1	6142.7	14.03	459.6	6113.8	34.60
211.1	6141.9	14.43	464.6	6113.4	35.03
216.1	6141.0	14.73	469.6	6112.9	35.42
221.0	6140.5	15.08	474.6	6112.4	35.78
226.0	6139.7	15.45	479.6	6111.9	36.17
230.9	6139.0	15.82	484.5	6111.3	36.60
235.8	6138.3	16.15	489.4	6110.6	36.97
240.8	6137.6	16.48	494.4	6109.8	37.33
245.7	6136.7	16.90	499.4	6109.1	37.60

[a]horizontal

<u>Treatment Capability of Barrier</u>.- Table 14 summarizes the conditions of operation and the performance of the barrier over a 92 day period of operation, that excludes three days of observations when the barrier was frozen. Table 15 lists detailed operating and performance data for the barrier.

TABLE 14: Summarized Operating Conditions and Performance of Barrier.

Parameter	Design	Mean	SD	Min	Max	No.rdgs
Flow treated, gpm	2.0	1.8	1.3	0.14	5.1	34
Influent pH	4.0	4.9	0.3	4.2	5.2	41
pH increase to 23% of length	-	0.8	0.7	0.1	2.8	33
pH at 23% of barrier length	-	5.8	0.7	4.9	7.8	33
Total pH increase	2.3	2.0	0.6	1.2	4.0	34
Effluent pH	6.3	6.8	0.5	6.1	8.2	34
Load factor at 23% of length[a]	-	47	6	4	3700	30
100% of length	500	680	3	110	12000	35
Removal of total copper, %	-	44	35	-19	97	31
zinc, %	-	15	32	-22	86	30
manganese, %	-	3	18	-21	73	32
iron, %	-	86	15	63	100	11

[a]Geometric mean and logarithmic standard deviation.

Figure 60 plots profiles of flow and pH vs. time over the period of operation. The barrier treated a mean flow of 1.8 gpm, and increased pH from a mean influent value of 4.9 to a mean effluent pH of 6.8, compared to the design pH increase from 4.0 to 6.3.

Neutralization.- The pH increase in a limestone barrier depends on its load factor, a parameter expressing the opportunity for contact between stone and acid water in terms of stone surface area per unit flow, and defined as the number of tons of crushed limestone in the barrier, divided by the stone size in inches, and by the flow treated by the barrier in cubic feet per second. Designed for a load factor of 500, the barrier actually attained a level of performance over the three month operating period equivalent to a geometric mean load factor of 680. Figure 61 plots effluent pH versus influent pH over the operating period, also showing contours of load factor theoretically required to produce specified pH changes from influent to effluent.

TABLE 15: Limestone Barrier Operating Data

Sampling identification: Serial number (1)	Sampling identification: Date (1982) (2)	Sampling identification: Time (3)	Temperature, °C: Acid mine drainage (4)	Temperature, °C: Air (5)	Acid mine drainage total flow, gpm[a] (6)	Limestone barrier influent: pH (7)	Limestone barrier influent: Copper mg/L (8)	Limestone barrier influent: Zinc mg/L (9)	Limestone barrier influent: Manganese mg/L (10)	Limestone barrier influent: Iron mg/L (11)	Flow in limestone barrier, gpm (12)	At 23% of barrier length pH (13)	Limestone barrier effluent: pH (14)	Limestone barrier effluent: Copper mg/L (15)	Limestone barrier effluent: Zinc mg/L (16)	Limestone barrier effluent: Manganese mg/L (17)	Limestone barrier effluent: Iron mg/L (18)	"Free" copper by colorimetric analysis, mg/L: Influent (19)	"Free" copper by colorimetric analysis, mg/L: 23% length (20)	"Free" copper by colorimetric analysis, mg/L: Effluent (21)
1	9/ 2	1400	-	-	-	4.4	-	-	-	-	-	-	-	-	-	-	-	12	-	-
2	9/ 3	1430	-	-	-	4.2	-	-	-	-	-	-	-	-	-	-	-	12	-	-
3	9/ 4	1430	-	-	-	4.2	-	-	-	-	3.5	-	8.2	-	-	-	-	12	-	0.01
4	9/ 5	1430	-	-	120	4.3	-	-	-	-	3.4	-	7.5	-	-	-	-	12	-	1.0
5	9/ 6	1200	-	-	100	4.2	-	-	-	-	3.2	-	7.6	-	-	-	-	12	2.2	0.8
6	9/11	1630	-	-	120	4.9	-	-	-	-	3.0	6.6	7.1	-	-	-	-	13	13	8
7	9/12	1600	-	-	110	4.5	-	-	-	-	2.7	-	6.2	-	-	-	-	13	-	8
8	9/13	1730	-	-	140	4.7	-	-	-	-	2.3	-	6.4	-	-	-	-	13	-	3
9	9/14	1100	6	14	140	5.0	-	-	-	-	2.2	6.4	6.9	-	-	-	-	13	10	7
10	9/15	1000	6	3	110	5.1	16.8	0.67	2.58	0.41	2.2	5.7	6.9	11.3	0.67	2.70	0.00	10	7	3
11	9/15	1300	-	-	120	5.1	17.0	0.69	2.60	-	-	-	-	4.7	0.21	1.10	-	12	-	-
12	9/18	1800	6	6	120	5.1	16.6	0.69	2.55	-	0.53	6.0	6.9	1.8	0.46	2.50	-	12	3	0.4
13	9/19	1300	6	9	100	5.0	16.4	0.67	2.59	0.34	5.1	5.3	6.7	11.4	0.69	2.67	0.00	10	4	3.5
14	9/20	0700	6	15	110	5.0	16.7	0.68	2.55	-	5.1	5.3	6.6	7.5	0.65	2.67	-	10	5	2.5
15	9/21	0700	6	-	115	5.1	-	0.73	2.61	-	2.8	5.5	6.6	14.2	0.68	2.67	-	10	9	7
16	10/ 2	1530	6	11	75	5.0	16.3	0.73	2.67	0.52	0.90	7.8	8.0	0.5	-	2.26	0.04	8	-	0.13
17	10/ 3	1100	6	14	85	4.9	15.8	0.68	2.56	-	3.1	6.8	7.1	3.0	0.67	2.78	-	9	7	2
18	10/ 9	1530	6	12	65	5.1	15.3	0.66	2.58	-	2.2	6.3	6.9	3.5	0.58	2.70	-	13	5	1.8
19	10/10	1430	6	14	70	5.1	15.3	0.68	2.62	0.45	0.66	6.5	7.4	0.9	0.10	2.46	0.00	10	2.5	0.3
20	10/11	1500	6	15	75	5.0	14.9	0.65	2.51	-	0.46	6.7	7.3	0.8	0.09	2.40	-	8	1.2	0.5
21	10/12	1630	6	15	75	4.9	14.9	0.67	2.55	-	1.1	6.5	7.5	0.8	0.16	2.57	-	9	7	0.3
22	10/13	1530	6	14	80	4.8	14.8	0.64	2.60	0.37	1.8	6.4	7.3	1.4	0.10	2.42	0.02	10	5	0.4
23	10/16	1630	6	15	70	4.7	14.5	0.65	2.57	-	1.6	5.4	6.2	6.2	0.65	2.65	-	8	8	4
24	10/17	1230	6	15	80	4.8	14.5	0.65	2.56	-	1.2	5.2	6.4	6.8	0.55	2.62	-	8	5	4
25	10/18	1530	6	13	60	4.8	14.5	0.64	2.59	0.34	1.1	5.4	6.7	3.7	0.52	2.62	0.04	8	6	2.5
26	10/19	1400	6	10	65	4.8	14.3	0.64	2.56	-	1.1	5.2	6.4	7.1	0.58	2.64	-	9	4	4
27	10/20	1000	6	7	60	4.9	14.3	0.64	2.53	-	1.1	5.1	6.3	10.7	0.64	2.60	-	8	6	3
28	10/23	1630	6	6	75	4.7	14.1	0.63	2.59	0.34	1.1	5.7	6.6	7.9	0.58	2.41	0.11	10	8	0.5
29	10/24	1000	6	9	65	5.0	14.0	0.65	2.59	-	1.1	6.7	7.2	10.0	0.63	2.67	-	10	5	1.2
30	10/25	0830	6	4	60	4.9	14.0	0.66	2.61	-	1.1	5.5	6.1	9.8	0.60	2.53	-	14	13	4
31	10/26	1000	-	-	60	5.0	14.7	0.66	2.68	0.37	1.2	5.6	6.5	11.6	0.65	2.66	0.00	11	8	5
32	10/30	1330	6	3	70	5.0	15.2	0.68	2.72	-	1.2	5.2	6.5	10.4	0.61	2.62	-	10	8	3.5
33	10/31	0930	6	0	70	5.1	15.2	0.69	2.74	-	0.32	5.3	6.6	16.5	0.84	3.29	-	10	8	7
34	11/ 6	0200	5	3	75	4.8	14.8	0.67	2.76	0.30	0.70	5.0	6.3	14.3	0.71	2.83	0.11	9	7	6
35	11/ 6	1545	6	8	75	4.8	14.6	0.68	2.71	-	1.2	4.9	6.1	8.7	0.66	2.71	-	9	7	4
36	11/ 7	0700	-	-2	80	5.2	14.6	0.68	2.71	-	0.50	5.5	6.6	14.2	0.73	2.89	-	9	8	6
37	11/ 8	0730	-	-4	75	5.2	14.4	0.66	2.69	0.45	0.14	5.4	6.9	11.3	0.67	3.25	0.15	11	7	6
38	11/13	0130	5	-10	65	5.1	13.8	0.66	2.68	-	0.0[b]	5.4	-	12.8	2.08	2.38	-	9	7	-
39	11/13	1500	6	2	70	4.9	13.8	0.66	2.66	-	0.0	5.3	-	13.7	0.76	2.68	-	10	9	-
40	11/14	0800	6	-2	75	5.0	14.0	0.63	2.75	0.41	0.0	5.3	-	14.0	0.72	2.72	0.11	10	9	-
41	11/15	0715	6	-2	75	5.2	13.9	0.66	2.67	-	0.0	5.4	-	16.6	0.78	2.71	-	9	7	-
42	11/20	1300	6	-2	-	5.1	-	-	-	-	0.55	5.2	6.6	-	-	-	-	-	-	-
43	12/ 7	1500	-	-	-	-	-	-	-	-	0.8	-	6.4	-	-	-	-	-	-	-

[a]treated flow plus bypassed flow metered by a Palmer-Bowlus flume. [b]zero flow indicates that lower part of barrier was frozen.

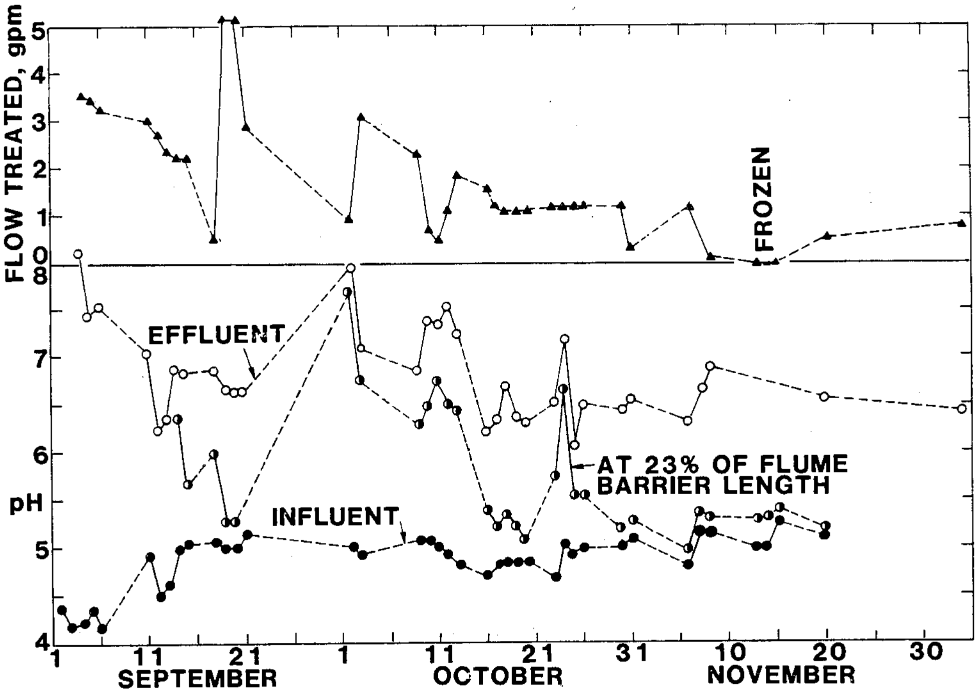


Fig. 60: Flow Treated By Limestone Barrier, and Influent and Effluent pH vs. Time.

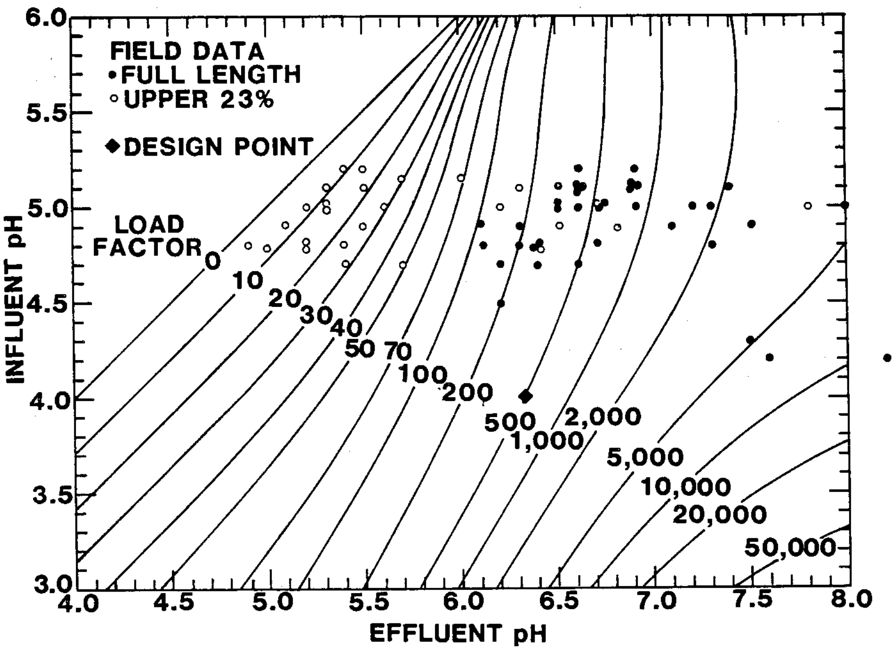


Fig. 61 : Effluent pH vs. Influent pH For Limestone Barrier, With Predicted Relationships For Specified Values of Load Factor.

Figures 60 and 61 show that the highest effluent pH values and the highest load factors occurred immediately after startup, when limestone crusher dust adhering to the stone enhanced performance. During the first week of operation effluent pH declined from approx 8 to near 7, then over the following month to approx pH 6.5, continuing with a mean value of 6.5 and a standard deviation of 0.3 units for the remaining 7 weeks of the reporting period. The barrier continues in operation over the winter. Over the latter 7 week period the mean influent pH was 4.9 with a standard deviation of 0.2, and the geometric mean load factor was 320, with a logarithmic standard deviation of 2.0[1].

Two reasons may explain the reducing reactivity of limestone during the acclimation period. First, the fall of mean daytime temperature from 12°C during the acclimation period to 5°C mean thereafter would cause a 40% reduction in reactivity, assuming as typical a two-fold change of reaction rate per 10°C temperature change. Second, copper removed from the water by the barrier formed a green sediment that may be detrimental to barrier performance. Figures 62 to 65 show barrier influent and effluent concentrations vs. time profiles for copper, zinc, manganese and iron.

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[1] Above pH 5, the central tendency of a set of load factor values is more accurately described by the geometric mean than by the arithmetic mean, because in this pH range load factor varies somewhat exponentially with pH, as the right side of Fig. 61 shows. The principal disturbance to this pseudo-linearity arises on account of the variation with pH of the buffer intensity of carbonic species introduced to the water from the limestone. In limestone neutralization, as the pH rises towards 6.3 where carbonic buffering is maximal an increased resistance to further neutralization is encountered, represented in Fig. 61 by the crowding of the load factor contours near pH 6. Conversely, near pH 8 where buffer intensity approaches a minimum the contours become more widely spread. However, load factor contours above 1,000 are conjectural, assuming that reaction rate constants observed at lower pH values remain valid in the alkaline range. In contrast to the exponential variation of load factor with pH above pH 5, below pH 4 load factor varies linearly with pH, such that 35 load factor units will achieve a one unit pH increase. Because the presence of non-carbonic pH-buffering species such as aluminum can increase the load factor needed to attain a specified pH increment, batch limestone neutralization tests with the particular water to be neutralized and the particular limestone to be used are advised for design. In principle, load factor curves can be computed from water and stone analyses, provided the limestone surface remains reasonably clean in service. Pilot scale tests are the best predictors of limestone barrier performance.

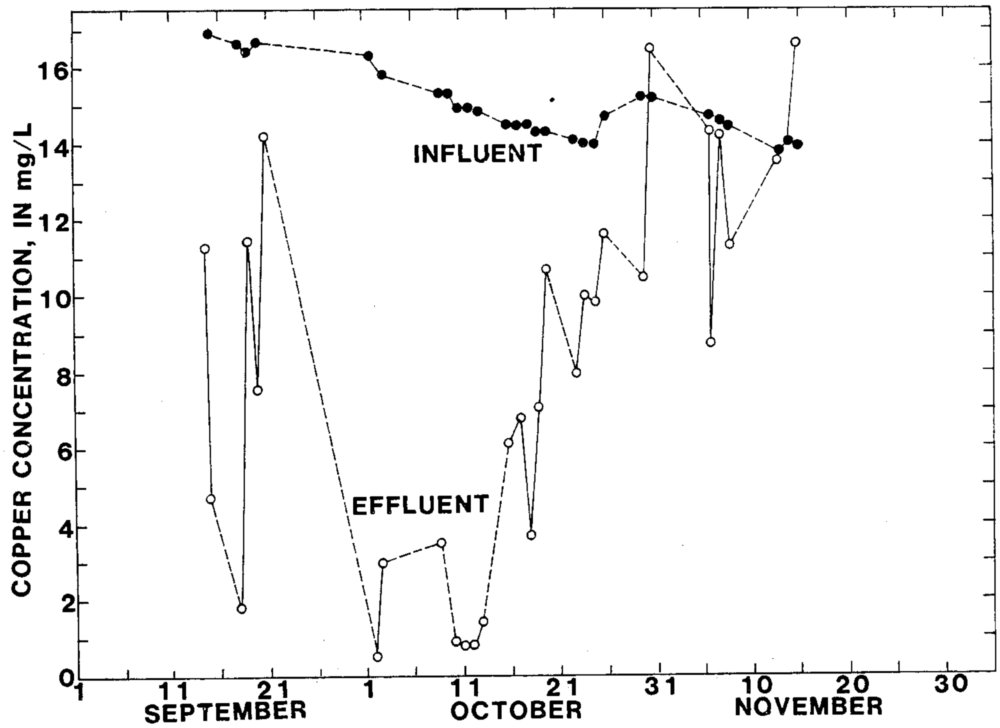


Fig. 62: Limestone Barrier Influent and Effluent Total Copper Versus Time.

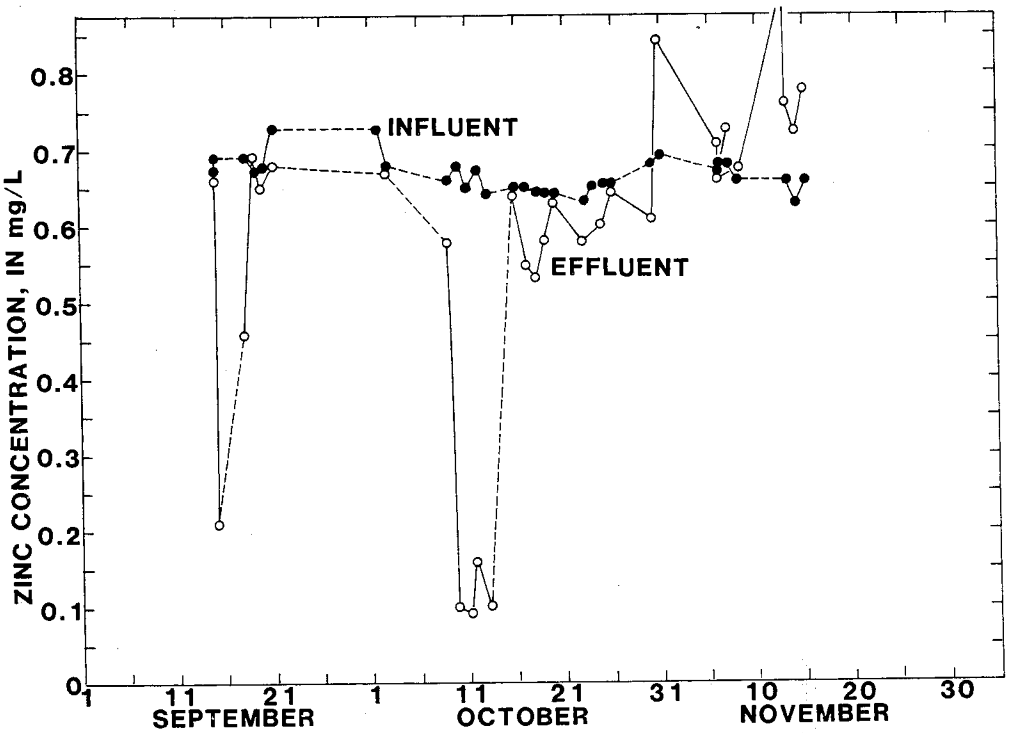


Fig. 63: Limestone Barrier Influent and Effluent Total Zinc Versus Time.

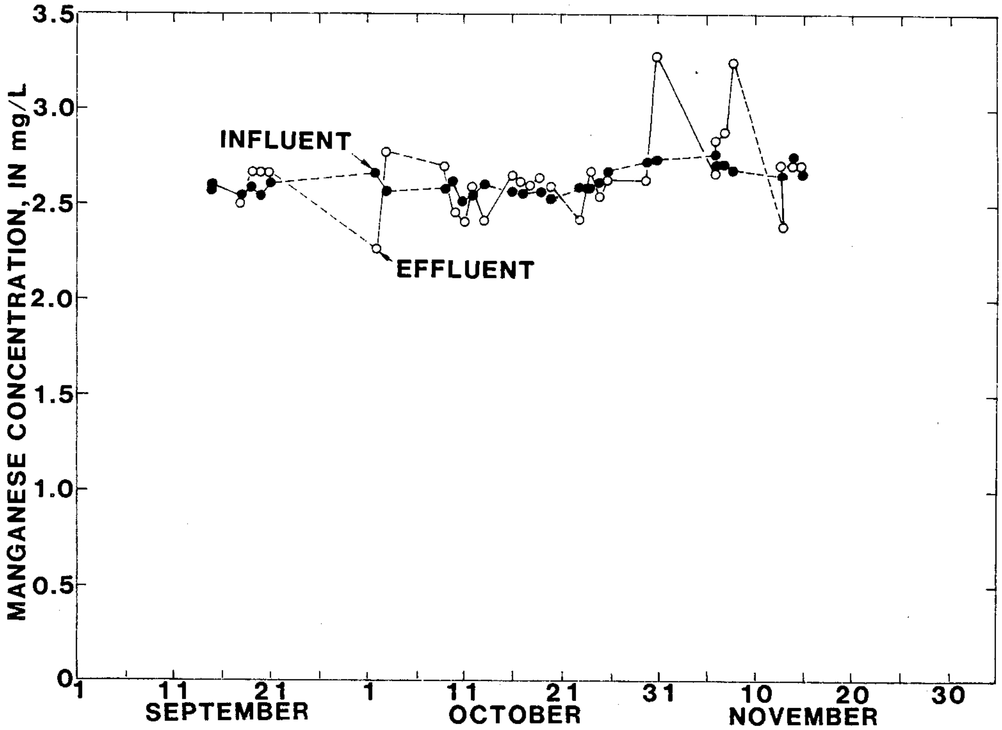


Fig. 64: Limestone Barrier Influent and Effluent Total Manganese Versus Time.

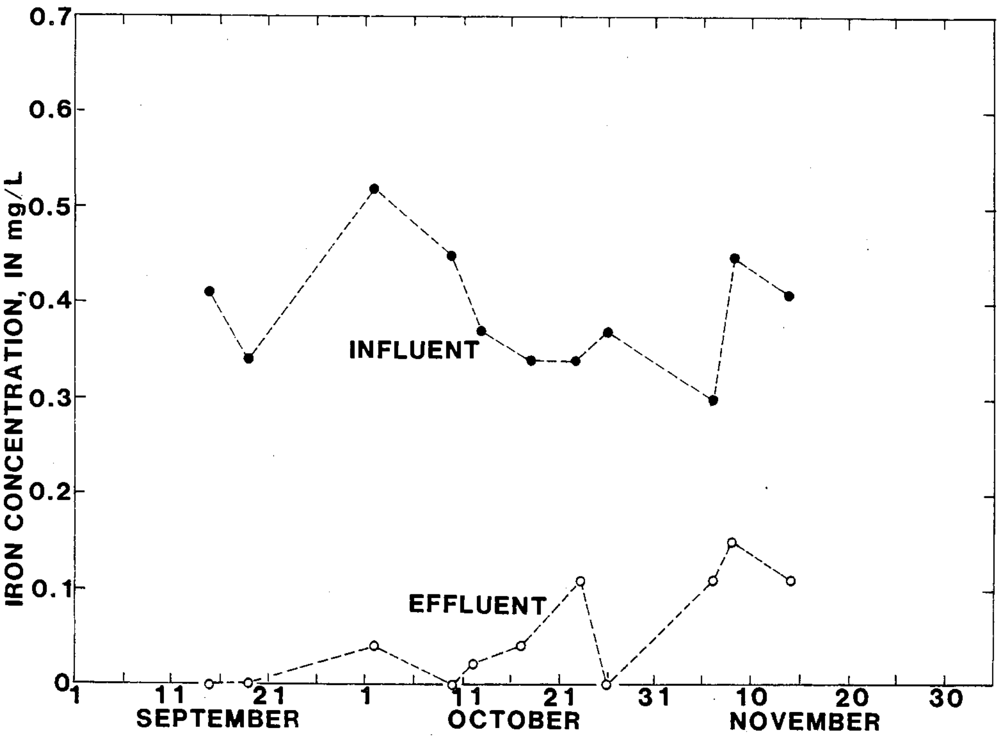


Fig. 65: Limestone Barrier Influent and Effluent Total Iron Versus Time.

Metals Removal.- By comparing initial and final levels of metals in Figs 62 to 65 the barrier is seen to have removed a substantial proportion of copper and iron, but not zinc or manganese. In particular, a maximum removal of approx 90% of total copper to a residual of approx 1.5 mg/L was observed over the period October 2 to 13, at which time Fig. 60 shows the effluent pH to have been approx 7. Recall that batch neutralization by chemical reagents to pH 7 attained approx 90% removal of total copper. After this approx two week period of maximum removal of total copper the barrier continued to produce effluent at approx pH 6.5, while effluent total copper sporadically increased, finally to approximate the barrier influent concentration of total copper.

Early in operation of the barrier a green tinge appeared on the limestone, and a verdigris-colored sediment began to accumulate in the invert of the flume, occasionally to appear as colored flecs in sunlit barrier effluent. An ochre-colored coating formed on the upper surface of the stone over the upper approx 50 ft, that was possibly a bacterial slime such as Thiobacillus sp., although the coating was not examined.

When the lower approx 200 ft of the barrier was found to have completely frozen on November 13, a frozen wedge 2.2 ft long was removed at a point 80 ft upstream of the barrier outlet. This wedge melted down to approx 5 L of loose stone, 0.11 L of limestone flakes and sand, and 3.1 L of a verdigris-colored slurry that was analyzed for total metals. Table 16 lists these analyses, together with loads of these metals in the entire barrier assuming they were uniformly distributed, and loads of metals (computed from water analyses and flow measurements) that entered the barrier and left the barrier up to that time. According to Table 16, 40% to 60% of metals considered to have been removed by the barrier on account of the decreased concentration between influent and effluent may have been stored in the interstitial slurry at the time the barrier froze. Metal-rich sediment settled readily from the meltwater, but the supernatant was not analyzed.

TABLE 16: Metals Recovered In Meltwater From Frozen Segment of Barrier Versus Loads Of Metals Removed From Water Flowing Through Barrier.

Metal	Assayed in slurry from frozen barrier, mg/L	Load of metal, in pounds			
		Estim. total stored in barrier	Influent	Effluent	Influent minus effluent
Total copper	2530	4.0	16.2	7.6	8.7
zinc	17.4	0.03	0.71	0.63	0.08
manganese	51	0.1	1.8	1.8	0.0
iron	160	0.25	0.44	0.03	0.41
Free copper	-	-	10.2	3.2	7.0

Fluctuations in the concentration of total copper in barrier effluent, shown in Fig. 62, suggest that the difference between the estimated load of copper stored in the barrier and the observed removed load of copper would result from sporadic leakage of metal-rich slurry into the effluent, in a manner not reflected by grab sampling as employed. Although the design function of a barrier as envisaged to date is to provide inexpensive preneutralization to reduce consumption of chemical neutralizing reagent, copper chemically altered by exposure to limestone may be largely removable by sedimentation. Insuficient information is presently available for design of a barrier explicitly for copper removal.

Limestone Barrier as a Pre-Neutralization Unit.- To estimate the savings in neutralization chemical resulting from limestone pre-neutralization, it is necessary to convert barrier influent and effluent pH values (measures of intensity, cf temperature) to acidity values (measures of capacity, cf heat). The decrease in acidity through a barrier equals the calcium carbonate equivalent of the mass of neutralization chemical saved by the barrier, per unit volume of water treated. The annual tonnage of calcium carbonate equivalent of neutralization chemical per cfs of flow is 0.984 times the acidity neutralized by the chemical, in mg $CaCO_3$/L. Thus, each mg/L of acidity neutralized per cfs of flow requires 0.98 tons per year of limestone, or 1.04 tons of soda ash, or 0.79 tons of caustic soda. Alkalimetric titration curves, expressing the alkalinity added to various Walker water waters to attain specified pH values, are listed in Table 17.

TABLE 17: Alkalimetric Titration Curves.

Sample collection date (Sept 1982)	Raw mine drainage		Tumbling drum effluent		Sediment-ation basin effluent		Straw filter effluent		Limestone barrier effluent	
	pH	Alk[a]	pH	Alk	pH	Alk	pH	Alk	pH	Alk
2	4.4	0	5.3	0	7.1	0	-	-	-	-
	5.4	20	6.0	23	7.8	4	-	-	-	-
	5.7	30	8.7	42	8.1	7	-	-	-	-
	6.2	40	9.1	48	8.5	8	-	-	-	-
	7.3	47	9.4	54	8.6	11	-	-	-	-
	8.1	51	9.7	61	8.8	13	-	-	-	-
	8.5	52	9.9	68	9.3	25	-	-	-	-
	9.9	83	10.1	73	10.0	56	-	-	-	-
3	4.2	0	4.6	0	4.8	0	-	-	-	-
	5.2	26	5.0	19	5.0	10	-	-	-	-
	6.0	45	5.5	34	5.4	21	-	-	-	-
	8.1	56	7.2	49	5.6	30	-	-	-	-
	8.9	67	8.1	55	6.0	40	-	-	-	-
	9.4	75	8.7	61	8.7	54	-	-	-	-
	9.7	85	9.3	70	9.5	67	-	-	-	-
	10.0	95	10.0	91	10.0	83	-	-	-	-
4	4.2	0	4.5	0	4.6	0	-	-	8.2	0
	5.8	43	5.3	23	5.3	25	-	-	8.6	4
	8.3	54	6.1	43	6.2	43	-	-	9.1	12
	9.4	70	8.7	54	8.6	51	-	-	9.4	21
	9.8	83	9.6	70	9.3	71	-	-	9.7	31
	10.0	89	10.0	83	10.1	81	-	-	10.0	42
5	4.4	0	4.5	0	4.4	0	4.5	0	7.6	0
	4.8	19	5.0	21	5.1	22	5.1	18	8.9	10
	5.6	40	5.6	39	5.9	40	5.7	38	9.5	23
	7.9	53	8.8	58	7.9	48	8.4	57	10.1	43
	9.3	70	9.7	75	10.0	86	9.9	83	-	-
	9.9	88	10.1	96	-	-	-	-	-	-
6	4.2	0	4.6	0	4.6	0	4.6	0	7.6	0
	5.5	37	5.2	20	5.2	21	5.2	21	8.3	4
	7.2	54	5.9	40	6.0	40	5.9	41	10.1	39
	8.7	61	9.1	60	8.9	56	8.2	48	-	-
	9.4	71	9.7	73	9.9	76	10.2	85	-	-
	10.2	97	10.1	84	-	-	-	-	-	-
11	4.9	0	4.9	0	5.0	0	5.0	0	7.1	0
	6.3	30	6.5	34	6.3	30	6.5	31	7.7	8
	8.3	46	8.0	46	8.3	45	8.6	45	9.2	15
	10.0	74	10.1	72	10.3	75	10.2	75	10.3	44
12	4.5	0	4.7	0	4.7	0	4.7	0	6.2	0
	5.8	27	5.9	26	5.8	25	5.9	26	6.7	11
	8.3	51	8.2	46	8.1	45	8.2	46	8.2	18
	9.9	75	10.0	75	10.1	75	10.0	75	10.1	47
13	4.7	0	-	-	9.4	0	9.4	0	6.4	0
	6.1	46	-	-	10.0	36	9.9	31	8.8	14
	9.8	76	-	-	-	-	-	-	9.9	37
14	5.0	0	-	-	9.9	0	9.9	0	6.9	0
	8.9	55	-	-	10.0	13	9.9	13	8.9	18
	10.0	76	-	-	-	-	-	-	10.1	40
15	5.1	0	-	-	-	-	-	-	6.9	0
	8.0	51	-	-	-	-	-	-	7.9	18
	10.0	76	-	-	-	-	-	-	10.6	54

[a]Alkalinity added to attain specified pH, as mg $CaCO_3$/L.

Interpretation of alkalimetric titration curves in Table 17 requires that the curves be corrected for differing initial pH values, mainly resulting from differing points in the treatment processes from which the waters were obtained. Before comparing the curves, each curve is first standardized by subtracting from each alkalinity value on that curve the (interpolated) alkalinity corresponding to a fixed arbitary reference pH, taken as pH 7.5. Resulting standardized titration curves are composited, and plotted as acidity neutralized from pH 7.5 (equal in magnitude and opposite in sign to alkalinity added minus alkalinity added to pH 7.5) versus pH in Fig. 66, being described by the trend line

$$Acy_{7.5} = \max \{ 25(pH-6.5), \min [5(pH-7.5), 25(pH-8.5)] \} \qquad (8)$$

where $Acy_{7.5}$ = acidity neutralized from pH 7.5; and pH = initial pH, $4<pH<11$.

According to Eq. 8, the buffer intensity of the water, equal to the derivative of acidity with respect to pH is 25 mg $CaCO_3$/L-pH unit from pH 4 to pH 11, except from pH 6.25 to pH 8.75 buffer intensity is 5 mg $CaCO_3$/L-pH unit. This buffer intensity is higher than for pure water as shown by Fig. 66, due to weakly acidic species present in Walker water. The more highly charged cations, e.g. aluminum and iron, and weakly acidic anion formers, e.g. carbonic acid, exert acidity in the weakly acid pH range, while less charged cations, e.g. copper, exerts acidity in the alkaline range. According to the reaction

$$Cu^{+2} + CaCO_3 + H_2O = CuO_{(s)} + H^+ + HCO_3^- + Ca^{+2}$$

63.5 g of copper react with 100 g of calcium carbonate equivalent, so that the 15 mg/L of copper present in Walker mine drainage exerts 15 x 100 / 63.5 = 24 mg $CaCO_3$/L of acidity, which is comparable to the alkaline range shift in acidity from the theoretical pure water titration curve shown in Fig. 66 to the observed titration curve.

Figure 67 superposes Eq. 8 on plotted individual titration curves for raw Walker mine drainage, also showing pH values attained by the tumbling drum and by the limestone barrier on those sampling occasions. (These units treated separate streams; they were not in series.) The reduction in acidity due to the drum and the barrier can be read as the abscissa acidity value corresponding to the effluent pH value for the respective units plotted as ordinates. In the case of the limestone barrier the acidity reduction gradually settled to approx 40 mg $CaCO_3$/L for an effluent pH of approx 6.5, equivalent to 40 x 0.79 x 0.5 ≃ 16 tons of caustic soda annually if the mean flow from the Walker mine is approx 0.5 cfs (220 gpm) as believed.

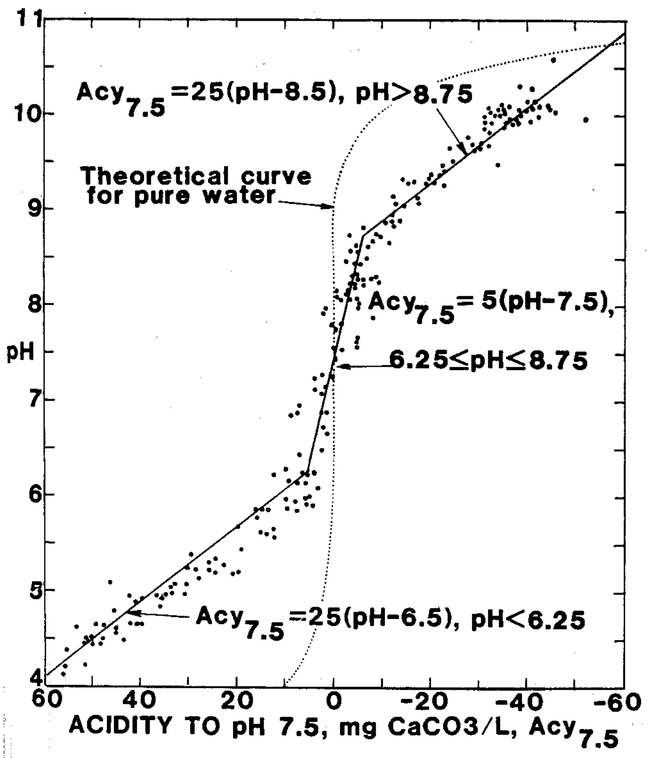


Fig. 66: Composite Alkalimetric Titration Curve For All Process Waters.

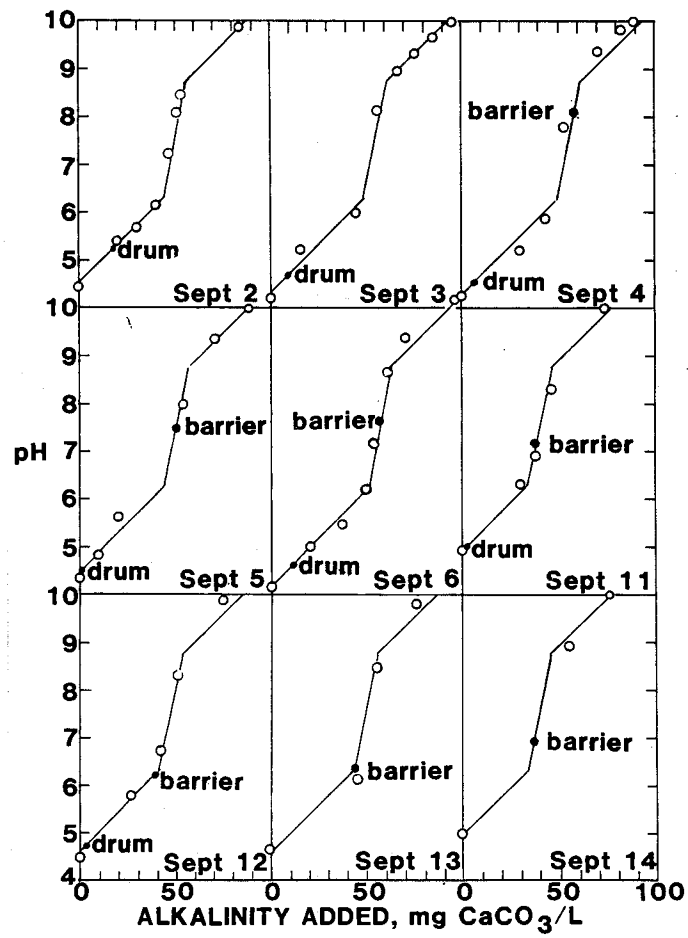


Fig. 67: Alkalimetric Titration Curves For Raw Walker Mine Drainage.

As shown by Figs. 68 and 69, Eq. 8 represented the alkalimetric titration curves for effluent from the tumbling drum and from the limestone barrier as well as it represented the titration curve for raw mine drainage. (There is one exception; the initial effluent from the barrier that was clouded with limestone fines had a titration curve that was not represented by Eq. 8.) The finding that the standardized titration curve for process waters does not vary through the treatment processes supports the use of Eq. 8 for optimization of the proportioning of neutralization between pre-neutralization by the limestone barrier and chemical neutralization to attain the required process pH, for the purpose of economic design of the prototype process train.

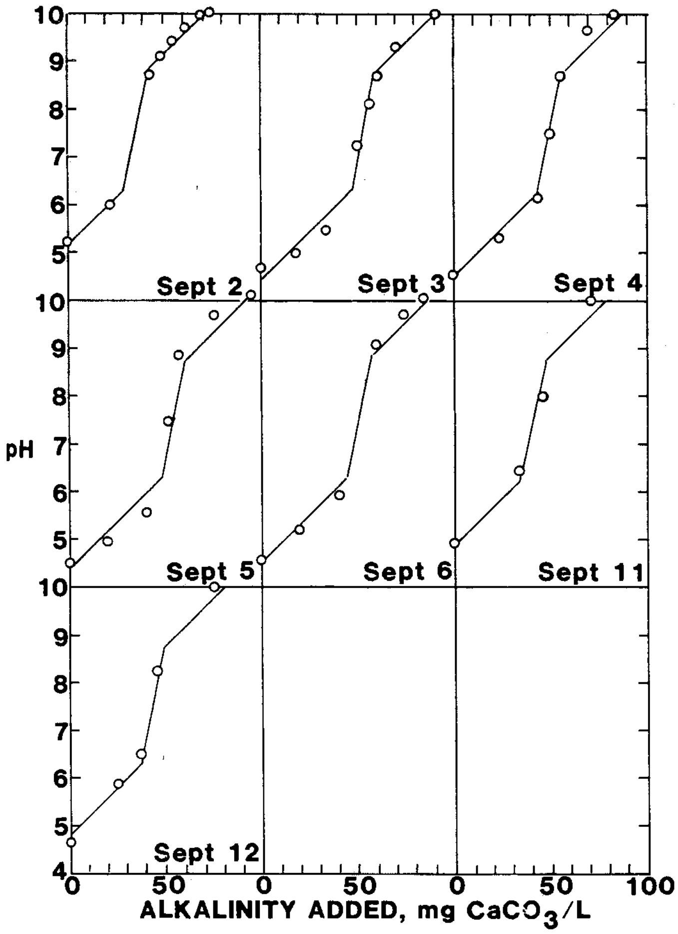


Fig. 68: Alkalimetric Titration Curves For Tumbling Drum Effluent.

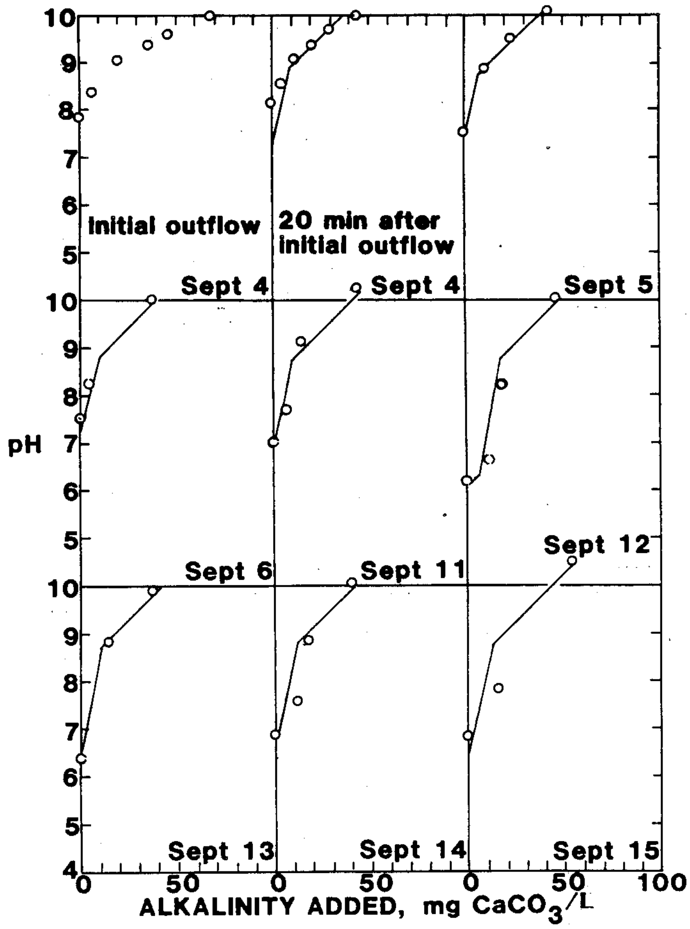


Fig. 69: Alkalimetric Titration Curves For Limestone Barrier Effluent.

ECONOMIC OPTIMIZATION OF PROCESS TRAIN

General Approach.- To remove copper from Walker mine drainage by precipitation and sedimentation, neutralization facilities and a sedimentation basin are needed. Referring to Eq. 6 (p 53), to attain a specified final effluent concentration of total copper, it is necessary to select the degree of neutralization and the efficiency of sedimentation.

Neutralization process pH defines the minimum attainable residual copper concentration following quiescent settlement, c_m (referred to as unprecipitated copper) according to Eq. 5 (p 53). Increasing expenditure on neutralization (at least up to approx pH 11) reduces the concentration of unprecipitated copper, that has a lower limit of the solubility of copper, shown in Fig. 53.

In sedimentation basin effluent the actual concentration of copper exceeds the concentration of unprecipitated copper because quiescent conditions necessary for attainment of the lower unprecipitated copper concentrations are disturbed by currents in the basin and wind-induced turbulence. However, the larger the basin, the closer the actual effluent copper concentration will approach its lower limit, the unprecipitated concentration.

Thus expenditures for neutralization and for sedimentation are in competition. Both processes are needed, but a given effluent concentration of copper can be produced by a range of allocations of total expenditures between the two processes. Under a specified cost structure for neutralization and for amortization of basin construction cost, the particular combination of neutralization pH and basin size is sought that will produce a specified concentration of copper in the final effluent at minimum total cost.

Identification of this optimum requires costing of the neutralization process for various degrees of neutralization (i.e. various pH values).

Of the two neutralization processes available - limestone pre-neutralization and chemical neutralization - one or other or a combination of both must be selected and costed, before trading off the total cost of neutralization against the basin construction cost.

The two neutralization processes are distinct from the viewpoint of chemical kinetics. Chemical neutralization is almost instantaneous in contrast to limestone neutralization; the latter requires comparatively prolonged contact between water and limestone, and therefore a relatively extensive flow-through process unit. Further, while limestone rather rapidly neutralizes acid water below pH 4 (provided the stone remains fairly clean), above pH 4 the size and cost of a barrier to produce further fixed pH increments progressively increases, as Fig. 61 (p 62) indicates.

A situation recurs where amortization costs (for limestone barrier construction) compete with operating costs (for neutralization chemical). Chemical neutralization marginal costs with respect to acidity neutralized are constant, while limestone barrier amortization marginal costs increase with increasing degree of neutralization. For economically optimized neutralization, the condition for equal marginal costs between the two processes needs to be identified to establish the optimal partitioning of neutralization; if marginal costs are never equal then the process with lower marginal cost should be used alone.

Economic, Finance and Cost Assumptions.-

Economic Parameters.- The relative weights assigned to construction costs and operating costs in determining total cost depends on two parameters - discount rate and the economic life of the facility. Discount rate is frequently taken as the interest rate for risk-free investment minus the inflation rate, i.e. the net cost of money. The economic life of a facility ends when either: i) continued costs of upkeep exceed benefits; or ii) upkeep costs exceed costs of alternative technology; or iii) the physical life of the facility has ended.

Preliminary optimization of the Walker mine drainage treatment plant was examined for two sets of values of these parameters: i) discount rate of 10% per year and economic life of 10 years; and ii) discount rate of 5% per year and economic life of 20 years. Corresponding values of the present worth factor are 6.1446 and 12.4622 computed by

$$W = 100 \left[1 - (1 + r / 100)^{-n} \right] / r \tag{9}$$

where W = present worth factor; r = discount rate, %/yr; and n = economic life, yr. For a capital cost of $\$_c$ and an annual operating cost of $\$_o$, the present worth of total cost is $\$_c + W \$_o$, and the annual total cost including amortization of capital is $\$_c / W + \$_o$. One may interpolate approximate optimal costs and approximate optimal design parameter values for values of the present worth factor intermediate between values selected.

Financial Limitations.- The availability of $150,000 for construction of the Walker mine drainage treatment plant was recognized, although, because this limit precludes many economically optimal designs that may be considered desirable, optimal construction costs were not necessarily constrained below this limit. To some extent financial constraints can be minimized by selecting economic parameters corresponding to a low value of the present worth factor, that effectively enhances the value of cash in hand. However, this ruse minimizes the impact of operating costs on optimization, that therefore tend to rise.

Costs.- Unit costs for crushed limestone and for neutralization chemicals used in this analysis are the prices paid in mid-1982 for those supplies used in the pilot plant, assuming that price breaks for future purchase of larger quantities would approximately cancel price escalation.

Prices paid for crushed limestone and neutralization chemicals were:

crushed limestone, 1-1/2 in. 50 percentile size (Fig. 39)	\$10.00/ton
truck rental and gas, 389 miles RT, 20 ton capacity	\$15.40/ton
driver (estimated)	\$20.00/ton
total cost per ton	\$45.40
soda ash, \$330/ton, 1 ton lot, powdered anhydrous	\$350/ton $CaCO_3$ equiv.
caustic soda, \$169/700 lb drum of 50% caustic soda	\$771/ton $CaCO_3$ equiv.

Limestone barrier economics were evaluated on the cost of crushed limestone per unit load factor, equal to cost of limestone per ton; multiplied by the stone size, in.; and by design flow, cfs; and by the ratio of predicted to observed mean load factor from analysis of pilot barrier data (p 63). For a stone size of 1-1/2 in., a design flow of 2 cfs = 900 gpm (cf 830 gpm peak flow recorded during the severe 1981-82 winter), and for predicted and geometric mean observed load factors of 500 and 320 respectively, the cost per load factor is \$45.40 x 1.5 x 2 x 500 / 320 = \$213 / LF. (Note that sizing of physical facilities is based on peak flow, 2 cfs, and that consumption of chemicals is based on mean flow, taken as 0.5 cfs.) Although use of finer crushed limestone would increase the number of load factors per ton, increased cost would be incurred for recrushing, and there is increased risk of damage to the barrier by scouring and by clogging by sediment during sediment-laden high spring thaw flow. Larger stone better resists both scour and sediment deposition, the latter because under a given hydraulic gradient flow velocity increases with stone size (Eq. 7, p 58). Scour and sedimentation will be considered in detailed design of the barrier.

With regard to chemical neutralizing reagents, although the above prices indicate caustic soda to be more expensive than soda ash, estimates of chemical costs were based on the use of caustic soda because in 14% solution it resists freezing better than soda ash (p 16). However, operating experience with soda ash during the current 1982-83 winter may vindicate the use of soda ash. The annual cost of chemical neutralization is 0.984 x flow, cfs x acidity neutralized by chemical, mg $CaCO_3$/L x cost of chemical per ton of $CaCO_3$ equivalent. For caustic soda to treat a mean flow of 0.5 cfs this amounts to \$0.984 x 0.5 x 771 / mg $CaCO_3$/L = \$379 per mg $CaCO_3$/L of acidity neutralized by caustic soda.

If chemical neutralization is used, a chemical neutralization plant is needed. Figure 70 is a conceptual sketch of such a plant, located at the downstream end of the limestone barrier, for which a construction cost of \$30,000 was allowed for the purpose of preliminary optimization.

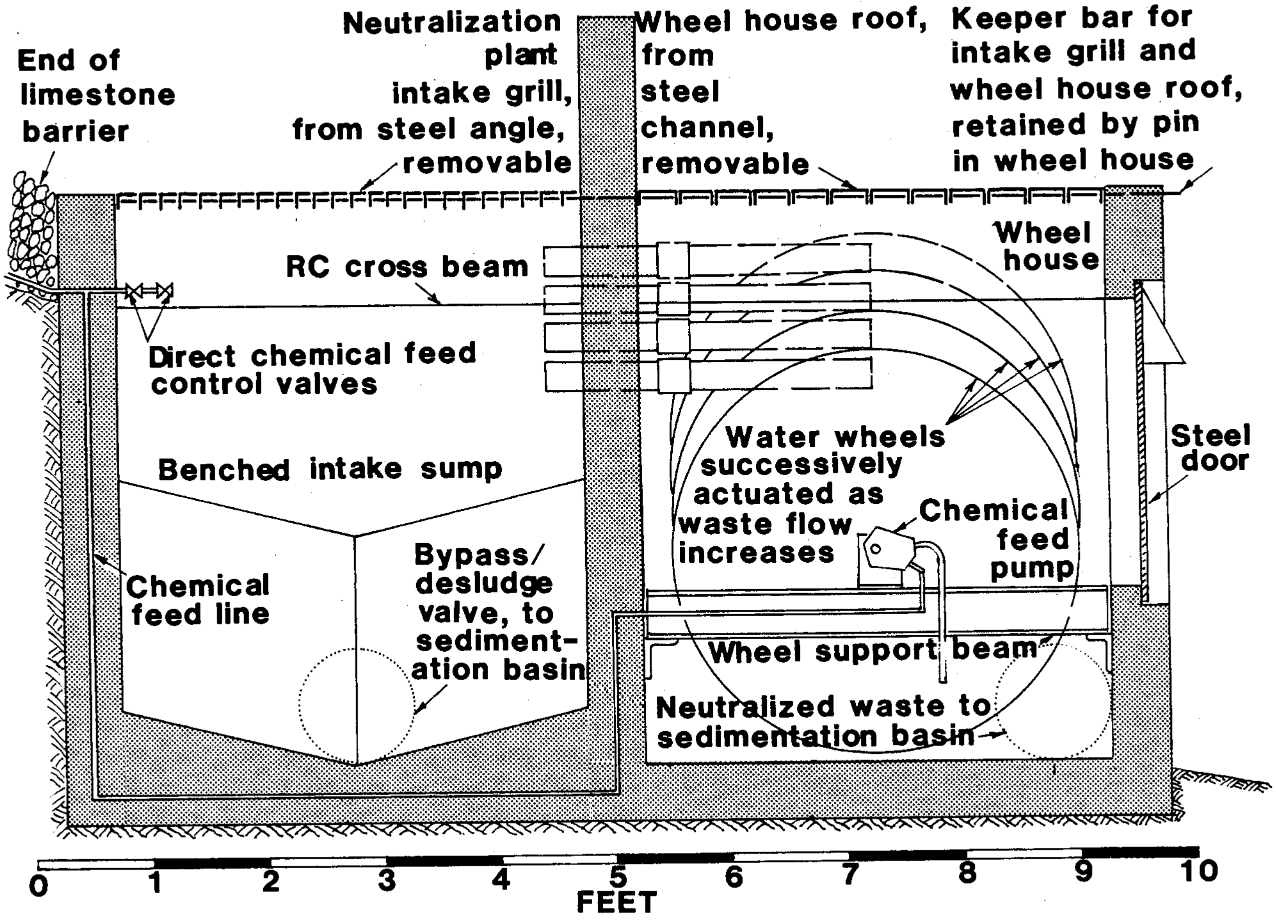


Fig. 70: Conceptual Sketch of Chemical Neutralization Plant.

Figures 71-74 show the vicinity of the neutralization plant site, approx 330 ft below the outlet of the 30 in. pipe discharging from the mine, and near the tailings area where the sedimentation basin would be built.

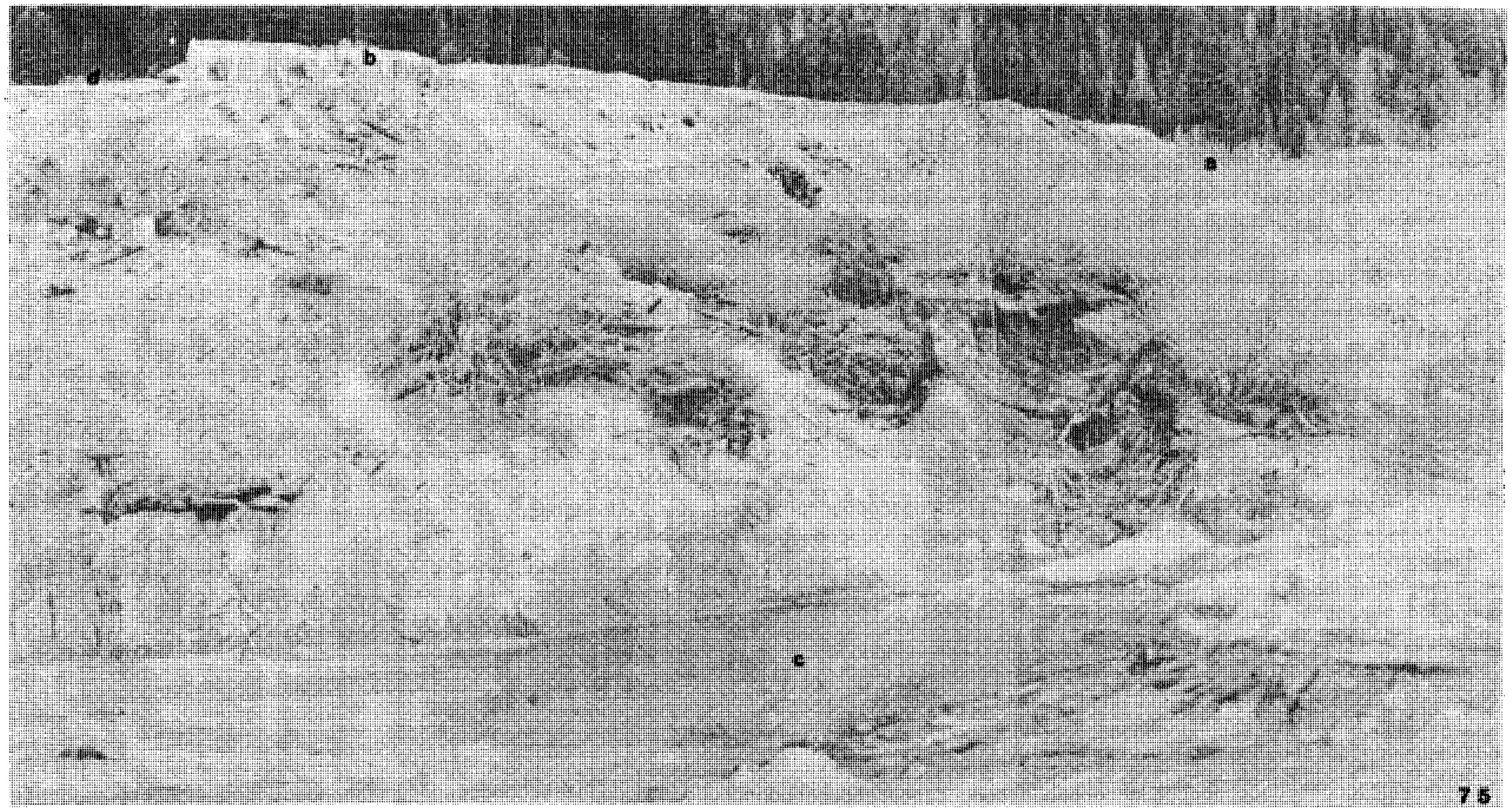
Code: a) end of 30 in. pipe discharging from mine; b) Dolly Ck. confluence; c) 0.05 acre pond;
d) stake shown in Figs. 71-73 near neutralization plant site; and e) pilot limestone barrier.

Fig. 71: Upstream From Vicinity of Neutralization Plant Site.
Fig. 72: Downstream From Vicinity of Neutralization Plant Site.
Fig. 73: Towards Vicinity of Sedimentation Basin Site From Vicinity of Neutralization Plant Site.
Fig. 74: Towards Vicinity of Neutralization Plant Site From Vicinity of Sedimentation Basin Site.

The major item of construction cost is the sedimentation basin. Figures 75-81 are a series of photographs forming a 360^0 panorama taken from a point identified in Fig. 5 (p 7) as near the middle of the word "foundations". The tailings area on which the sedimentation basin would be located is fine sand (tailings) intermixed with occasional boulders. Structural considerations for construction of a dam of this material include provision of a spillway of adequate size and erosion resistance (the size depending on the extent to which surface runoff is diverted from the basin), and avoidance of piping of sand under the hydraulic gradient created by the head of water in the basin. These are matters of detail design, but may be sufficiently intractable to compel consideration of a wall surrounding the basin, e.g. of lumber or steel sheet piling.

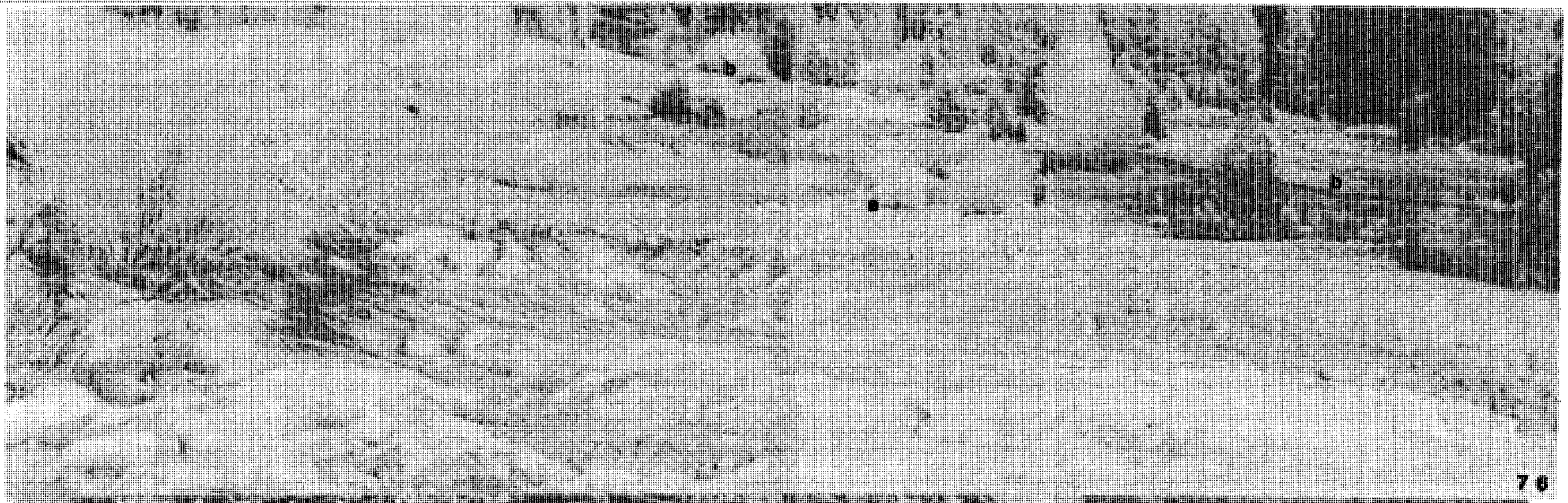
Accordingly, for the purpose of preliminary optimization of the design of the sedimentation basin, costs were considered as alternatively depending on basin area (e.g. as a surrogate for earthworks volume), and on basin peripheral length (e.g. for a surrounding wall). Ranges of unit costs per unit area and per unit length were considered, because construction details, therefore unit costs, await detail design. In fact, the range of unit costs considered exceeds what presently appears to be realistic limits, because apparently outlandish unit costs for sedimentation basin construction were sometimes optimally associated with conditions of chemical neutralization that might attract consideration if not ridiculed by association with possibly unreasonable basin construction costs. Marginal sedimentation basin costs considered ranged from $1,000 to $2,000,000 per acre of basin surface area, or from $10 to $5,000 per lineal foot of basin peripheral length. Walled basins were assumed to be circular.

Marginal Cost Versus Average Cost.- Economic optimization concerns not total costs or average costs of an activity or expense (e.g. moving soil or constructing a wall), but marginal costs for the final unit of that



Code: a) mine discharge point; b) pilot neutralization plant site; c) 0.05 acre pond; and d) 12 in. pipe.

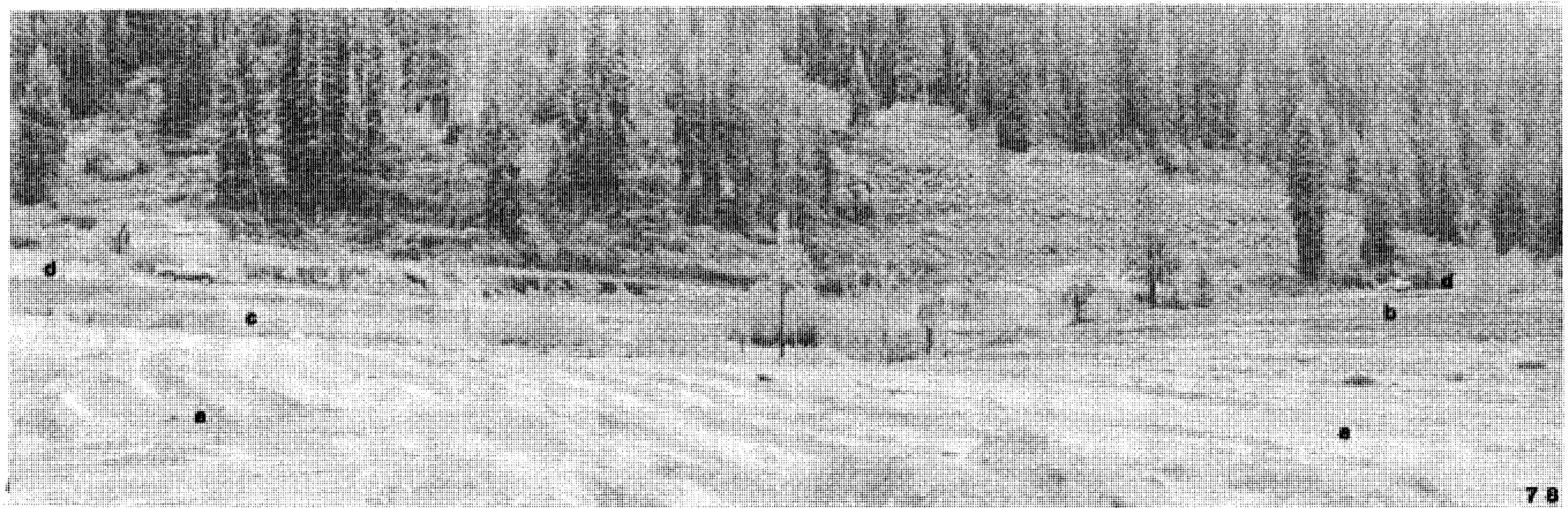
Fig. 75: Towards Mine Portal (hidden) From Reference Point In Sedimentation Basin Area (Panorama part 1).

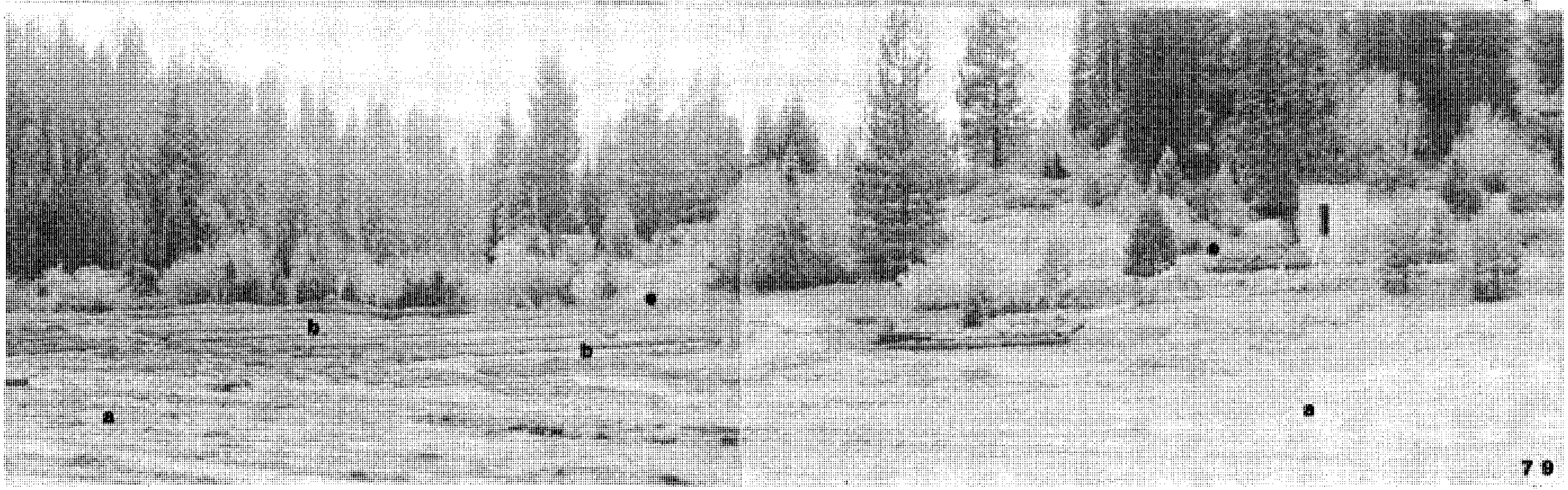


Code: a) stake, near prototype neutralization plant site, shown in Figs. 69-71; b) pilot limestone barrier; c) outlet structure of pond (usually dry) connects by 6 in. AC pipe to Dolly Ck; d) Dolly Ck; and e) road.

Fig. 76: Towards Neutralization Plant Site From Reference Point In Sedimentation Basin Area (Panorama part 2).

Fig. 77: Towards Dolly Creek Upstream Of Walker Mine From Reference Point In Sedimentation Basin Area (Panorama part 3).





Code: a) tailings area; b) abandoned mine drainage treatment channels (5), 60-140 ft long, 1 ft deep, 8 ft wide;
c) Dolly Creek; d) County / U.S. Forest Service road to Portola; and e) Walker mine access road.

Fig. 78: Towards Dolly Ck Downstream of Walker Mine From Reference Point in Sedimentation Basin Area (Panorama part 4)

Fig. 79: Towards Walker Mine Access Road From Reference Point in Sedimentation Basin Area (Panorama part 5).

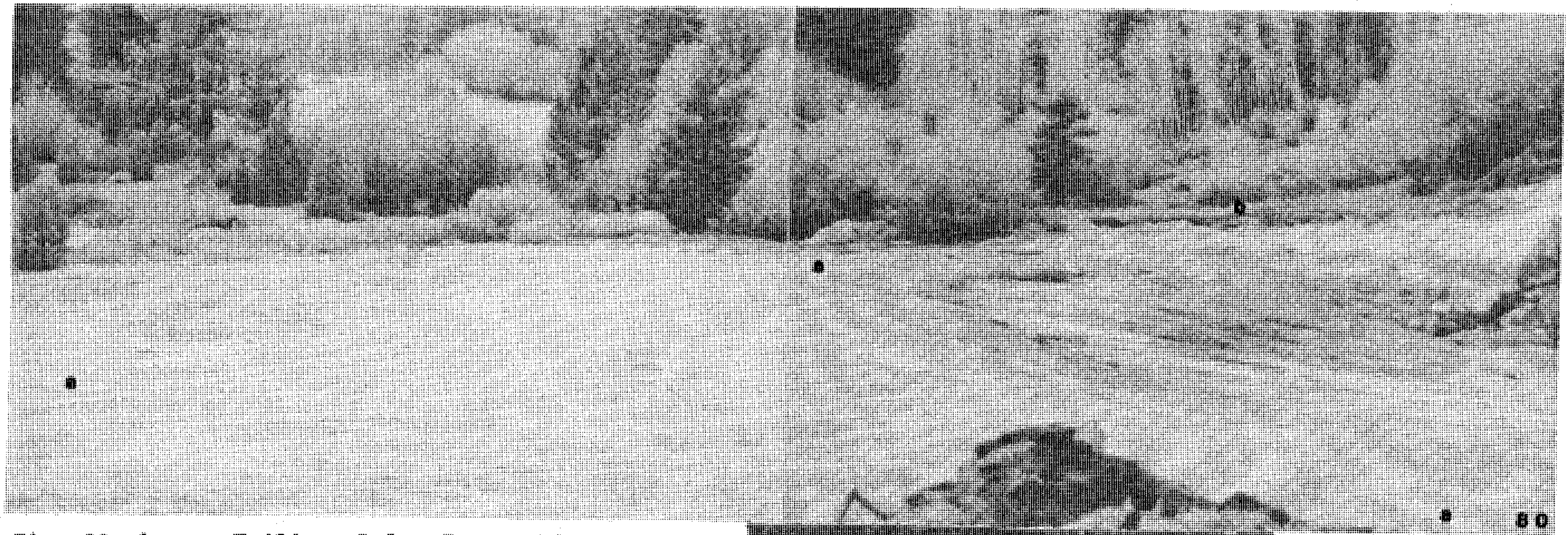
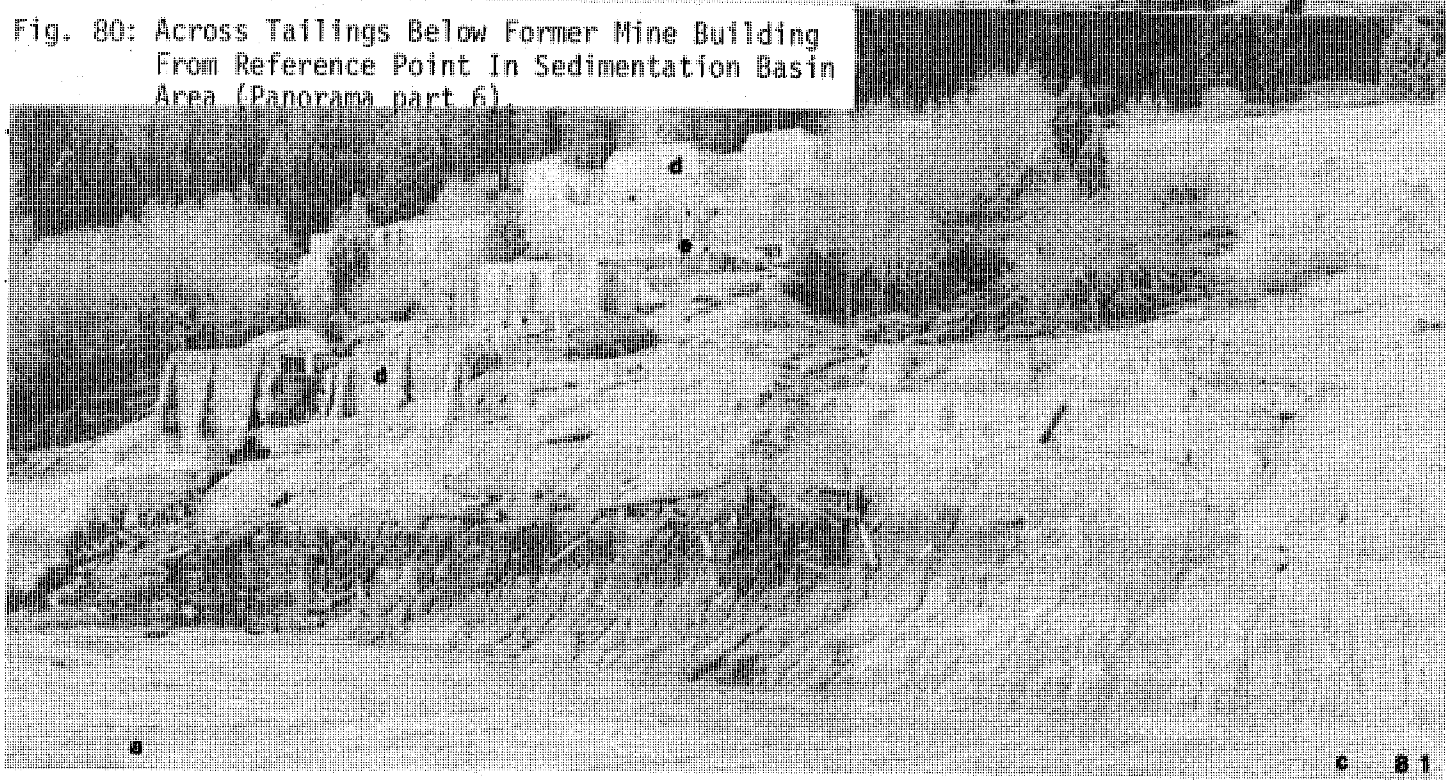


Fig. 80: Across Tailings Below Former Mine Building From Reference Point In Sedimentation Basin Area (Panorama part 6).



Code:
a) tailings area;
b) alternative mine drainage discharge channel;
c) 0.05 acre pond;
d) derelict mine building concrete foundations; and
e) evaporation ponds near pilot plant sedimentation basin.

Fig. 81: Towards Mine Building Foundations From Reference Point In Sedimentation Basin Area (Panorama part 7).

activity or expense (e.g. for the last cubic yard of soil moved or the last lineal foot of wall constructed). For activities costed and paid for by unit pricing, average costs may adequately represent marginal costs, provided fixed costs (e.g. for establishment) are provided for.

To illustrate the pitfalls in marginal costing where the unit costs fail to represent the activity involved, consider marginal costing per acre of a sedimentation basin for which the major cost is a circular surrounding wall, that requires a particular expenditure per lineal foot to construct. If the length of the wall is L ft, the basin area is $A = L^2/(4\pi \times 43560)$ acres. Then the average cost per acre is the unit cost per foot, $\$_{b,\ell}$, $\times L/A = 4\pi \times 43560\ \$_{b,\ell}/L$. However, the marginal cost per acre is $\$_{b,\ell} \times dL/dA = 2\pi \times 43560\ \$_{b,\ell}/L$. The area of a circular basin increases twice as fast with increasing wall length as the quantity obtained by dividing area by wall length. It follows that costing a circular basin as the marginal cost per acre (identified as optimal by economic analysis as subsequently described) multiplied by the basin area in acres would result in a basin cost only one half of actual cost. This explains the need to distinguish between unit areal costing and unit length costing in the following treatment of optimality.

Basis For Cost Computations.- Construction costs for the prototype treatment plant were taken as twice the sum of the following costs:

- number of load factors of limestone in barrier x $213 per load factor;
- construction of chemical neutralization plant (if required) at $30,000;
- provision of first year's supply of chemical at $379 per mg $CaCO_3$/L; and
- unit cost of sedimentation basin x number of area or length units required.

Construction costs implicitly included by doubling the above costs include:

- construction of limestone barrier;
- pipework, and chemical storage and mixing facilities;

- sedimentation basin inlet and outlet structures, and baffles;
- sludge handling facilities; and
- contractor's establishment, supervision, overhead and profit.

The only operating cost considered for the purpose of preliminary optimization was for provision of neutralization chemical. Other operating costs were assumed to be sufficiently independent of variations in design parameter values as not to affect optimization. These may include:

- operator's salary, benefits and insurance;
- maintenance, repair and replacement of components;
- monitoring plant performance; and
- disposal of sludge (perhaps offset by revenue from sale of sludge).

Total cost was computed as construction cost plus the annual cost of chemicals multiplied by the present worth factor reduced by unity.

Optimization of Neutralization.- Neutralization is optimally proportioned between the limestone barrier and caustic soda when the annual marginal cost of neutralization by the barrier equals \$379 per mg $CaCO_3$/L, the marginal cost for neutralization by caustic soda.

To develop an expression for the marginal cost of neutralization by the limestone barrier in terms of barrier design and operational parameters, the barrier performance equation (16) is first written

$$\frac{d\,pH}{d\lambda} = \frac{1}{[H^+]\ln(10)} \times \frac{d[H^+]}{d\lambda} = - \frac{0.0665/\ln(10)}{1 + \dfrac{C_{T_0}(2 + K_1/[H^+]_0) + [H^+]_0 - [H^+]}{(2[H^+] + K_1)\;([H^+] + K_1)}} \qquad (10a)$$

where λ = load factor, tons of limestone per inch of stone size per cfs of flow treated; $[H^+]$ = proton activity, molar; C_{T_0} = initial concentration of carbon dioxide in water = $10^{-4.77}$ molar; and K_1 = first dissociation constant of carbonic acid = $10^{-6.46}$ at 10^oC. The value 0.0665 is a rate constant.

The subscript signifies initial conditions, i.e. $[H^+]_o$ is the initial $[H^+]$.

Figure 82 plots the numerical solution of Eq. 10a, together with data from the Walker pilot barrier after stabilization, and an emperically fitted equation approximating the solution valid for $100 \leq \lambda \leq 1000$:

$$[H^+] = 10^{-6.8} \left(1 + [H^+]_o / 10^{-4.87} \right)^{0.54} \left(500 / \lambda \right)^{0.8} \tag{10b}$$

Figure 83 combines the Fig. 82 solution of Eq. 10a with Eq. 10a itself to produce the required representation of the derivative of load factor with respect to pH in terms of load factor and influent pH.

Recalling from Eq. 8 (p 69) that the derivative of acidity with respect to pH is -25 mg $CaCO_3$/L - pH unit, and that the costs of caustic soda and crushed limestone are \$379 / mg $CaCO_3$/L / yr, and \$213 / load factor respectively, the imputed derivative of load factor can be written

$$\frac{d\,\lambda}{d\,pH} = -W \frac{d\,\$_{n,c}}{d\,Acy} \times \frac{d\,Acy}{d\,pH} \div \frac{d\,\$_{n,b}}{d\,\lambda} = \frac{379 \times 25\,W}{213} = 44.5\,W \tag{11}$$

$\simeq$ 270 for W = 6.14 (10% discount rate, 10 yr economic life)

or 550 for W = 12.46 (5% discount rate, 20 yr economic life)

where $d\,\$_{n,c} / d\,Acy$ = marginal annual cost of chemical neutralization per unit concentration of acidity neutralized, \$ per mg $CaCO_3$/L = \$379; $d\,\$_{n,b} / d\,\lambda$ = marginal cost of construction of barrier per unit load factor, \$ / λ = \$213; and d Acy / d pH = buffer intensity of Walker mine water = -25 mg $CaCO_3$/L-pH unit, $4 \leq pH < 6.25$, or -5 mg $CaCO_3$/L, $6.25 \leq pH < 8.75$.

By entering the values 270 and 550 for $d\lambda/dpH$ as ordinates into Fig. 83, and crossing to an influent pH of 4.0 (observed in June 1981), optimal load factors of 120 and 200 respectively are obtained at the abscissa. Reentering these load factors into the abscissa of Fig. 82 to the pH 4.0 line, optimal limestone barrier effluent pH values of 5.8 and 6.0 are obtained on the ordinate. Limestone neutralization is more economical up to whichever of these pH values corresponds to the preferred economic

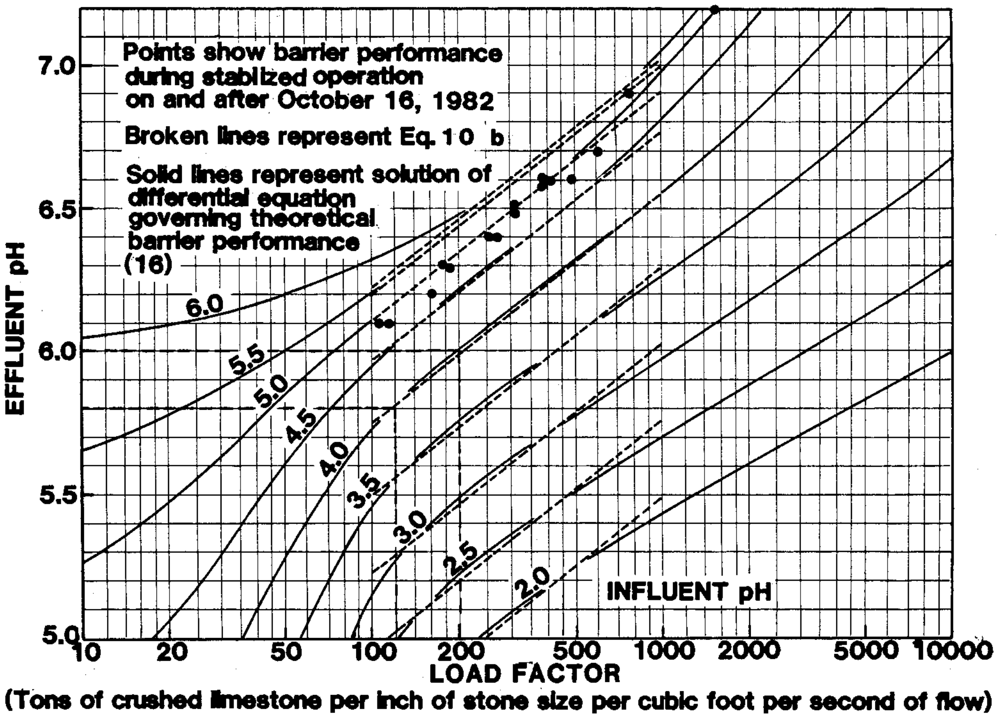


Fig. 82: Effluent pH versus Load Factor and Influent pH

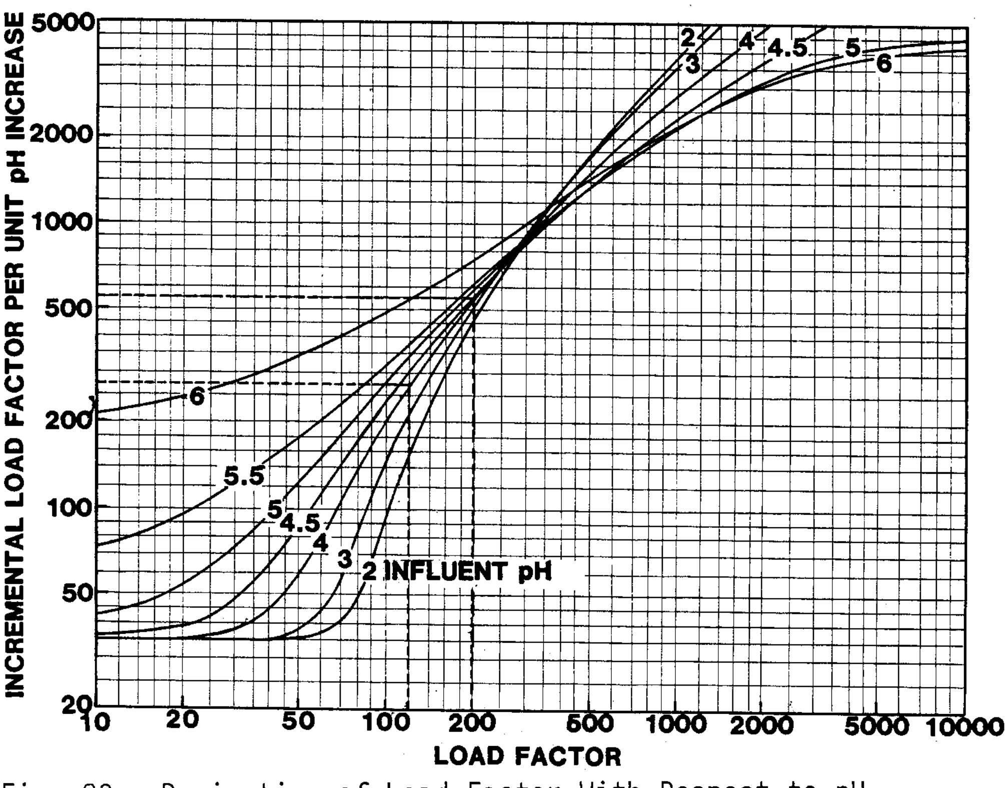


Fig. 83: Derivative of Load Factor With Respect to pH versus Load Factor and Influent pH.

assumption (pH 5.8 for the 10%, 10 yr case, or pH 6.0 for the 5%, 20 yr case), but chemical neutralization by caustic soda is more economical than limestone for neutralization beyond this pH value.

Net savings resulting from installation of these optimally sized barriers are the annual cost of the chemical usage substituted by limestone minus the annual amortization cost of the barrier, i.e. $d\,\$_{n,c} / d\,Acy \times d\,Acy / d\,pH \times (pH - pH_o) - d\,\$_{n,b} / d\,\lambda \times \lambda / W$, where pH_o = initial pH = 4.0. Under the 10%, 10 yr scenario net annual savings amount to

$379 x 25 x (5.8 - 4.0) - $213 x 120 / 6.12 = $12,900 per year

while for the 5%, 20 yr case the net annual savings are

$379 x 25 x (6.0 - 4.0) - $213 x 200 / 12.46 = $15,500 per year.

But these savings can be eroded or reversed by installing a larger-than-optimal barrier, even if the cost of a chemical neutralization plant is thereby eliminated, as shown in Fig. 84.

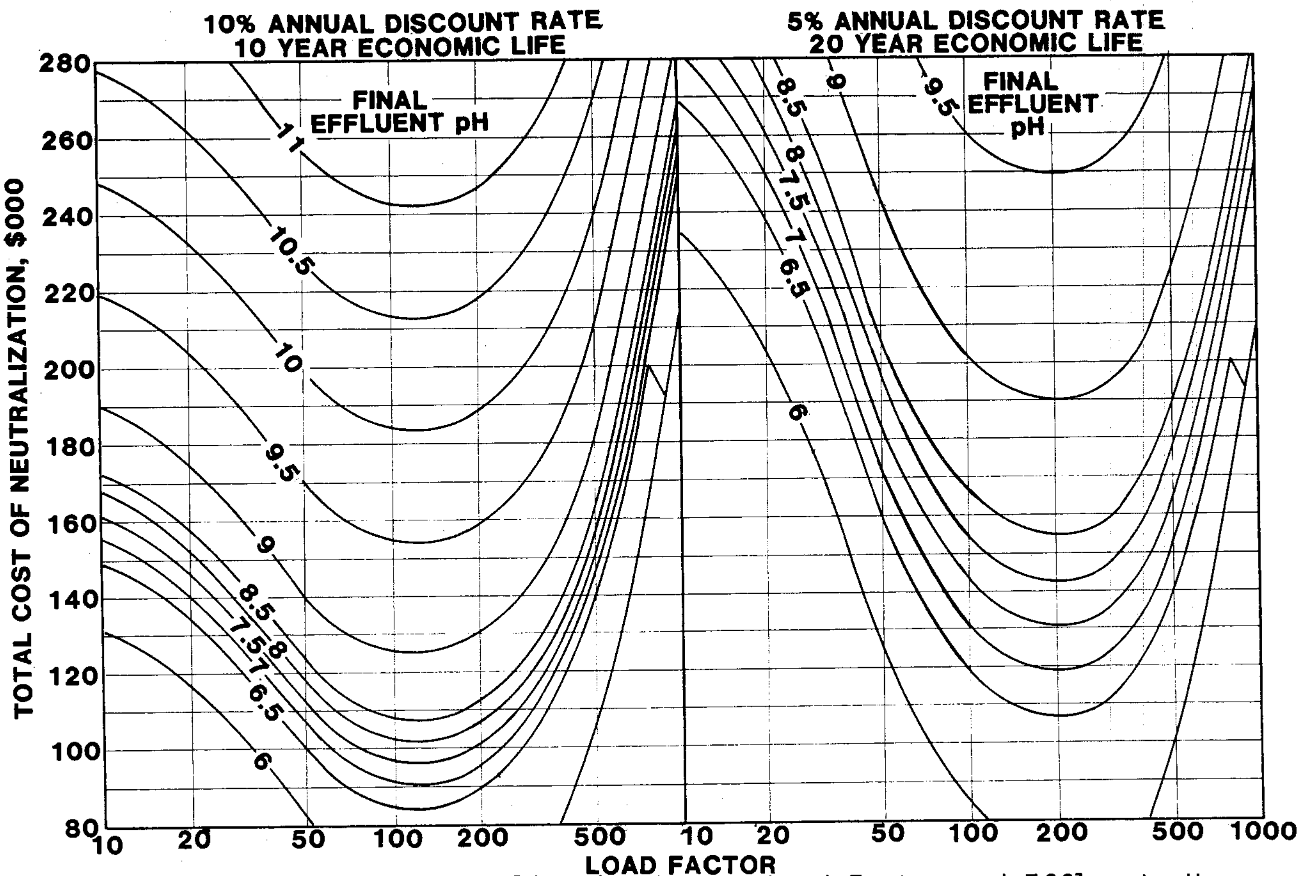


Fig. 84: Total Cost of Neutralization Versus Load Factor and Effluent pH.

For example, if neutralization to pH 6.5 is required, Fig. 82 shows that a load factor of 870 will suffice, without chemical neutralization. Indeed, Fig. 84 shows a break in cost on each of the pH 6.5 lines near a load factor of 870, although total costs still exceed those for optimally sized barriers.

Thus, the minimum total cost of neutralization is

$$\$_n^* = \frac{d\ \$_{n,b}}{d\ \lambda} \lambda^* - W \frac{d\ \$_{n,c}}{d\ Acy} \int_{pH_b^*}^{pH^*} \frac{d\ Acy}{d\ pH}\ d\ pH \tag{12}$$

where $\$_n^*$ = minimum total cost of neutralization; λ^* = optimal barrier load factor; pH_b^* = optimal barrier effluent pH; and pH^* = final effluent pH, associated with a chemical cost that is optimally traded off against the cost of the sedimentation basin, as subsequently described. For Walker mine water, Eq. 8 (p 69) shows d Acy / d pH to be -25 mg $CaCO_3$ / L, $4<pH<11$, except for $6.25<pH<8.75$ d Acy / d pH = -5 mg $CaCO_3$ / L.

Neutralization Cost In Terms of Copper Removal.- Observe from Eq. 5

$$\frac{d\ pH}{d\ c_m} = \infty, \qquad 4\leq pH<6 \tag{13a}$$

$$\text{or } \frac{1}{c_m\ \ln(10)}, \qquad 6\leq pH<7 \tag{13b}$$

$$\text{or } \frac{4}{c_m\ \ln(10)}, \qquad 7\leq pH<11 \tag{13c}$$

Further, the marginal cost of neutralization per unit pH change is by Eq. 8

$$\frac{d\ \$_n}{d\ pH} = \frac{d\ \$_n}{d\ Acy} \times \frac{d\ Acy}{d\ pH}$$

$$= 379 \times 25\ W, \quad 4\leq pH<6.25 \tag{14a}$$

$$\text{or } 379 \times 5\ W, \quad 6.25\leq pH<8.75 \tag{14b}$$

$$\text{or } 379 \times 25\ W, \quad 8.75\leq pH<11 \tag{14c}$$

Combining Eqs. 13 and 14

$$\frac{d\ \$_n}{d\ c_m} = \frac{d\ \$_n}{d\ pH} \times \frac{d\ pH}{d\ c_m} = \frac{379\ W\ f}{c_m\ \ln(10)} \tag{15a}$$

where $f = \infty$, $4.00 \leq pH < 6.00$; or
$f = 1 \times 25 = 25$, $6.00 \leq pH < 6.25$; or
$f = 1 \times 5 = 5$, $6.25 \leq pH < 7.00$; or
$f = 4 \times 5 = 20$, $7.00 \leq pH < 8.75$; or
$f = 4 \times 25 = 100$, $8.75 \leq pH < 11$.

Equation 15a may be more readily visualized when expressed in terms of the marginal cost of neutralization per pound of copper precipitated from the water. Precipitated copper is potentially removable in the sedimentation basin. On the basis that 984 pounds of copper are precipitated annually from a mean flow of 0.5 cfs per mg/L of copper precipitated, the marginal cost of neutralization per pound of copper precipitated is

$$\frac{1}{984 \quad W} \times \frac{d\ \$_n}{d\ c_m} = \frac{379\ f}{984\ \ln(10)\ c_m} \tag{15b}$$

which is plotted versus unprecipitated copper concentration, c_m, and also versus process pH in Fig. 85.

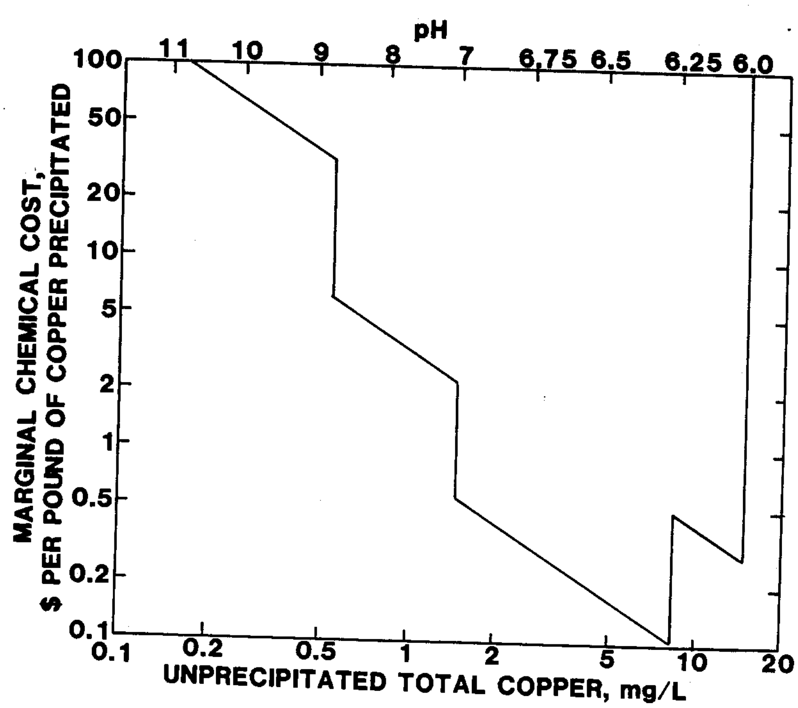


Fig. 85: Marginal Chemical Cost Per Pound of Copper Precipitated versus Unprecipitated Copper Concentration and pH.

Copper Recovery Economics.- Figure 85 implies a possible need for operator motivation beyond maximization of profits from sale of copper in settled sludge, at current market prices for copper about \$0.70 per pound, if more than approx 70% removal of copper from the Walker mine discharge is desired. A knowledgeable operator who pays for neutralization chemical would neutralize only to such a pH that the total marginal cost of recovering copper equals the price received.

For example, if handling costs amount to say \$0.20 per pound of copper precipitated, and 80% of precipitated copper is settled and recovered for sale, and if the sludge sells for \$0.50 per pound of copper, then it would be rational from the profit maximization viewpoint to neutralize to a pH such that the marginal cost of neutralizing chemical is \$0.50 x 80% - \$0.20 = \$0.20 per pound of copper precipitated.

Figure 85 shows that this amount corresponds to a concentration of unprecipitated copper of 4.2 mg/L, so 4.2/0.8 = 5.2 mg/L of copper would appear in the sedimentation basin effluent, corresponding to 65% removal of copper from the mine drainage, initially containing 15 mg/L of copper. This amount also corresponds to a pH (on the upper scale) of 6.55. Assuming the limestone barrier neutralizes to pH 6.0, the concentration of alkalinity required to raise the pH to 6.55 is 25(6.25-6.00) + 5(6.55-6.25) $\simeq$ 8 mg $CaCO_3$/L, at an annual cost of \$379 x 8 $\simeq$ \$3,000. The quantity of copper recovered annually is 80% recovery efficiency x 0.984 tons - cfs / yr x 0.5 cfs mean flow x (15 - 5.2) mg/L copper removed = 3.9 tons. The net revenue generated is \$0.50 x 2000 x 3.9 from sale of copper, minus \$3,000, the annual cost of chemical, minus \$0.20/0.8 x 2000 x 3.9 for handling costs, or -\$1,000.

At a price for copper in recovered sludge of \$2.00 per pound, neutralization to pH 7 would be economically justified at an annual cost for chemicals of \$3,800, to produce an effluent containing 1.5/0.8 $\simeq$ 2 mg/L of copper, and to yield 5.1 tons/yr of copper with a net value after deducting costs for chemical and handling of \$14,000. The interests of water quality may well be best served by substituting for the operator's salary a price support system for copper in sludge harvested from the sedimentation basin, particularly if the price paid per pound of copper increases with the quantity of copper recovered.

Optimal Tradeoff Between Neutralization Cost and Sedimentation Basin Cost.- Equations 5 and 6 (p 53) relate sedimentation basin surface overflow rate to process pH (through the common variable unprecipitated copper concentration, c_m) for any specified final effluent copper concentration, $c_e > c_m$. Surface overflow rate is not a basis for costing, and requires transformation to surface area by the relation $A = 14.84\ Q / S$, where A = basin surface area, acres; and Q = basin design flow, cfs = 2 cfs. Surface area may in turn be transformed to basin peripheral length, using $L = [4\pi \times 43560\ A]^{0.5}$, where L = peripheral length of circular basin, ft, if peripheral length-based costing is pertinent to the construction of the basin.

After transforming surface overflow rate to basin area, Eq. 6 becomes

$$A = \left[\frac{0.139}{1 - 0.99\dfrac{c_i - c_e}{c_i - c_m}}\right]^{0.32} \qquad (16a)$$

for which the derivative with respect to c_m is

$$\frac{d\,A}{d\,c_m} = \frac{0.32 \times 0.99 \times 0.139^{0.32}}{\left[1 - 0.99\dfrac{c_i - c_e}{c_i - c_m}\right]^{1.32}} \times \frac{c_i - c_e}{(c_i - c_m)^2} \qquad (16b)$$

where c_i = concentration of copper in raw mine drainage, mg/L = 15 mg/L. Figure 86 plots the marginal decrease of basin area with diminishing concentration of unprecipitated copper according to Eq. 16b, demonstrating that the required basin size rapidly increases as c_e approaches c_m.

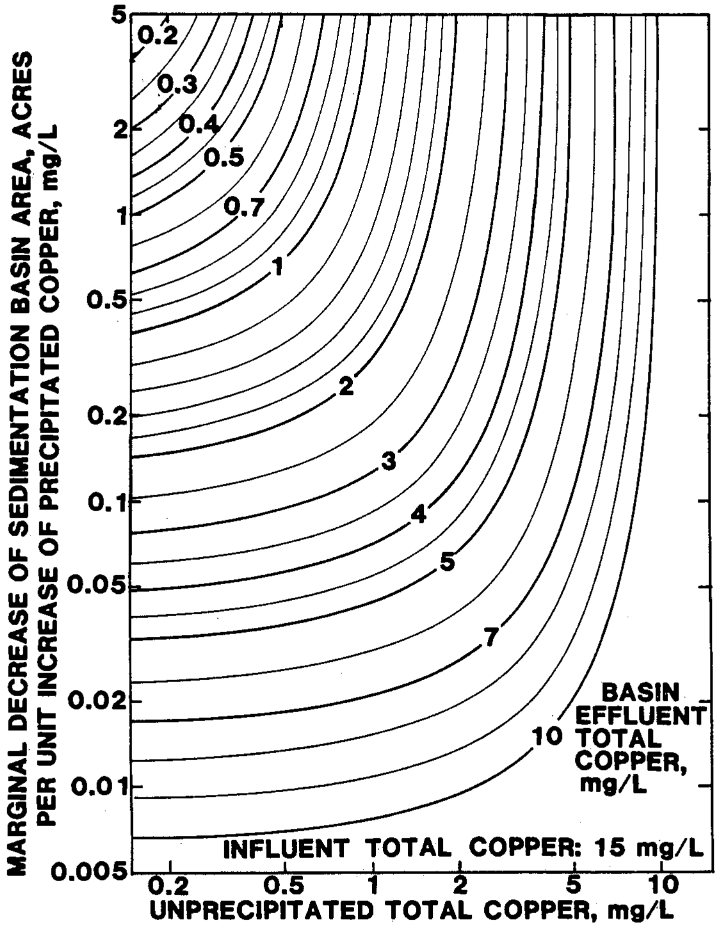


Fig. 86: Marginal Change In Sedimentation Basin Surface Area Per Unit Change In Concentration of Unprecipitated Copper versus Unprecipitated Copper Concentration and Effluent Copper Concentration.

If the marginal (and average) cost of sedimentation basin construction is $d\,\$_{s,a}\,/\,d\,A$ per acre, then the required condition for optimality of equal marginal costs for neutralization and sedimentation is (from Eqs. 15a and 16b)

$$\frac{d\,\$_{s,a}}{d\,A} = \frac{d\,\$_n}{d\,c_m} \times \frac{d\,c_m}{d\,A}$$

$$= \frac{379\ W\ f}{0.32 \times 0.99 \times 0.139^{0.32} \times \ln(10)} \left[1 - 0.99\frac{c_i - c_e}{c_i - c_m}\right]^{1.32} \frac{(c_i - c_m)^2}{c_m(c_i - c_e)} \quad (17a)$$

In the case of sedimentation basin costing per unit length of periphery, recall that

$$L = [\ 4\pi \times 43560\ A\ ]^{0.5} \tag{18a}$$

for which the derivative may be combined with Eq. 16a to produce

$$\frac{d\ L}{d\ A} = \left[\frac{\pi \times 43560}{A}\right]^{0.5}$$

$$= [\pi \times 43560]^{0.5} \left[\frac{1 - .99\dfrac{c_i - c_e}{c_i - c_m}}{0.139}\right]^{0.16} \tag{18b}$$

Then, if the marginal (and average) cost of sedimentation basin construction per unit length of periphery is d $\$_{s,\ell}$ / dL per foot, the condition for optimality of equal marginal costs for neutralization and sedimentation is

$$\frac{d\ \$_{s,\ell}}{d\ L} = \frac{d\ \$_n}{d\ c_m} \times \frac{d\ c_m}{d\ A} \times \frac{d\ A}{d\ L}$$

$$= \frac{379\ W\ f\ [\pi \times 43560]^{-0.5}}{0.32 \times 0.99 \times 0.139^{0.16} \ln(10)} \left[1 - 0.99\frac{c_i - c_e}{c_i - c_m}\right]^{1.16} \frac{(c_i - c_m)^2}{c_m(c_i - c_e)} \tag{17b}$$

To solve Eqs. 17, the numeric procedure of successive halving of intervals is used to evaluate c_m at the zero of the function

$$\frac{d\ \$_{s,a}}{d\ A} - f\ \phi_a(c_m) \qquad \text{or} \qquad \frac{d\ \$_{s,\ell}}{d\ L} - f\ \phi_\ell(c_m) \qquad \text{as appropriate}$$

where ϕ_a and ϕ_ℓ are respectively the right hand sides of Eqs. 17a and 17b excluding the term f in each case. But first it is necessary to determine the particular value of f corresponding to the c_m range where the zero occurs, since f depends on c_m. For this purpose pH limits for the various values of f (Eq. 15a) are converted to c_m limits by Eq. 5, examining the function for a change in sign within each range of c_m, using the corresponding f value. Having identified the interval containing c_m, one then solves for c_m. Figures 87 and 88 show d $\$_{s,a}$ / d A and d $\$_{s,\ell}$ / d L versus pH and c_e.

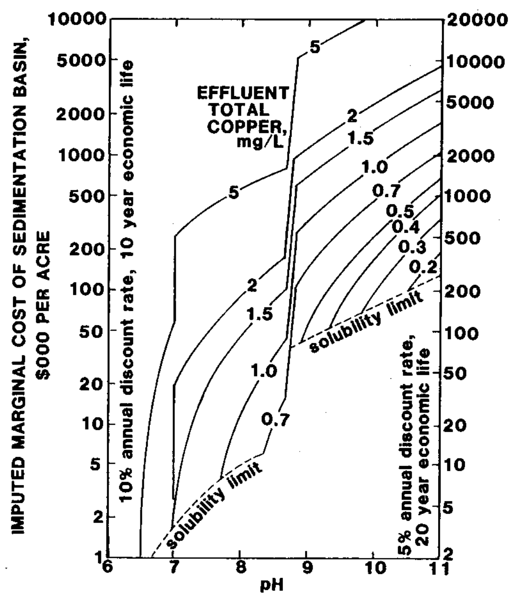


Fig. 87: Imputed Marginal Cost Of Sedimentation Basin per Acre Of Surface Area, versus Effluent Copper Concentration and pH.

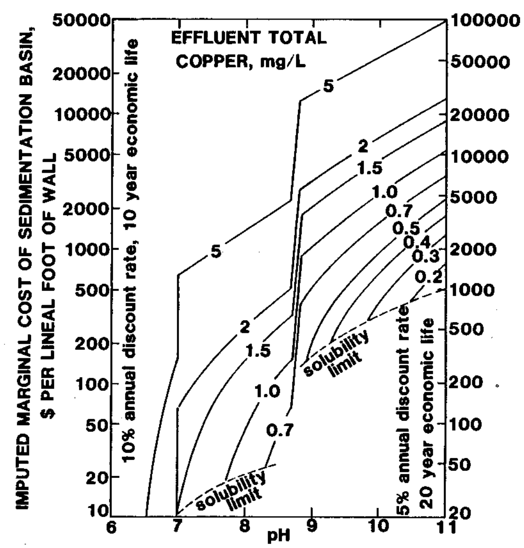


Fig. 88: Imputed Marginal Cost Of Sedimentation Basin per Foot Of Perpheral Length, versus Effluent Copper Concentration and pH.

To complete the description of optimal conditions corresponding to the selected values of c_e and d $\$_{s,a}$ / d A or d $\$_{s,\ell}$ / d L, the process pH and sedimentation basin area corresponding to the computed value of c_m are computed by Eqs. 5 and 16a respectively, and if peripheral length costing is used (by specifying d $\$_{s,\ell}$ / d L), the peripheral length is computed by Eq. 18a. All parameter values are now available for computation of construction cost, annual chemical cost and total cost.

Graphical Solution Of Optimal Design Parameter Values.- Figures 89 to 92, computed as described above, optimally relate the parameters process pH, effluent copper concentration, basin size (area for areal costing or diameter for peripheral costing) and marginal basin cost (per unit area or per unit peripheral length). Given values for any two of these parameters, the remaining two parameter values for an optimal design may be read from Figs.89 or 90 (for costing per unit area), or Figs. 91 or 92 (for costing per unit peripheral length). Odd-numbered figures apply for an annual discount rate of 10% and an economic life of 10 years, while even numbered figures are used for a discount rate of 5% per year and an economic life of 20 years.

Consider first the case that construction of the sedimentation basin is largely a matter of earthworks, it being assumed for the purpose of this example that a scour-resistant spillway can be constructed through the rock ridge to the east of the tailings area, and that the basin embankment can be constructed sufficiently broad to avoid piping on the downstream face. Suppose that an effluent copper concentration of 4.5 mg/L is selected, and that basin construction costs are estimated at $15,000 per acre of basin surface area. Example points in Figs. 89 and 90 show that for the 10%, 10 yr case the optimal pH is 6.8 and the optimal basin area 0.95 acres, while for the 5%, 20 yr case the optimal pH and basin area are respectively 6.7 and 1.05 acres, not much difference.

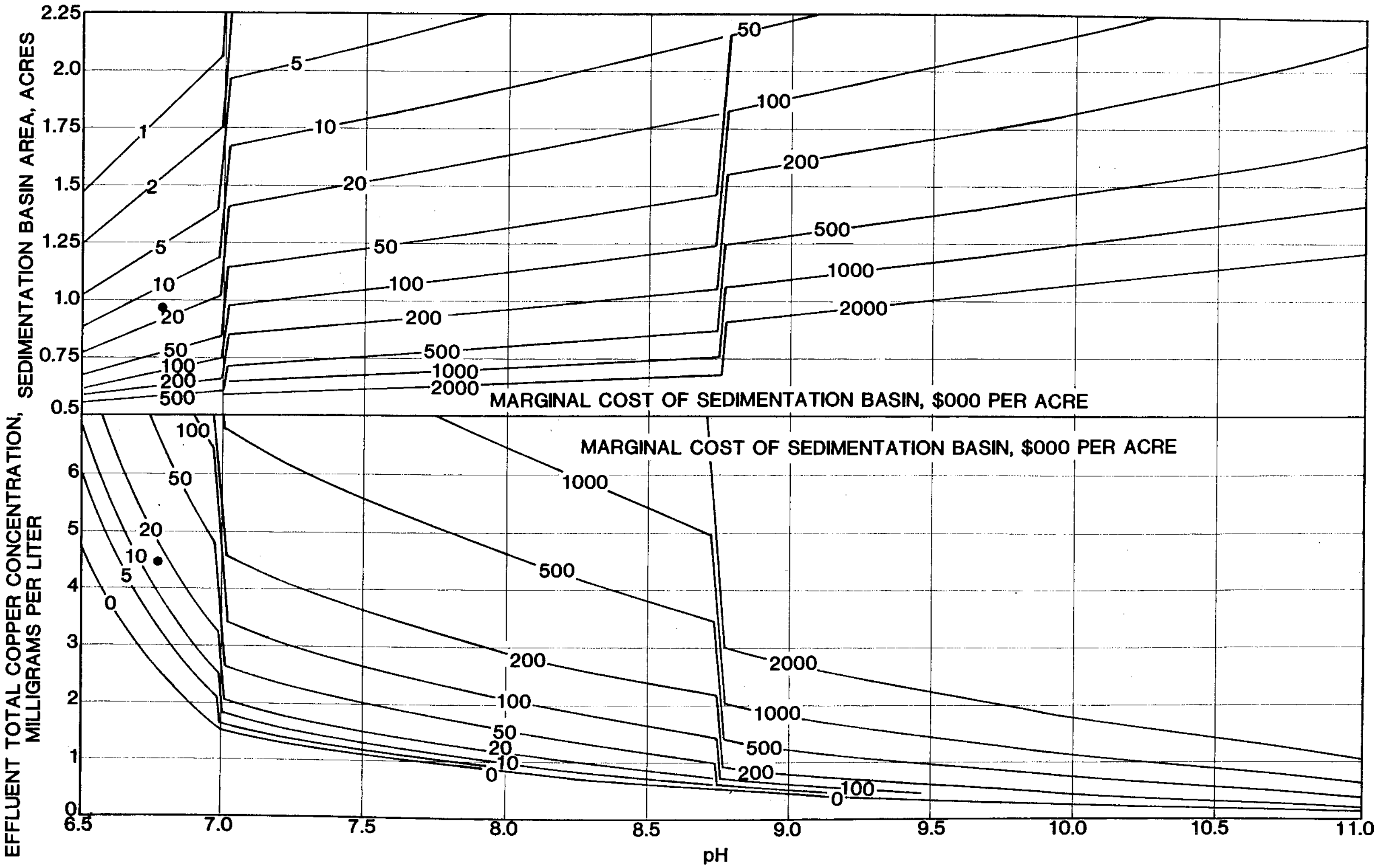


Fig. 89: Optimized Design of Walker Mine Drainage Treatment Plant, For Sedimentation Basin Costed Per Unit Area, For 10% Annual Discount Rate and 10 Year Economic Life of Facility.

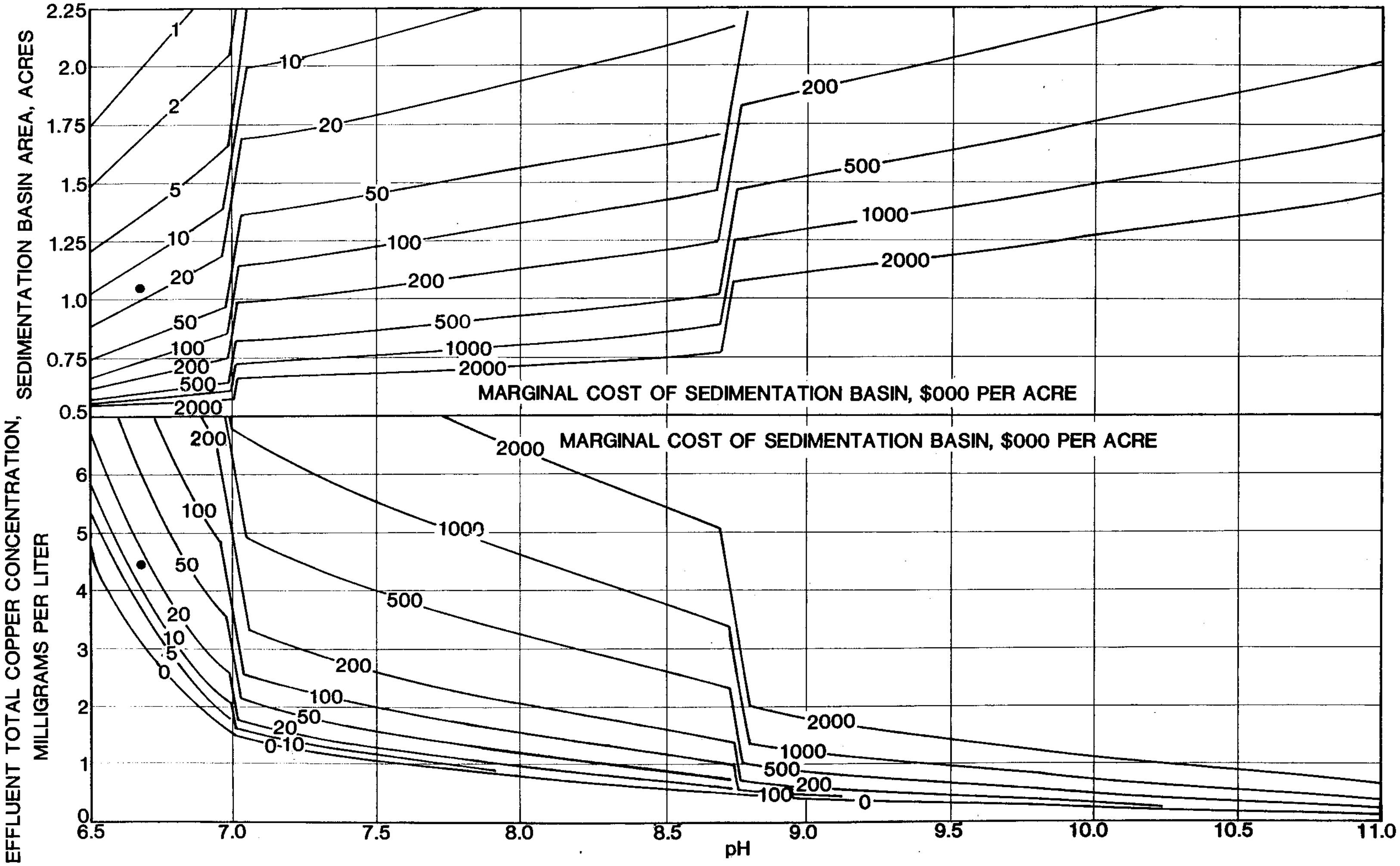


Fig. 90: Optimized Design of Walker Mine Drainage Treatment Plant, For Sedimentation Basin Costed Per Unit Area, For 5% Annual Discount Rate and 20 Year Economic Life of Facility.

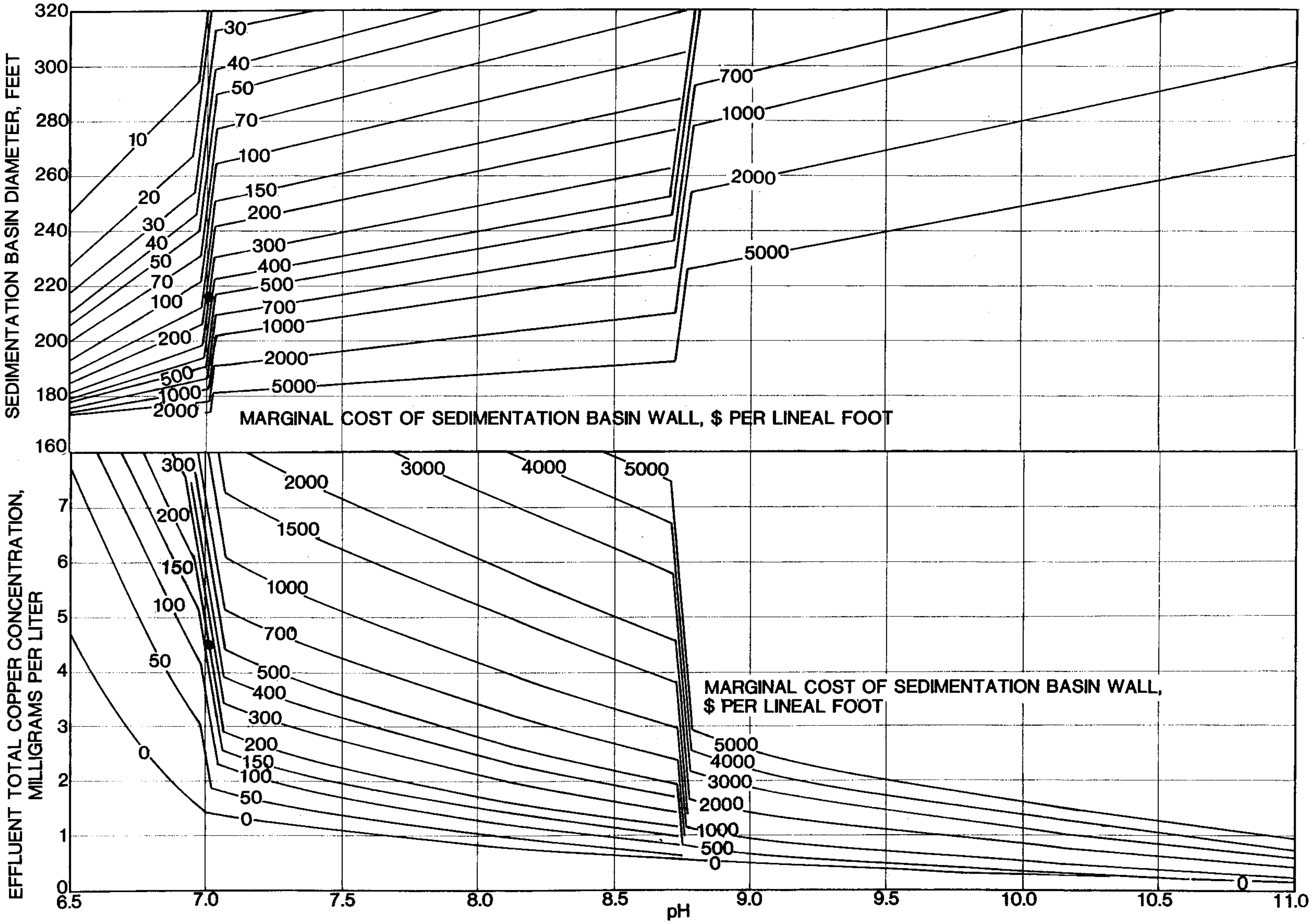


Fig. 91: Optimized Design of Walker Mine Drainage Treatment Plant, For Sedimentation Basin Costed Per Unit Length of Periphery, For 10% Annual Discount Rate and 10 Year Economic Life of Facility.

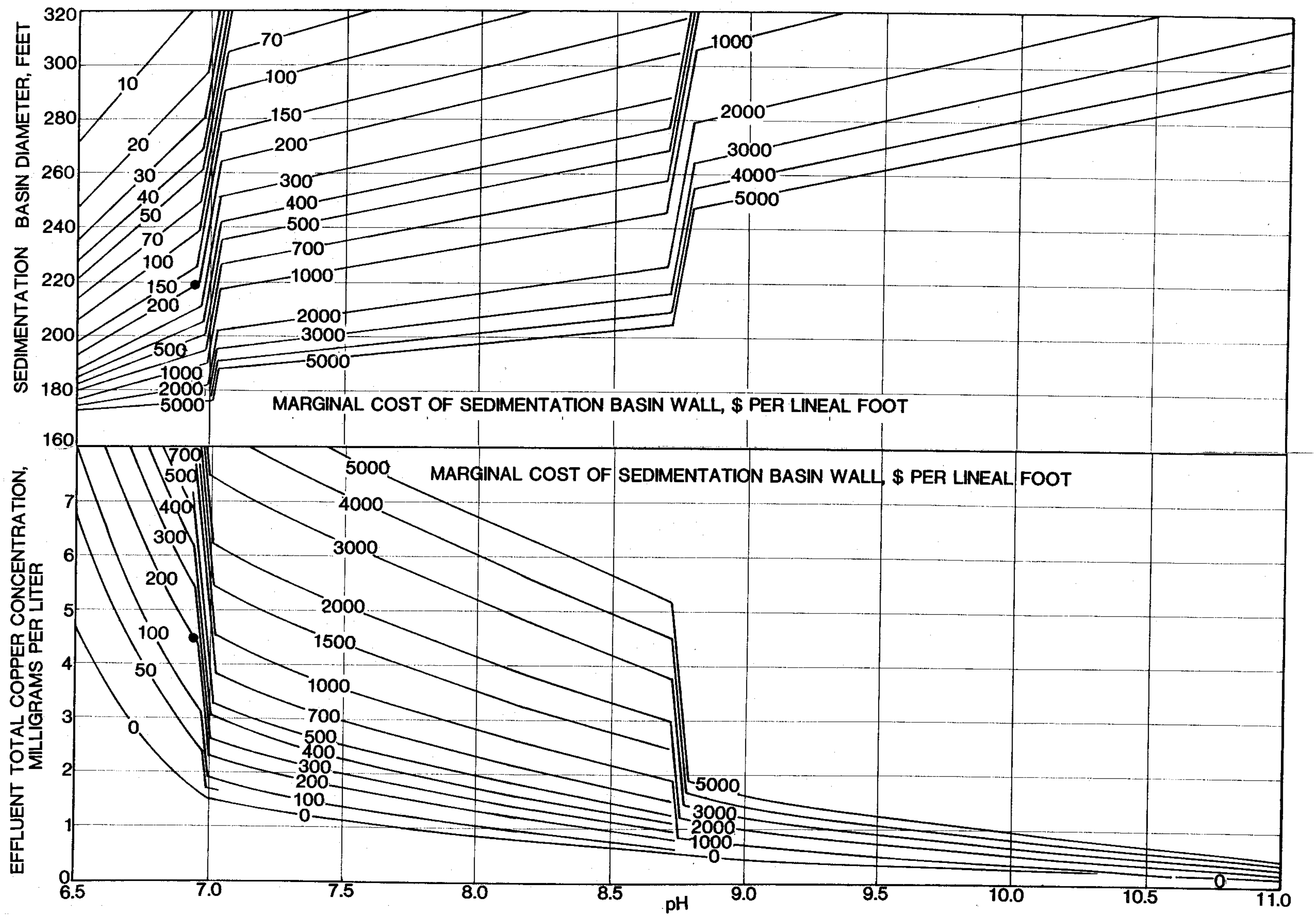


Fig. 92 : Optimized Design of Walker Mine Drainage Treatment Plant, for Sedimentation Basin Costed Per Unit Length of Periphery, For 5% Annual Discount Rate and 20 Year Economic Life of Facility.

For the case that a wall is constructed around a circular sedimentation basin at a unit cost of $200 per lineal foot, and an effluent copper concentration of 4.5 mg/L is again required, example points on Figs. 91 and 92 are used to determine the optimal process pH and optimal basin diameter for the two economic conditions. For a 10% annual discount rate and a 10 yr life these values are respectively 7.0 and 217 ft (for a basin area of 0.85 acres), while for 5%/yr and 20 yr they are pH 6.9 and 219 ft (for a basin area of 0.87 acres). For both types of costing the higher present worth factor is associated with a slightly larger sedimentation basin, and a slightly lower process pH.

<u>Graphical Solution Of Costs Associated With Optimal Designs</u>.- Figures 93 to 96 show for the less expensive optimal designs (and therefore those associated with higher levels of effluent copper and/or lower unit rates for construction) the following costs: total cost, construction cost, annual chemical cost, and annual total cost including amortization of capital. Independent variables entered into these graphs are the process pH and the unit cost of basin construction, either per unit area (in the case of Figs. 93 and 94), or per unit length of periphery (for Figs. 95 and 96). As for the process parameter graphs, even-numbered graphs are used where the discount rate is 10% per year and the economic life 10 yr, while odd-numbered graphs are used for an annual discount rate of 5% and an economic life of 20 yr.

Example points on the figures show costs corresponding to the optimal designs previously developed. From Fig. 93 for a sedimentation basin costed at $15,000 per acre, use of a discount rate of 10% per annum and a 10 year economic life for the facility results in the following costs: total cost $177,000; construction cost $150,000 (equal to funds available); annual chemical cost $5,200; and annual total cost $29,000. Total cost and total annual cost exclude any operator expenses, maintenance and repair, monitoring, or sludge disposal expenses.

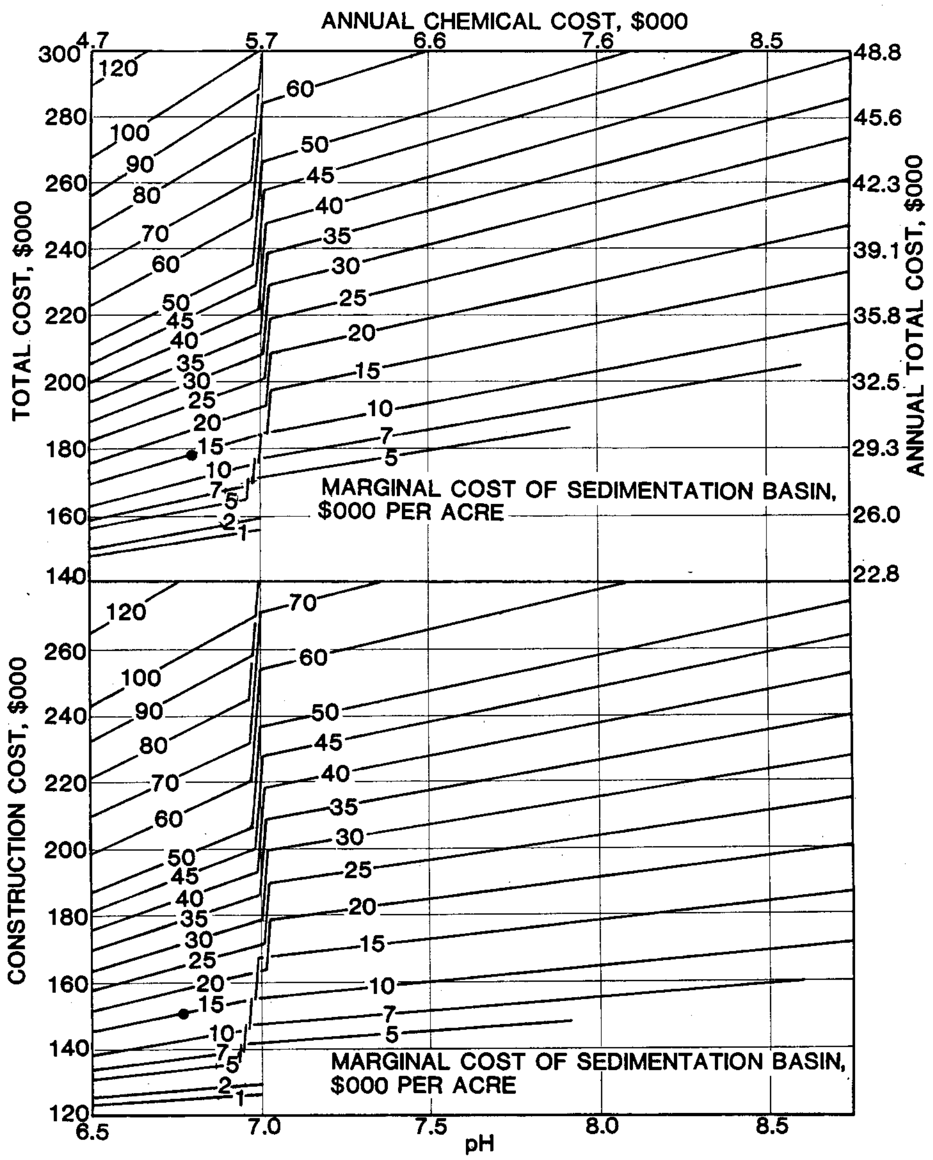


Fig. 93: Costs Of Optimized Designs Of Walker Mine Drainage Treatment Plant, For Sedimentation Basin Costed Per Unit Area, and For 10% Annual Discount Rate and 10 Year Economic Life of Facility.

Again with $15000/acre for the sedimentation basin, but with a discount rate of 5%/yr and an economic life of 20 yr, Fig. 94 shows the following costs: total cost $219,000; construction cost $183,000; annual chemical cost $3,200; and annual total cost $17,600; total cost based as before. In this case, the larger barrier as well as the slightly larger sedimentation basin both contribute to the increase in construction cost, although both also produce savings in annual chemical cost.

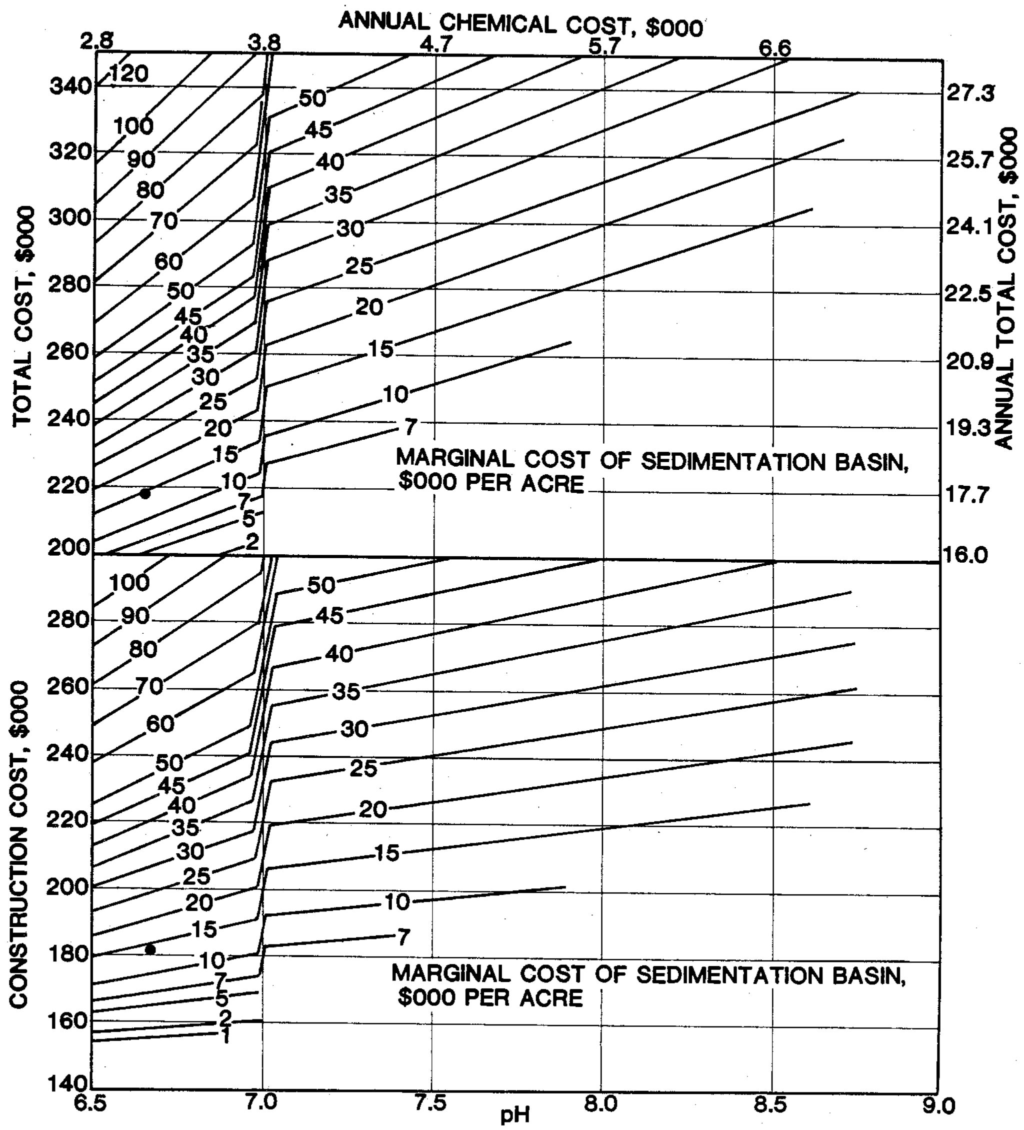


Fig. 94: Costs Of Optimized Designs Of Walker Mine Drainage Treatment Plant, For Sedimentation Basin Costed Per Unit Area, and For 5% Annual Discount Rate and 20 Year Economic Life Of Facility.

For basin costing at $200 per lineal foot of periphery, with a discount rate of 10% per year and an economic life of 10 years Fig. 95 shows: total cost $425,000; construction cost $396,000, annual chemical cost $5,700 and annual total cost $69,000, for a plant producing an effluent containing 4.5 mg/L of copper, as for the previous cases. Economies of scale present in peripheral length-based costs (but not in areal costs) penalize attempts to minimize construction cost.

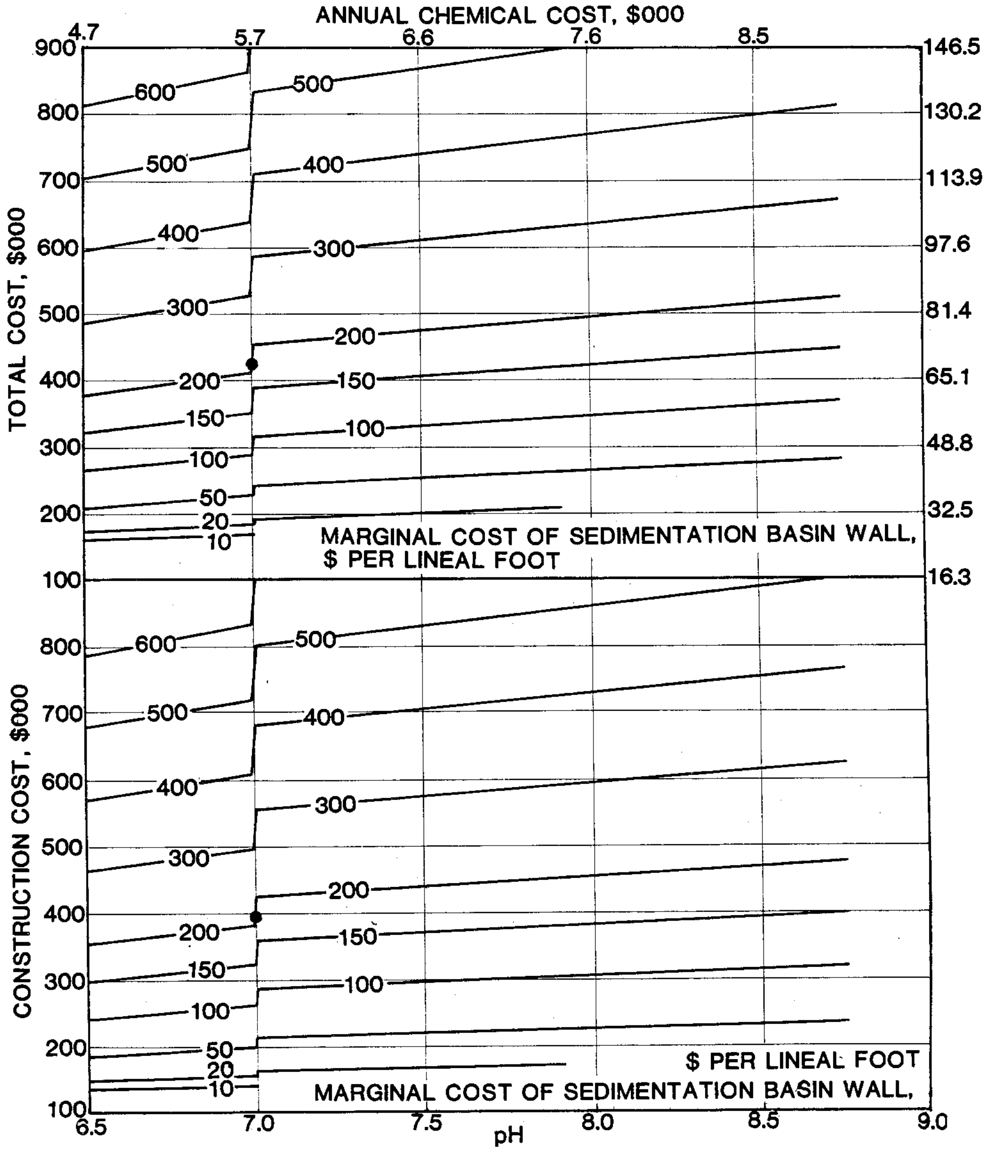


Fig. 95: Costs Of Optimized Designs Of Walker Mine Drianage Treatment Plant, For Sedimentation Basin Costed Per Unit Peripheral Length, and For 10% Annual Discount Rate and 10 Year Economic Life.

Finally, for a basin costing $200 per foot of peripheral length, with a discount rate of 5% per annum and a 20 year economic life Fig. 96 shows: total cost $470,000; construction cost $428,000; annual chemical cost $3,700; and annual total cost $37,700; total costs on the previous basis. As for unit area pricing, raising the present worth factor reduces operating costs at the expense of increased construction cost, annual total cost being reduced although total cost increases.

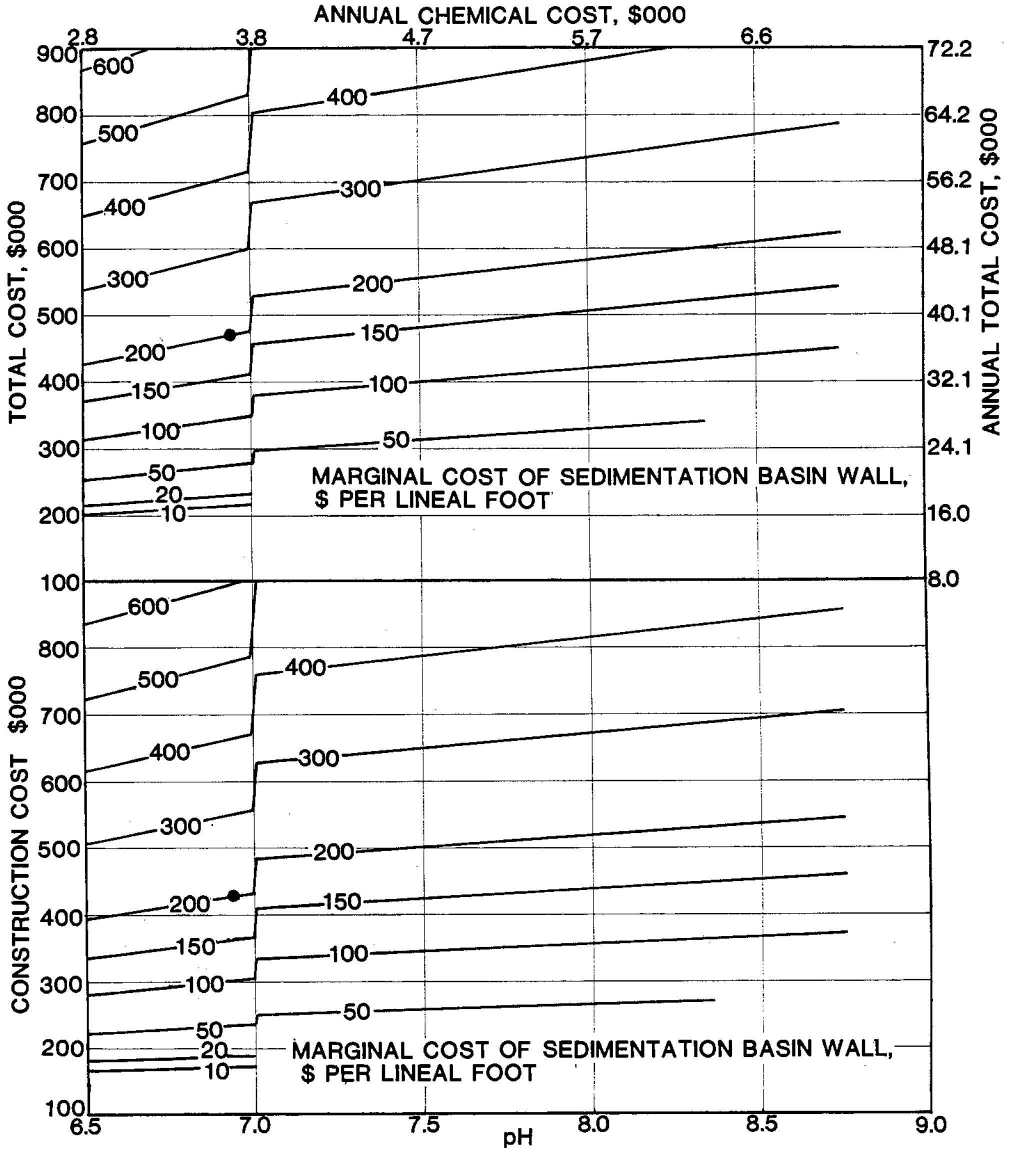


Fig. 96: Costs Of Optimized Designs Of Walker Mine Drainage Treatment Plant, For Sedimentation Basin Costed Per Unit Peripheral Length, and For 5% Annual Discount Rate and 20 Year Economic Life.

The previously mentioned economy of scale effect that arises with peripheral length costing (with an economy of scale exponent of 0.5) acts to disadvantage when capital is limited. Such is this effect that the unit cost per foot of basin periphery would have to be only $20 to reduce the construction cost to $150,000 for an optimal design based on a 10% annual discount rate and 10 yr economic life (or to an even lesser rate for the 5%/yr, 20 yr case). It appears safe to state that available finance permits only basin construction in materials (sand) on site.

Design For Minimum Construction Cost.- In view of the limited finance for construction it is reasonable to consider optimization for minimum construction cost, rather than minimum total cost as in the preceding analysis. (Such an approach could be ecomonically justifiable if a sufficiently low value of the present worth factor could be tolerated.) However, as Figs. 97 and 98 show, construction costs predominate so heavily in optimization with respect to total cost that a reduction in construction costs of no more than 5% would result from their minimization. Figure 97, prepared for areal costing at $15,000 per acre, reproduces the total and construction costs and the pH identified as optimal for an effluent copper concentration of 4.5 in the preceding analysis. Similarly, Fig. 98 reproduces the previously identified optima (with respect to total cost) for a basin cost of $200 per peripheral foot and an effluent copper concentration of 4.5 mg/L.

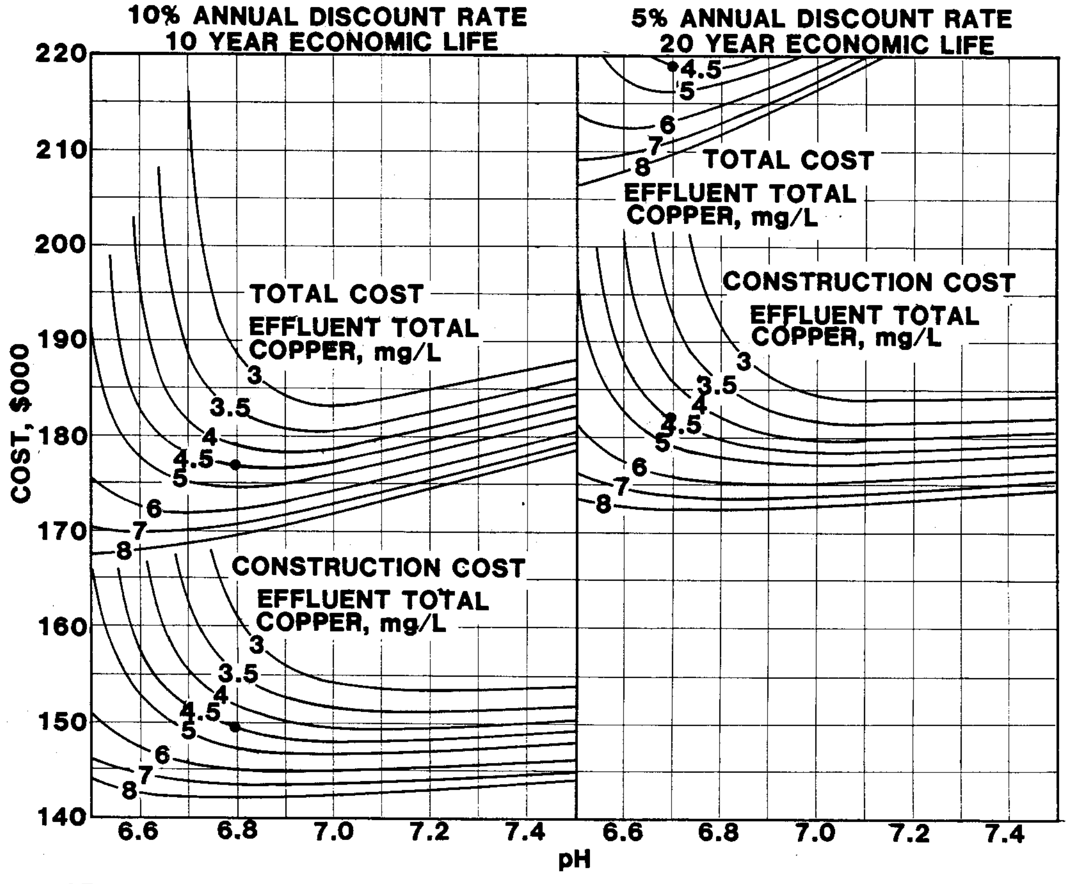


Fig. 97: Costs In Vicinity Of Total Cost Optima, For Sedimentation Basin Costed At $15,000 Per Acre Of Surface Area.

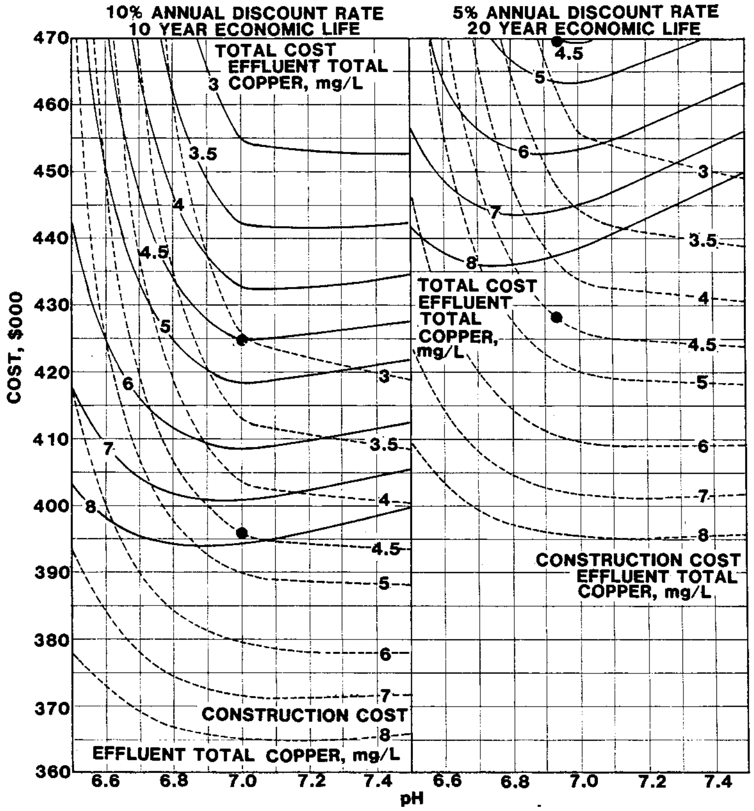


Fig. 98: Costs In Vicinity Of Total Cost Optima, For Sedimentation Basin Costed At $200 Per Lineal Foot Of Peripheral Length.

<u>Some Characteristics Of Optimal Designs</u>.- The left frame of Fig. 98 shows how, as the pH for minimum cost attainment of successively lower levels of effluent copper rises, the pH approaches pH 7, then remains at pH 7 for several decremental steps of effluent copper, then breaks clear to higher pH values. This step in process parameter values and costs at pH 7 (and also at pH 8.75) is a striking feature of Figs. 89 - 96, that results from the stepped functional representation of the parameter f. Equation 15a (pp 89-90) shows that f increases from 5 to 20 at pH 7, and from 20 to 100 at pH 8.75. The stepped nature of the parameter f results from abrupt changes observed in the chemistry of Walker mine water with increasing pH, though possibly less abrupt in fact than represented funtionally.

Recall from Eq. 5 that from pH 6 to pH 7 the rate of change of the logarithm of unprecipitated copper concentration (i.e. $\log(c_m/c_i)$)was four times its value from pH 7 to pH 11, refecting the four-fold change in slope of the trend line shown in Fig. 51 (p 51) at pH 7. Because adding chemical to increase pH for the purpose of reducing the concentration of unprecipitated copper abruptly becomes less efficient as the pH rises through 7, the optimization procedure perceives that it is more economical to increase basin area (or peripheral wall length) rather than increasing chemical dose when the pH reaches 7. Basin area (or wall length) is increased at pH 7 until the increasing marginal cost of basin area (or wall length) per unit reduction in unprecipitated copper concentration rises to match the stepwise increased marginal cost of adding chemical per unit change in unprecipitated copper concentration. Thereafter, basin area (or wall length) and chemical dose increase in unison, until the pH reaches 8.75.

At pH 8.75, because the buffer intensity of Walker mine water increases five-fold (Fig. 66, p 70 and Eq. 8, p 69), the efficiency of adding chemical for the purpose of increasing pH abruptly declines. Five times as much neutralizing agent per unit change in pH is needed above pH 8.75 as below pH 8.75. This translates to a five-fold increase in chemical dose to obtain a given fractional reduction of unprecipitated copper concentration (or a given absolute reduction in $\log(c_m/c_i)$), as pH rises through 8.75. In all, the fractional reduction of unprecipitated copper concentration per unit dose of neutralization chemical diminishes by a factor of 20, as the pH increases from below 7 to above 8.75. Once again at pH 8.75, the optimization procedure perceives that the more economical way to reduce effluent copper concentration is to increase the basin size until marginal costs of basin and chemical are again equal. Subsequently, basin size and chemical dose are both increased to further enhance effluent quality.

Relationship Between Areal Costing and Peripheral Costing.- For a circular basin it is possible, though inconvenient, to translate between optimal parameter values and total and construction costs for the two types of costing, for a particular value of the present worth factor. Consider a circular basin of area A acres. If the area-based cost is $\$_{s,a}$ per acre, then the total and construction costs are the same as for a peripheral length-costed basin of the same area with a unit cost of $\$_{s,\ell} = \$_{s,a}\ [\ A\ /\ (4\pi \times 43560)\ ]^{0.5}$ per lineal foot of periphery. Thus, the cost per peripheral foot of a circular basin with a construction cost of \$150,000 based on a 10% annual discount rate and a 10 year economic life can be obtained from the area in acres, and cost per acre of a basin with the same construction cost. Referring to p 95, for the 10%/yr, 10 yr example case of a plant with a construction cost of \$150,000 to produce an effluent containing 4.5 mg/L of copper, the area of the basin is 0.95 acres, and the marginal cost of construction \$15,000 per acre. Then, for a peripheral length-costed basin of the same area, and producing the same quality of effluent, the unit cost is $\$15{,}000[0.95/(4\pi \times 43560)]^{0.5} = \$20/\text{ft}$, adequate for wave protection of an earthworks basin, but not for total cost.

Cost Estimates.- Procedures described above do not provide estimates of total cost or construction cost (except for chemicals and limestone). They are simply a tool for testing "what if" hypotheses, such as: "What are the cost implications of a basin construction cost of \$X/acre or \$Y/ft?" or "Is an effluent copper concentration of Z mg/L and a construction cost ceiling of \$W compatible with reasonable unit rates for basin construction per acre or per foot?" As such, the procedures serve a design screening function, indicating that for a neutralization-sedimentation plant, neither a walled basin nor an earthworks basin is financially feasible, for effluent with below 5mg/L of copper within a budget for construction of \$150,000, for basin construction costs considerably in excess of \$20 per peripheral foot.

Design For Minimum Residual Copper.- The preceding discussion concerns optimal design of plants to provide a relatively modest removal of copper from raw mine drainage, 70% removal, corresponding to 4.5 mg/L of total copper in the treated effluent. Lower residuals are technically feasible but financially infeasible with $150,000 available for construct-ion. But because an effluent containing 4.5 mg/L of copper would continue to violate the Basin Plan limit for copper of 0.01 mg/L (albeit to a lesser extent than before treatment), it is pertinent to discuss characteristics of treatment to produce effluent of higher quality, more compatible with the Basin Plan limit.

Figures 87 and 88 indicate the technical feasibility of an effluent copper concentration as low as 0.2 mg/L, although process conditions for attainment of this level are quite restrictive. A pH range of 10.5 to 11 is shown as suitable for producing an effluent containing 0.2 mg/L of total copper, and marginal costs for the basin are also shown, depending on whether basin costing is according to surface area or peripheral length, and on values employed for the discount rate and economic life. Specification of marginal basin cost as a function of effluent copper concentration and pH is a consequence of equating the marginal costs of neutralization and sedimentation during optimization. If the actual marginal cost of basin construction differs from that corresponding to a selected effluent copper-pH value pair according to Fig. 87 or 88, then total cost could be reduced by adjusting the pH and basin size in opposite directions in order to satisfy the effluent copper-pH-marginal basin cost relationship in Fig. 87 or 88. For example, if the actual marginal cost of basin construction is below that shown in the figures, then the figures suggest that overall economy would result from spending more on basin construction (at the relatively low rate) and less on chemical (to produce a lower pH than originally contemplated).

For effluent with a copper concentration of 0.2 mg/L, Fig. 87 shows marginal costs per acre associated with optimal designs of approx $150,000 to $300,000 per acre, depending on whether the discount rate and economic life are 10%/yr and 10 yr or 5%/yr and 20 yr respectively. Corresponding marginal costs of optimal designs for a walled sedimentation basin are from Fig. 88 approx $500 and $1,000 per lineal foot. Although these unit costs are not unreasonable for certain types of construction, detail design with a strict eye for construction economy may produce rather lower unit costs.

To identify the optimum pH of treatment, one locates the intercept of the 0.2 mg/L effluent copper line and the horizontal line representing marginal cost from whichever ordinate curve applies in a particular situation, then reads down to the pH value on the abscissa. If the effluent copper line and the marginal cost line do not intersect, then either the would-be optimum pH exceeds 11 (beyond the range of process validation) or the pH corresponds to a concentration of unprecipitated copper (by Eq. 5) above the required effluent copper concentration, (e.g. pH 10.5 is minimal by Eq. 5 for $c_e \geq c_m$ = 0.2 mg/L). In such cases, total cost is minimized by neutralizing to the pH for which $c_e = c_m$ (e.g. to pH 10.5 for c_e = 0.2 mg/L), and constructing a basin of size computed by substituting $c_e = c_m$ in Eq. 16a (2.2 acres or 350 ft diam).

Then, if the site will not accomodate a basin of this size, it may be possible to recompute by Eq. 16a for the feasible basin size the required value of $c_m < c_e$, thence the required pH by Eq. 5. Figures 89-92 show these calculations graphically, although for high quality effluent the parameter values are beyond the ranges of the graphs.

Both construction and operating costs for a plant to produce an effluent containing only 0.2 mg/L of total copper are substantial. If the tailings area on which the basin would be constructed is sufficient only for a walled basin (and not for the embankments of an earthworks basin) the minimum construction cost may range from \$600,000 to \$1,300,000 for steel sheet piling basin walls with a unit cost in the range \$200 to \$500 per lineal foot of periphery. The annual cost of chemicals would approximate \$15,000. But because a basin of this 2.2 acre size would cover most of the tailings area, metals loadings to receiving waters due to seepage from the tailings area would be substantially reduced, particularly if part of the high pH effluent were disposed of subsurface to intercept and neutralize seepage from that part of the tailings area not covered by the basin. Metals precipitated underground would gradually clog subsurface pores until tailings seepage appeared at the ground surface, at which stage the seepage could be diverted into a second sedimentation basin (with a lower water level than the first) for treatment as for the mine discharge.

EFFECT OF WALKER MINE ON WATER QUALITY IN DOLLY CREEK

Water Quality and Streamflow Data.- Incidentally to operation of the pilot plant, on most sampling occasions water samples were taken for analysis from Dolly Creek upstream and downstream of the Walker mine area, at points identified in Fig. 5. During the sampling program a stream gauging flume near the Dolly Ck downstream sampling point (thought to have been washed out) was discovered, partially restored, and thereafter used to obtain measurements for estimating Dolly Ck streamflow. The calculated rating equation for this 45.3 in. wide rectangular weir was: flow, in gpm = 1.06 (head, in mm)$^{1.5}$, according to the millimeter calibration of the gauging rod. It would be possible to retroactively improve the accuracy of flow estimates by calibrating the gauge.

Figures 99-103 present data on pH, as well as total copper, zinc, manganese and iron, for the mine discharge and for the two Dolly Ck sampling points.

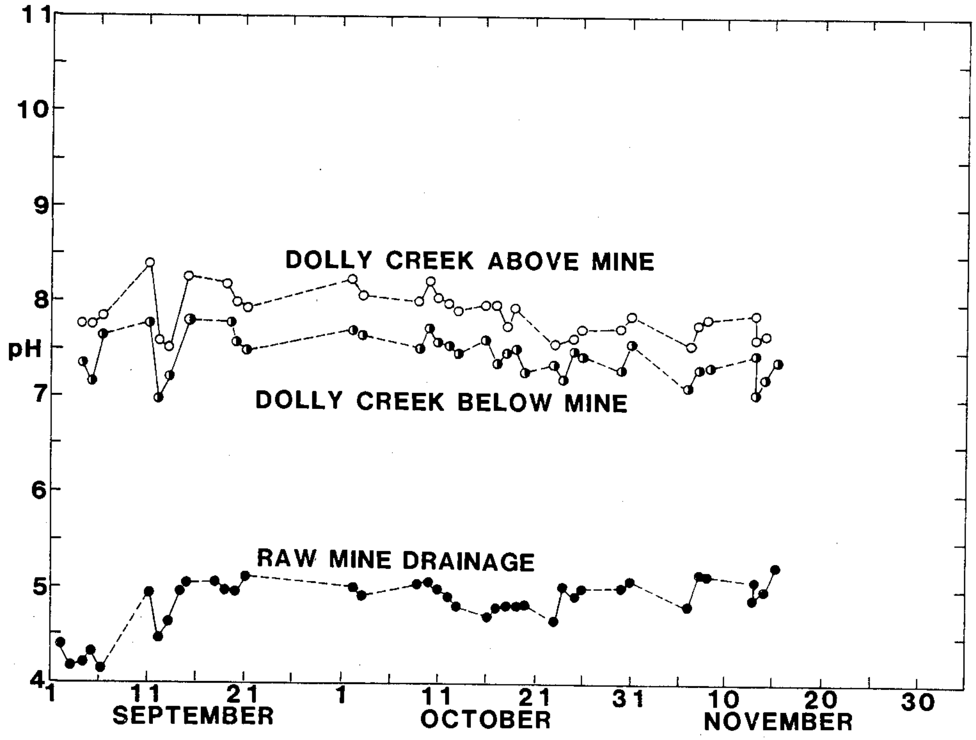


Fig. 99: pH In Dolly Creek Upstream and Downstream Of Walker Mine, and In Walker Raw Mine Drainage.

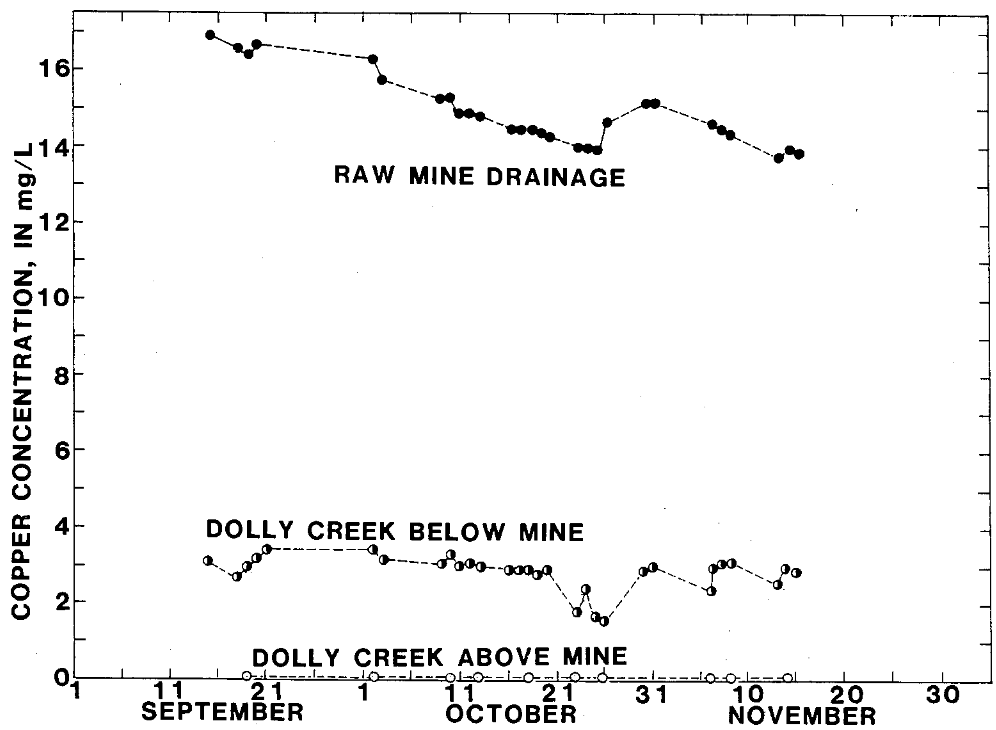


Fig. 100: Total Copper In Dolly Creek Upstream and Downstream Of Walker Mine, and In Walker Raw Mine Drainage.

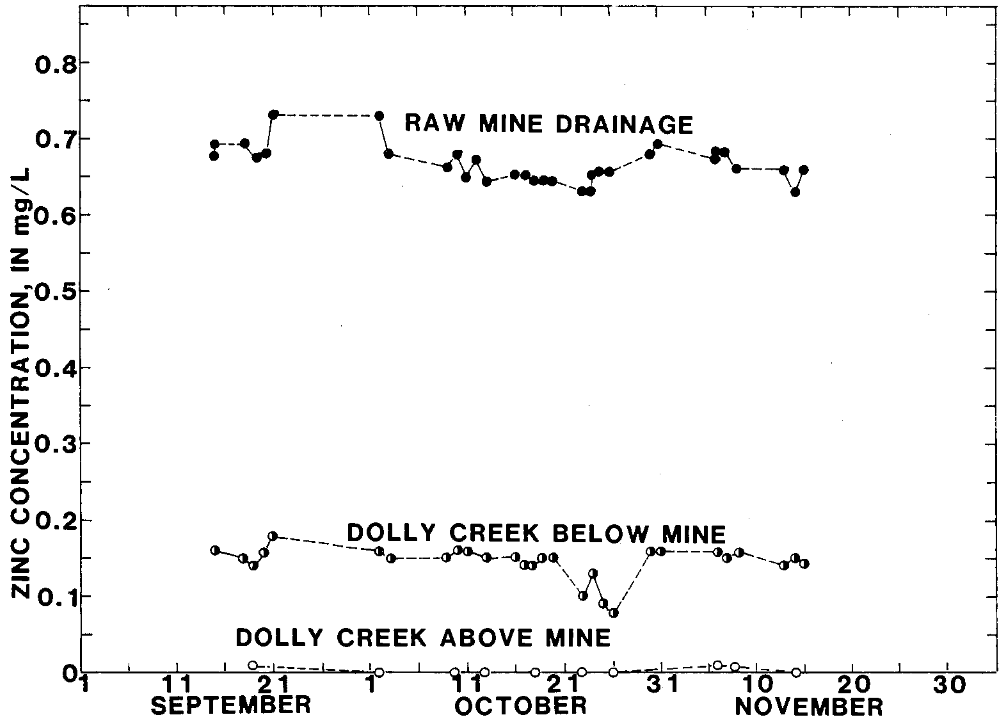


Fig. 101: Total Zinc In Dolly Creek Upstream and Downstream Of Walker Mine, and In Walker Raw Mine Drainage.

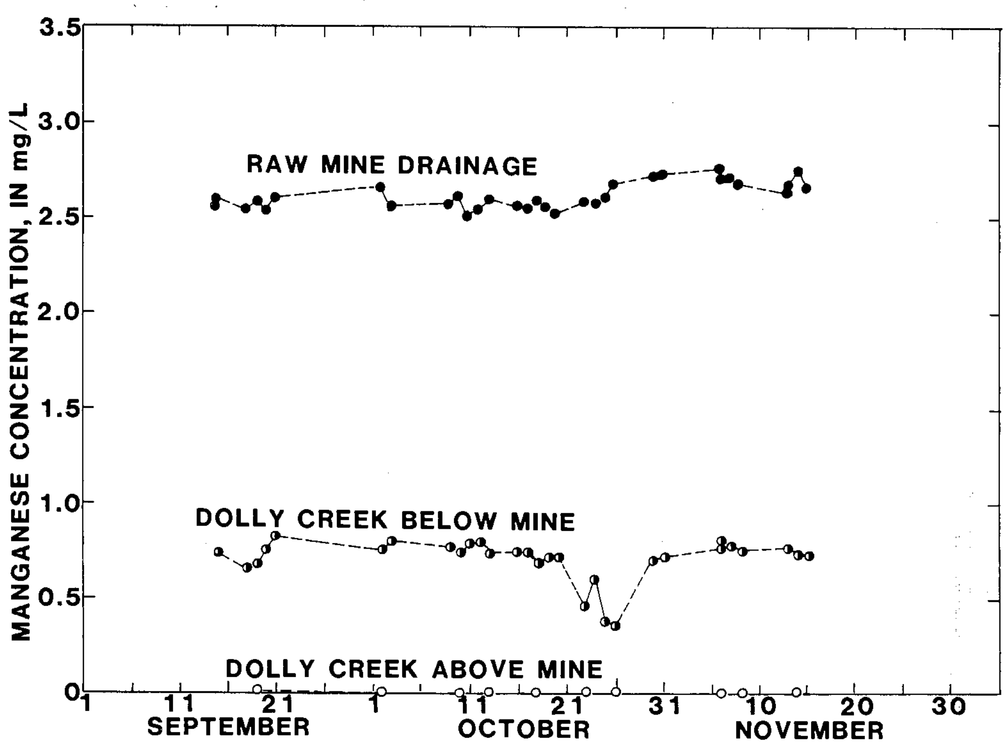


Fig. 102: Total Manganese In Dolly Creek Upstream and Downstream Of Walker Mine, and In Walker Raw Mine Drainage.

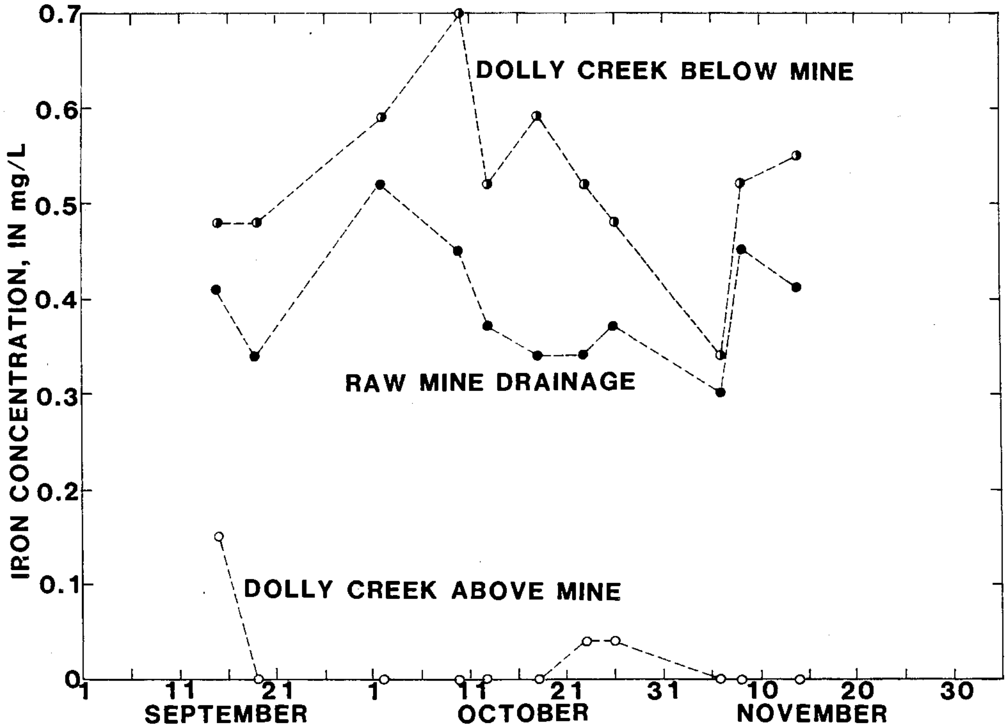


Fig. 103: Total Iron In Dolly Creek Upstream and Downstream Of Walker Mine, and In Walker Raw Mine Drainage.

Table 18 summarizes the changes in pH and in concentrations of total metals from upstream of Walker mine to downstream of Walker mine, notably characterized by a mean fall in pH of 0.4 units, and a mean increase in copper concentration of 2.7 mg/L over the monitoring period.

TABLE 18: Summarized Water Quality Analyses For Dolly Creek Upstream and Downstream Of Walker Mine.

Parameter	Upstream					Downstream					Increase[a]	
	Mean	SD	Min	Max	No. rdgs	Mean	SD	Min	Max	No. rdgs	Mean	SD
pH	7.9	0.2	7.5	8.4	33	7.4	0.2	7.0	7.8	36	-0.4	0.1
Total copper, mg/L	<0.1	-	<0.1	<0.1	10	2.8	0.5	1.6	3.4	31	2.7	~0.5
zinc, mg/L	<.01	-	.00	.01	10	.15	.02	.08	.18	30	.14	~.03
manganese, mg/L	.01	.01	.00	.02	10	.71	.11	.16	.36	31	.66	.14
iron, mg/L	.02	.05	.00	.15	11	.52	.09	.34	.70	11	.50	.11
Free copper, mg/L	.01	-	.01	.01	37	1.7	0.7	0.3	3	36	1.7	0.7

[a]for those occasions when paired samples were analyzed.

Although the analytical method employed for determination of total copper was insufficiently sensitive to detect whether water from Dolly

TABLE 19: Walker Mine Discharge and Dolly Creek Water Quality

Sampling identification			Temperature, °C		Acid mine drainage total flow, gpm[a]	Raw mine drainage					Dolly Creek above mine					Dolly Creek below mine						"Free" copper by colorimetric analysis, mg/L		
Serial number	Date (1982)	Time	Acid mine drainage	Air		pH	Copper mg/L	Zinc mg/L	Manganese mg/L	Iron mg/L	pH	Copper mg/L	Zinc mg/L	Manganese mg/L	Iron mg/L	pH	Copper mg/L	Zinc mg/L	Manganese mg/L	Iron mg/L	Flow gpm	Raw mine drainage	Dolly Creek above mine	Dolly Creek below mine
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)	(13)	(14)	(15)	(16)	(17)	(18)	(19)	(20)	(21)	(22)	(23)	(24)	(25)
1	9/ 2	1400	-	-	-	4.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12	-	-
2	9/ 3	1430	-	-	-	4.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12	-	-
3	9/ 4	1430	-	-	-	4.2	-	-	-	-	7.8	-	-	-	-	7.4	-	-	-	-	-	12	0.01	2.5
4	9/ 5	1430	-	-	120	4.3	-	-	-	-	7.8	-	-	-	-	7.2	-	-	-	-	-	12	0.01	2.5
5	9/ 6	1200	-	-	100	4.2	-	-	-	-	7.8	-	-	-	-	7.7	-	-	-	-	-	12	0.01	2.5
6	9/11	1630	-	-	120	4.9	-	-	-	-	8.4	-	-	-	-	7.7	-	-	-	-	-	13	0.01	3
7	9/12	1600	-	-	110	4.5	-	-	-	-	7.6	-	-	-	-	7.0	-	-	-	-	-	13	0.01	2.5
8	9/13	1730	-	-	140	4.7	-	-	-	-	7.5	-	-	-	-	7.2	-	-	-	-	-	13	0.01	3
9	9/14	1100	6	14	140	5.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	13	0.01	2
10	9/15	1000	6	3	110	5.1	16.8	0.67	2.58	0.41	8.3	-	-	-	0.15	7.8	-	-	-	0.48	-	10	0.01	2.5
11	9/15	1300	-	-	120	5.1	17.0	0.69	2.60	-	-	-	-	-	-	-	3.1	0.16	0.74	-	-	12	0.01	-
12	9/18	1800	6	6	120	5.1	16.6	0.69	2.55	-	-	-	-	-	-	-	2.7	0.15	0.66	-	-	12	0.01	0.3
13	9/19	1300	6	9	100	5.0	16.4	0.67	2.59	0.34	8.2	<0.1	0.01	0.02	0.00	7.8	3.0	0.14	0.68	0.48	-	10	0.005	0.8
14	9/20	0700	6	15	110	5.0	16.7	0.68	2.55	-	8.0	-	-	-	-	7.6	3.2	0.16	0.76	-	-	10	0.01	1.2
15	9/21	0700	6	-	115	5.1	-	0.73	2.61	-	8.0	-	-	-	-	7.5	3.4	0.18	0.82	-	-	10	0.01	1.5
16	10/ 2	1530	6	11	75	5.0	16.3	0.73	2.67	0.52	8.3	<0.1	0.00	0.00	0.00	7.7	3.4	0.16	0.76	0.59	-	8	0.01	1.8
17	10/ 3	1100	6	14	85	4.9	15.8	0.68	2.56	-	8.1	-	-	-	-	7.7	3.2	0.15	0.80	-	-	9	0.01	1.5
18	10/ 9	1530	6	12	65	5.1	15.3	0.66	2.58	-	8.0	-	-	-	-	7.5	3.1	0.15	0.77	-	-	13	0.01	1.3
19	10/10	1430	6	14	70	5.1	15.3	0.68	2.62	0.45	8.2	<0.1	0.00	0.00	0.00	7.7	3.3	0.16	0.75	0.70	-	10	0.01	1.5
20	10/11	1500	6	15	75	5.0	14.9	0.65	2.51	-	8.0	-	-	-	-	7.6	3.0	0.16	0.79	-	-	8	0.01	2.0
21	10/12	1630	6	15	75	4.9	14.9	0.67	2.55	-	8.0	-	-	-	-	7.6	3.1	-	0.79	-	-	9	0.005	1.5
22	10/13	1530	6	14	80	4.8	14.8	0.64	2.60	0.37	7.9	<0.1	0.00	0.01	0.00	7.5	3.0	0.15	0.74	0.52	-	10	0.01	1.5
23	10/16	1630	6	15	70	4.7	14.5	0.65	2.57	-	8.0	-	-	-	-	7.6	2.9	0.15	0.75	-	-	8	0.01	1.3
24	10/17	1230	6	15	80	4.8	14.5	0.65	2.56	-	8.0	-	-	-	-	7.4	2.9	0.14	0.73	-	-	8	0.01	1.2
25	10/18	1530	6	13	60	4.8	14.5	0.64	2.59	0.34	7.8	<0.1	0.00	0.00	0.00	7.5	2.9	0.14	0.69	0.59	-	8	0.01	1.2
26	10/19	1400	6	10	65	4.8	14.3	0.64	2.56	-	8.0	-	-	-	-	7.5	2.8	0.15	0.71	-	-	9	0.01	1.3
27	10/20	1000	6	7	60	4.9	14.3	0.64	2.53	-	-	-	-	-	-	7.3	2.9	0.15	0.71	-	-	8	0.01	1.5
28	10/23	1630	6	6	75	4.7	14.1	0.63	2.59	0.34	7.6	<0.1	0.00	0.01	0.04	7.4	1.8	0.10	0.46	0.52	-	10	0.01	1.0
29	10/24	1000	6	9	65	5.0	14.0	0.65	2.59	-	-	-	-	-	-	7.2	2.4	0.13	0.60	-	-	10	0.01	1.2
30	10/25	0830	6	4	60	4.9	14.0	0.66	2.61	-	7.6	-	-	-	-	7.5	1.7	0.09	0.39	-	2,200	14	0.01	1.2
31	10/26	1000	-	-	60	5.0	14.7	0.66	2.68	0.37	7.7	<0.1	0.00	0.01	0.04	7.5	1.6	0.08	0.36	0.48	2,300	11	0.01	0.8
32	10/30	1330	6	3	70	5.0	15.2	0.68	2.72	-	7.8	-	-	-	-	7.3	2.9	0.16	0.70	-	770	10	0.01	1.2
33	10/31	0930	6	0	70	5.1	15.2	0.69	2.74	-	7.9	-	-	-	-	7.6	3.0	0.16	0.72	-	690	10	0.01	1.5
34	11/ 6	0200	5	3	75	4.8	14.8	0.67	2.76	0.30	7.6	<0.1	0.01	0.00	0.00	7.1	2.4	0.16	0.77	0.34	590	9	0.01	2.5
35	11/ 6	1545	6	8	75	4.8	14.6	0.68	2.71	-	7.6	-	-	-	-	7.1	3.0	0.16	0.80	-	580	9	0.01	2.5
36	11/ 7	0700	-	-2	80	5.2	14.6	0.68	2.71	-	7.8	-	-	-	-	7.3	3.1	0.15	0.78	-	590	9	0.01	2.5
37	11/ 8	0730	-	-4	75	5.2	14.4	0.66	2.69	0.45	7.8	<0.1	0.01	0.01	0.00	7.3	3.1	0.16	0.76	0.52	590	11	0.01	1.8
38	11/13	0130	5	-10	65	5.1	13.8	0.66	2.68	-	7.9	-	-	-	-	7.4	2.7	0.14	0.76	-	520	9	-	-
39	11/13	1500	6	2	70	4.9	13.8	0.66	2.66	-	7.6	-	-	-	-	7.1	2.7	0.14	0.74	-	520	10	0.01	1.2
40	11/14	0800	6	-2	75	5.0	14.0	0.63	2.75	0.41	7.7	<0.1	0.00	0.01	0.00	7.2	3.0	0.15	0.74	0.55	500	10	0.01	1.3
41	11/15	0715	6	-2	75	5.2	13.9	0.66	2.67	-	-	-	-	-	-	7.4	2.9	0.14	0.74	-	510	9	-	-
42	11/20	1300	6	-2	-	5.1	-	-	-	-	-	-	-	-	-	7.7	-	-	-	-	-	-	-	-
43	12/ 7	1500	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

[a]treated flow plus bypassed flow metered by a Palmer-Bowlus flume.

TABLE 20: Mass Loadings and Recoveries of Copper, Zinc and Manganese.

Sampling serial number	Copper				Zinc				Manganese			
	Load, in pounds per day			Recovery percent[a]	Load, in pounds per day			Recovery percent[a]	Load, in pounds per day			Recovery percent[a]
	From mine	Dolly Ck above mine	Dolly Ck below mine		From mine	Dolly Ck above mine	Dolly Ck below mine		From mine	Dolly Ck above mine	Dolly Ck below mine	
11	24.4	-	-	-	0.99	-	-	-	3.7	-	-	-
12	23.9	-	-	-	0.99	-	-	-	3.7	-	-	-
13	19.7	-	-	-	0.80	-	-	-	3.1	-	-	-
14	22.0	-	-	-	0.90	-	-	-	3.4	-	-	-
15	-	-	-	-	1.00	-	-	-	3.6	-	-	-
16	14.7	-	-	-	0.66	-	-	-	2.4	-	-	-
17	16.1	-	-	-	0.69	-	-	-	2.6	-	-	-
18	11.9	-	-	-	0.51	-	-	-	2.0	-	-	-
19	12.9	-	-	-	0.57	-	-	-	2.2	-	-	-
20	13.4	-	-	-	0.59	-	-	-	2.3	-	-	-
21	13.4	-	-	-	0.60	-	-	-	2.3	-	-	-
22	14.2	-	-	-	0.61	-	-	-	2.5	-	-	-
23	12.2	-	-	-	0.55	-	-	-	2.2	-	-	-
24	13.9	-	-	-	0.62	-	-	-	2.5	-	-	-
25	10.4	-	-	-	0.46	-	-	-	1.9	-	-	-
26	11.2	-	-	-	0.50	-	-	-	2.0	-	-	-
27	10.3	-	-	-	0.46	-	-	-	1.8	-	-	-
28	12.7	-	-	-	0.57	-	-	-	2.3	-	-	-
29	10.9	-	-	-	0.51	-	-	-	2.0	-	-	-
30	10.1	0.77	44.9	414	0.48	0.08	2.4	430	1.9	0.15	10.3	506
31	10.6	0.81	44.2	388	0.48	0.08	2.2	397	1.9	0.16	9.9	475
32	12.8	0.25	26.8	206	0.57	0.03	1.5	248	2.3	0.05	6.5	277
33	12.8	0.22	24.8	191	0.58	0.02	1.3	220	2.3	0.04	6.0	254
34	13.3	0.19	17.0	126	0.60	0.02	1.1	182	2.5	0.04	5.5	216
35	13.1	0.18	20.9	157	0.61	0.02	1.1	177	2.4	0.04	5.6	225
36	14.0	0.18	21.9	155	0.65	0.02	1.1	158	2.6	0.04	5.5	209
37	13.0	0.19	21.9	167	0.59	0.02	1.1	185	2.4	0.04	5.4	219
38	10.8	0.16	16.8	154	0.51	0.02	0.87	164	2.1	0.03	4.7	223
39	11.6	0.16	16.8	143	0.55	0.02	0.87	153	2.2	0.03	4.6	204
40	12.6	0.15	18.0	141	0.57	0.02	0.90	155	2.5	0.03	4.4	177
41	12.5	0.16	17.7	140	0.59	0.02	0.86	141	2.4	0.03	4.5	186

[a]Daily load of substance in Dolly Creek below mine, as a percentage of daily load in Dolly Creek above mine plus daily load from mine.

Creek upstream of Walker mine contained less copper than the 0.01 mg/L Basin Plan limit, it was established with high statistical significance that the copper content of the water increases flowing past the mine, and that the copper content of Dolly Creek water downstream of the mine does exceed the Basin Plan limit of 0.01 mg/L. Downstream of the mine the concentration of iron in Dolly Creek water exceeded that even in Walker raw mine drainage, presumably on account of iron contributed to the water from scrap iron and steel abundant in the tailings area.

Mass Balancing Of Mine Discharge Pollutant Loads Versus Stream Loads.- Table 19 lists detailed water analyses, flow measurements and other data, from which metals loadings (in pounds per day) at the various points of sampling were computed, as shown in Table 20. Also listed in Table 20 for the portion of the monitoring period that Dolly Creek streamflow was measured are percentage recoveries of each metal, or the daily load of metal in Dolly Creek downstream of the mine as a percentage of the sum of the daily loads in the mine discharge and in Dolly Creek upstream of the mine.

In every case, more of each metal appeared in Dolly Creek downstream of the mine than was contributed from both the mine discharge and Dolly Creek above the mine, apparently due to metals contributed to Dolly Creek in diffuse seepage from the tailings, that was not monitored. The unaccounted for discrepancy between metals loads in Dolly Creek below the mine and the other two sources is referred to as the imputed tailings seepage load.

All samplings from which the imputed tailings seepage load was estimated were in the wet season, from late October, 1982. This load was highest for samplings 30 and 31, presumably on account of rather intense rain on the first of these days, October 25, with snow on the following day. Thereafter, for samplings 32-41, imputed tailings seepage loads became relatively stable. For copper for example, the mean loads for samplings 32-41 from the mine discharge and from Dolly Creek upstream and downstream of the mine were respectively 12.7, 0.2 and 20.3 lb/day, leaving 7.4 lb/day (57% of the mine discharge load) as the imputed tailings load. Presumably, a similar load of copper to this imputed tailings seepage load enters Dolly Creek from sources other than the mine discharge when the ground is saturated during the spring thaw period of maximum fish kill risk. (Incidentally, copper loads in Dolly Creek above the mine given above and in Table 20 are upper limit estimates.)

Field data are needed to assess the loads of metals in seepage from the tailings area, e.g. by analyzing groundwater samples from the tailings area. For maximum water quality benefit from the pollution control dollar, the marginal cost of removing copper from the mine discharge per unit concentration of copper in Dolly Creek should equal the marginal cost of abatement of the tailings source, per mg/L copper in the stream. As previously remarked, the tailings source might be substantially reduced by spreading high pH treated effluent on the tailings area. This might tend to shift the optimum pH of neutralization up from the value identified considering only optimization of the process itself.

Of possible concern is the impact of high pH treated effluent on the pH of Dolly Creek, for which the Basin Plan specifies a pH range of 6 to 9. Alkalimetric titration curves for Dolly Creek water on September 3, 4 and 5 showed acidities to pH 9 of 25, 23 and 22 mg $CaCO_3$/L, from initial pH values of 7.8, 7.9 and 7.9 respectively. If treated effluent has a pH during normal operation as high as pH 11, then Eq. 8 indicates that 25(11-9) = 50 mg $CaCO_3$/L of alkalinity require neutralization to pH 9 by the stream. This requires a minimum dilution of treated effluent by stream water of 50/22 = 2.3 times. Because flow measurements indicate that the streamflow is more typically ten times the flow of mine drainage (Table 19) the stream pH should be approx 8.4.

RECOMMENDED PLAN

Item 7 of the Conclusions section of this report recommends first stage construction of a limestone barrier and sedimentation basin, to the limit of the $150,000 available for construction funds. It presently appears that the cost of a chemical neutralization plant and storage tank would unduly deplete funds needed for basin construction, quite apart from the suboptimal inequality of marginal costs between neutralization and sedimentation that would occur. However, these matters, and others mentioned below await consideration in detail design.

Safety of a basin dam against catastrophic failure as a result of overtopping (or other failure modes such as piping) is a paramount design concern. In design, the reliability and suitability in other respects of several features to protect or strengthen the dam will be considered. These features may include an erodible plug in the dam approach channel to divert flow from the basin from the time of the spring flush until the plug is repaired in early summer. A series of small shallower basins may be less prone to failure than a larger (but probably more efficient) basin, although the greater possibility of damage to smaller basins by off-the-road vehicles needs consideration. If estimated surface runoff into the basin from the hillside is of concern, it may be feasible to arrange the basin so as not to disturb existing surface runoff patterns from the hillside. A related concern is to avoid the possibility of a spillway (constructed to remove water safely from the basin) diverting water into the basin to dangerous levels. Basin safety is of such concern that one wonders whether it may be preferable to release the copper-rich sediment from the barrier into the stream, where its effect on stream life appears likely to be much less deleterious than the dissolved copper presently discharged.

The degree of reduction of dissolved copper in the barrier is not yet subject to engineering analysis, because mechanisms for conversion of copper to solid form by contact with limestone may not necessarily be related to the pH of the water, as is the case for chemical precipitation. Chemical effects at the surface of the limestone, rather than in the bulk water, may well govern copper removal from water in a barrier. Because copper sediment formed from copper in the water does not appear to be gelatinous (as is the case for iron or aluminum), minimal visible fouling of the limestone surface occurs. Several effects observed in the pilot barrier are indicative of continued conversion of copper from dissolved form to sediment form by the limestone: 1) the 'wave' of removal of total copper by the barrier can plausibly be considered in terms of an accumulation of sediment within the stone that slowly moves down the barrier, finally to appear in barrier effluent at the same rate as dissolved copper enters the upper end of the barrier; 2) fluctuating levels of copper in barrier effluent after the wave has passed, when the concentration of copper in barrier influent is relatively stable, is consistent with the idea of effluent copper being largely in solid form rather than in solution; and 3) the consistency of the ratio of the imputed mass of metal stored in the pilot barrier to the mass removed by the barrier between metals, up to when barrier froze, is direct evidence for conversion of metals to sediment.

Consider that, while an understanding of the chemical mechanisms at work in barriers is advantageous for designing a barrier to perform as expected, lack of such knowledge does not prevent a barrier from working. Because effective, albeit poorly understood, metals removal mechanisms have been demonstrated in the Walker pilot barrier, and a limestone barrier also neutralizes the water, and can later be incorporated into a multi-process plant providing a higher level of treatment, there is much to be said for this unit as an almost operation free first stage of development. For these reasons, a limestone barrier is recommended for the Walker prototype plant, together with a sedimentation basin if feasible.

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Draft

PILOT PLANT OPERATION

DECEMBER 1982 TO JULY 1983

and

DESIGN REPORT

for

TREATMENT OF MINE DRAINAGE

WALKER MINE, PLUMAS COUNTY, CALIFORNIA

prepared for

CALIFORNIA REGIONAL WATER QUALITY CONTROL BOARD

CENTRAL VALLEY REGION

by

PEARSON AND ASSOCIATES

RICHMOND, CALIFORNIA

under

REGIONAL BOARD PROJECT 1 - 043 - 150 - 0

September, 1983

BOTTOM LINE

For treatment facilities at Walker mine to remove about eighty percent of copper from the adit discharge, the preliminary estimate of construction cost is about half a million dollars.

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PILOT PLANT OPERATION, DECEMBER 1982 TO JULY 1983

Visits to Walker Mine During 1983: Walker mine was visited on May 14-15 (winter walk-in visit) and July 16-20, 1983. A minimum winter temperature of -11C (12F) was recorded over the 1982-83 winter, using a thermometer located 12 ft above ground level and 6 in. beneath an uninsulated corrugated steel roof. In July, five days were spent at the mine running settlement tests on limestone barrier effluent, investigating the tailings as a source of pollution of Dolly Creek, and on July 20 inspecting the diversion ditches and the mine itself in the company of a group of visitors, namely: Robert Barry, Kathy Levine and Jan Donato representing mine ownership; Patricia Leary, Jack Del Conte and Bill Marshall representing the California Regional Water Quality Control Board - Central Valley Region; Charles St~~o~~urey and Kenneth Roby ~~Robles~~ representing the U.S. Forest Service; and Mark Pisano representing the California Department of Fish and Game.

Mine Drainage Flow: A peak flow of mine drainage from the portal of 830 gpm had been recorded over the 1981-82 winter. In mid-May 1983 the flow raw mine drainage was 190 gpm, with a 1,000 gpm peak recorded since December, 1982. Between May and July 1983 a 1,480 gpm peak was registered, the flow in mid-July being 320 gpm. Because the earlier 1,000 gpm peak occured before the time of peak snowmelt it may have been associated with mine repair work in progress in May. A flush would have followed removal from the mine of collapsed roof material that had impounded water in the mine; workers gained access to the mine in May by Snocat.

Dolly Creek Flow: In July 1983 the Dolly Creek weir gauging rod was found to be correctly zeroed, within the 1 mm tolerance of the check measurement. However, the weir calibration equation used for calculation of flows listed in the March 1983 pilot plant report required correction for approach velocity head.

Approach velocity head correction can be made on the basis of the 0.27 ft mean depth of the stream bed below the weir, as measured in July, 1983. Correction can also be made for the weir side contractions, by subtracting from the 3.77 ft width of the weir a contraction on each side equal to one-tenth the hydraulic head on the weir. The resulting theoretical calibration curve for the weir is listed in Table 1, together with the program used for its calculation in Table 2. For given stream stage readings Table 1 shows flows about 5% lower than those listed in the March 1983 operating report for the period September to December 1982.

This decrease in the estimate of flow in Dolly Creek slightly reduces the estimated mass loadings of metals from the tailings, although the recalibration adjustment is less than the probable tolerance in the previously calculated mass loadings.

In mid-May and mid-July of 1983, Dolly Creek weir flows were 1,340 and 1,280 gpm.

Neutralization-Sedimentation Pilot Plant: It had been intended

TABLE 1 : Theoretical Rating Curve for Dolly Ck Flow Measuring Weir

Head, millimeters	Flow, gallons per minute	Head, millimeters	Flow, gallons per minute	Head, millimeters	Flow, gallons per minute	Head, millimeters	Flow, gallons per minute	Head, millimeters	Flow, gallons per minute	Head, millimeters	Flow, gallons per minute
1	1	51	363	101	1,029	151	1,892	201	2,904	251	4,032
2	3	52	374	102	1,044	152	1,911	202	2,926	252	4,055
3	5	53	385	103	1,060	153	1,930	203	2,948	253	4,079
4	8	54	396	104	1,076	154	1,949	204	2,969	254	4,102
5	11	55	408	105	1,091	155	1,968	205	2,991	255	4,126
6	14	56	419	106	1,107	156	1,988	206	3,012	256	4,150
7	18	57	430	107	1,123	157	2,007	207	3,034	257	4,173
8	22	58	442	108	1,139	158	2,026	208	3,056	258	4,197
9	27	59	453	109	1,155	159	2,045	209	3,078	259	4,221
10	31	60	465	110	1,171	160	2,065	210	3,100	260	4,245
11	36	61	477	111	1,187	161	2,084	211	3,122	261	4,269
12	41	62	489	112	1,204	162	2,104	212	3,143	262	4,293
13	46	63	501	113	1,220	163	2,123	213	3,165	263	4,317
14	51	64	513	114	1,237	164	2,143	214	3,188	264	4,341
15	57	65	526	115	1,253	165	2,163	215	3,210	265	4,365
16	63	66	538	116	1,270	166	2,182	216	3,232	266	4,389
17	69	67	550	117	1,286	167	2,202	217	3,254	267	4,413
18	75	68	563	118	1,303	168	2,222	218	3,276	268	4,437
19	82	69	576	119	1,320	169	2,242	219	3,298	269	4,461
20	88	70	588	120	1,337	170	2,262	220	3,321	270	4,485
21	95	71	601	121	1,354	171	2,282	221	3,343	271	4,509
22	102	72	614	122	1,371	172	2,302	222	3,365	272	4,534
23	109	73	627	123	1,388	173	2,322	223	3,388	273	4,558
24	116	74	640	124	1,405	174	2,342	224	3,410	274	4,582
25	123	75	654	125	1,422	175	2,362	225	3,433	275	4,607
26	131	76	667	126	1,439	176	2,382	226	3,455	276	4,631
27	139	77	680	127	1,457	177	2,403	227	3,478	277	4,655
28	146	78	694	128	1,474	178	2,423	228	3,500	278	4,680
29	154	79	708	129	1,492	179	2,443	229	3,523	279	4,704
30	162	80	721	130	1,509	180	2,464	230	3,546	280	4,729
31	171	81	735	131	1,527	181	2,484	231	3,569	281	4,753
32	179	82	749	132	1,545	182	2,505	232	3,591	282	4,778
33	188	83	763	133	1,562	183	2,525	233	3,614	283	4,803
34	196	84	777	134	1,580	184	2,546	234	3,637	284	4,827
35	205	85	791	135	1,598	185	2,567	235	3,660	285	4,852
36	214	86	805	136	1,616	186	2,587	236	3,683	286	4,877
37	223	87	820	137	1,634	187	2,608	237	3,706	287	4,901
38	232	88	834	138	1,652	188	2,629	238	3,729	288	4,926
39	242	89	848	139	1,670	189	2,650	239	3,752	289	4,951
40	251	90	863	140	1,688	190	2,671	240	3,775	290	4,976
41	261	91	878	141	1,707	191	2,692	241	3,798	291	5,001
42	270	92	892	142	1,725	192	2,713	242	3,821	292	5,025
43	280	93	907	143	1,743	193	2,734	243	3,845	293	5,050
44	290	94	922	144	1,762	194	2,755	244	3,868	294	5,075
45	300	95	937	145	1,780	195	2,776	245	3,891	295	5,100
46	311	96	952	146	1,799	196	2,798	246	3,914	296	5,125
47	321	97	967	147	1,817	197	2,819	247	3,938	297	5,150
48	331	98	983	148	1,836	198	2,840	248	3,961	298	5,175
49	342	99	998	149	1,855	199	2,862	249	3,985	299	5,201
50	353	100	1,013	150	1,874	200	2,883	250	4,008	300	5,226

TABLE 2: Microsoft Basic Program To Compute Dolly Ck Weir Theoretical Rating Curve.

```
10 DIM A(300,2):FOR I=1 TO 300:A(I,1)=1.06*I^1.5:NEXT I:G=32.17
20 FOR I=1 TO 300:B=A(I,1)/448.83:H=I/304.8:F=SQR(G/1.5^3)*(3.77-.2*H)
30 A=F*(H+1.4*(B/(H+.27)/(H+5))^2/2/G)^1.5
40 IF ABS(B/A-1)>.00001 THEN B=A:GOTO 30
50 A(I,2)=448.83*A:PRINT I,A(I,1),A(I,2):NEXT I:L=2
60 LPRINT "TABLE   : Theoretical Rating Curve for Dolly Ck Flow Measuring Weir"
70 LPRINT:FOR I=1 TO 6:LPRINT TAB(15*I-14) "Head,  Flow,   ";:NEXT I
80 LPRINT:FOR I=1 TO 6:LPRINT TAB(15*I-14) "milli- gallons";:NEXT I
90 LPRINT:FOR I=1 TO 6:LPRINT TAB(15*I-14) "meters per     ";:NEXT I
100 LPRINT:FOR I=1 TO 6:LPRINT TAB(15*I-14) "       minute ";:NEXT I:LPRINT
110 LPRINT:FOR I=1 TO 50:FOR J=1 TO 6:LPRINT TAB(15*J-14);:K=I+50*(J-1)
120 LPRINT USING "###";K;:LPRINT STRING$(4,46);:LPRINT USING "####,";A(K,L);
130 NEXT J:LPRINT:NEXT I:STOP
```

that the pilot neutralization-sedimentation plant should operate unmanned over the winter period when access is blocked by snow. The plant was set up for winter operation in December 1982, with Snocat transportation to the site provided by the Beckworth office of the California Department of Water Resources.

On reinspection in May 1983 the plant was not operating. Rupture of a joint in the feed pipe upstream of the plant as a result of snow loading caused raw mine drainage to bypass the plant.

In May about three feet of snow covered the crest of the sedimentation basin dam. On the next visit, in July 1983, the dam was found to retain water, although the PVC feed pipe from the neutralization plant had snapped from snow loading, as had the sedimentation basin effluent drawoff pipe.

Snow loading had also collapsed the roof of the mine portal building (that was subsequently removed), and snapped about one half of the roof joists in the contiguous core storage building (that was slated for repair). An eight foot deep drift of compacted snow lay by these buildings in mid-May.

<u>Limestone Barrier:</u> The limestone barrier was found to be in operation in mid-May beneath a snow drift five feet deep at the outlet end of the barrier. The flow of effluent from the barrier was 0.2 gpm, indicative of partial clogging of the pipe feeding influent to the head of the barrier. The effluent pH of 6.8 was comparable with the mean for late 1982 of 6.5. Possibly the May

1983 pH was slightly higher than would have occurred in the absence of dilution by meltwater from overlying snow.

On the next visit in mid-July 1983 flow from the barrier had ceased. Investigation showed that the 1 in. PVC intake section of the barrier feed pipe had become blocked by stones. Flow was restarted and adjusted to 1.0 gpm, and the next day a pH profile was determined on samples at 25 ft intervals along the 500 ft long barrier. Results listed in Table 3 show that the effluent pH had risen to 7.5, suggesting that the restoration of barrier performance was at least partly caused by higher temperatures.

TABLE 3: pH Profile Along Limestone Barrier in July 1983.

Distance, feet	pH, units	Distance, feet	pH, units	Distance, feet	pH, units
0	4.6	175	6.3	350	7.3
25	4.8	200	6.5	375	7.4
50	4.9	225	6.8	400	7.4
75	5.2	250	6.9	425	7.4
100	5.4	275	7.0	450	7.5
125	5.9	300	7.2	475	7.5
150	6.1	325	7.2	500	7.5

<u>Limestone Barrier Effluent Settling Studies:</u> On the basis of 1982 experience, it appeared that a limestone barrier followed by a sedimentation basin might provide a useful degree of reduction of copper loading to Dolly Creek at minimal cost for construction and operation. Studies were made in July 1983 to investigate this possibility. Barrier effluent settleability tests were conducted in a settling column, and a sedimentation basin was set up, fed by barrier effluent.

Barrier Effluent Settling Column Studies: Three runs of a 15 ft x 6 in. diam settling column were made. In the first run copper precipitate residue remaining in the column from previous chemical neutralization tests was suspended in barrier effluent and settled. In the second run barrier effluent was settled in a cleaned column, and in the third a cleaned column was used to settle partly treated mine water from the midpoint of the barrier. The reason for using copper precipitate from earlier work in the first run was to permit comparison between the settleabilities of chemically precipitated copper and limestone-precipitated copper. However, the green precipitate observed in chemically neutralized mine water was not seen in barrier effluent, except as occasional glints in sunlit barrier effluent.

Table 4 lists copper concentrations determined by atomic adsorption spectroscopy in water samples drawn from the settling column during each of these three runs, for specified times after startup of the settling column, and specified distances up from the column base. Listed concentrations of copper in column water may be compared with the 17.4 mg/L concentration of copper in raw mine water.

Part A of Table 4 emphasises the protracted time of settlement needed to remove precipitated copper in a sedimentation basin, a matter discussed at length in the March 1983 pilot plant report.

Part B of Table 4 shows that only a small quantity of copper was captured from barrier effluent in the settling column (1.1 mg/L

TABLE 4 : Copper Concentrations in Samples From Settling Column Containing Mine Water Treated By Limestone Barrier.

A: Chemically Precipitated Copper Suspended in Barrier Effluent.

Time,	Distance from base of settling column, ft					
min	0	1	2	4	7	13
15	4,200	2,000	1,600	31	12	7.4
30	4,200	2,000	670	11	9.0	5.9
60	3,000	1,800	12	6.3	6.5	5.2
180	4,000	36	4.6	4.4	4.8	3.6
960	4,000	2.3	3.1	2.4	2.7	2.2

B: Barrier Effluent.

Time,	Distance from base of settling column, ft					
min	0	1	2	4	7	13
15	0.89	0.77	2.1	0.66	0.54	0.93
30	0.81	0.66	0.74	0.54	0.46	0.43
70	0.66	0.54	0.54	0.43	0.43	0.46
180	0.62	0.39	0.43	0.37	0.39	0.43
360	0.46	0.43	0.39	0.39	0.35	0.35
660	0.35	0.31	0.31	0.35	0.35	0.35
1440	0.39	0.31	0.35	0.35	0.31	0.50
1880	0.35	0.39	0.31	0.31	0.31	0.39

C: Partly Neutralized Mine Water From Mid-Point of Barrier.

Time,	Distance from base of settling column, ft					
min	0	1	2	4	7	13
15	-	1.6	1.8	0.89	0.46	0.31
30	1.8	1.2	0.93	0.85	0.58	0.39
60	1.4	0.85	0.85	0.70	0.58	0.39
180	0.62	0.77	1.4	0.66	0.85	0.62
900	1.4	0.50	0.83	0.54	0.97	0.35
1440	1.6	0.39	0.81	0.46	0.35	0.50
2100	4.0	0.52	1.1	0.35	0.66	1.4
3000	4.1	0.31	-	-	-	-

at 15 min) due to removal of 94% of copper in the barrier (from 17.4 mg/L in raw mine drainage). Above one foot from the base of the settling column the further removal of copper by sedimentation was essentially independent of depth, but increased with time as previously noted for chemically precipitated copper. The lower limit for copper in the settlement column was 0.34 mg/L, which matches the 0.3 mg/L theoretical solubility of copper at the prevailing barrier effluent pH of 7.5. The 0.34 mg/L of copper in settled barrier effluent represents 98% overall copper removal from the 17.4 mg/L of copper in barrier influent.

Part C of Table 4 shows similar trends, except that the level of copper in mine water after flowing down one half of the length of the barrier is rather higher than for barrier effluent. Initially (at 15 minutes after column startup) the mean concentration of copper in the settling column was 0.9 mg/L, 95% below that in barrier influent. At the midpoint of the barrier the pH was 6.9. After one day (1,440 minutes) of settlement the mean concentration of copper in water above one foot from the base of the column was 0.5 mg/L, or 97% below that in barrier influent.

Spurious fluctuations in copper concentration sometimes obscured trends for copper removal to increase with time of settlement. These fluctuations may have resulted from the particulate state of the copper and the small (50 mL) size of water samples removed from the column for analysis. Larger samples would have excessively depleted the volume of water in the column during the run.

Copper in barrier water does not appear to be constrained above the theoretical solubility. Concentrations of copper as low as 0.3 mg/L were observed in samples from the midpoint of the barrier, one-fourth of the 1.2 mg/L theoretical solubility of copper for the pH of 6.9 occurring at that point. This observation is consistent with the concept that pH measurements on mine water under neutralization in a limestone barrier may not represent chemical conditions in the liquid film at the surface of the limestone where neutralization reactions occur.

Protons are depleted from the acid water at the limestone surface to a greater extent than measured by the pH of the bulk liquid. Chemical reactions leading to the alkaline precipitation of copper should then proceed more rapidly in the relatively high-pH environment of the liquid film at the surface of the limestone than might be expected from readings of the pH of the bulk liquid. Consequently more copper should be removed in a barrier at a given pH than by chemical neutralization to the same pH.

However encouraging these limestone barrier performance data may be, they are yet insufficient to establish long term barrier performance. To determine long term performance one must monitor a barrier-basin system for a sufficient period to establish a reasonable balance over a significant interval of time between the mass of the source of copper entering the system, and the total mass of copper leaving the system by all sinks. The primary sinks for copper are sludge removed and residual copper in the treated effluent, although non-quantifiable sinks for copper such as deep percolation from the sedimentation basin may also occur.

Barrier Effluent Sedimentation Basin: In addition to settling column studies, a sedimentation basin was set up. The purpose of the basin was to capture any sporadic release of suspended solids from the barrier that might have escaped the grab samples collected for column studies. A green copper-rich sediment that covered the limestone for the length of the barrier could be partly removed by disturbing the stone. The sediment was not gelatinous, in contrast to the coatings formed from limestone neutralization of acid water containing iron.

The sedimentation basin was 12 ft diam by 3 ft deep, with a surface overflow rate of 12.7 gal/sf-day at the flow rate of 1.0 gpm. It was fed from a point on the limestone barrier 60 ft from its lower end by a 1 in. polyethylene pipe that narrowed to a 1/2 in. hose for the rising section over the basin wall, the basin rim being slightly below the level of the downstream end of the barrier.

Over the two day period that the basin was observed during filling no precipitate was seen. Evidently, no significant quantity of precipitable copper was released from the barrier during these two days. On the other hand, a green copper precipitate had been readily visible in samples of chemically-neutralized mine water above pH 7, even in samples as small as one liter or less. Copper removed from the mine water in the limestone barrier was evidently retained as sediment within the barrier, the barrier stone being an efficient solids-capture device. The settleability of this sediment has not been determined.

Feasibility of Barrier-Basin Scheme: A treatment plan comprising a limestone barrier followed by a sedimentation basin continues to offer the prospect of avoiding the construction and operating costs and problems associated with a chemical feed system. But without knowing how copper would be released from the barrier and settled in the basin, no positive statement can be made on the long term degree of reduction of copper loading to the receiving water that such a barrier-basin system would achieve. On the other hand, procedures presented in the March 1983 pilot plant operating report may be used to estimate the performance of a particular design for a chemical neutralization-sedimentation plant.

Should this have been done?

Precipitation Of Copper With Phosphate or Iodide: Walker mine drainage collected on July 20, 1983 was divided into 100 mL aliquots. Each aliquot was dosed with either monobasic sodium phosphate or with sodium iodide at one of the following concentrations: 5, 10, 25, 50, 75, 100, 150, 200, 300 or 500 mg/L as P or I. Since no visible precipitate or color change occurred in any sample after one day, the dosed samples were not analyzed.

Tailings Area As A Source Of Pollutants

In the March 1983 pilot plant operating report about one-third of wet season metals loadings in Dolly Creek below Walker mine were attributed to seepage from the tailings. This contribution of metals to Dolly Creek from the non-point tailings source was inferred as the difference between mass loadings of analyzed metals in Dolly Creek below the mine and the sum of the point source mass loadings.

To attempt to obtain more direct information about the tailings source, the tailings area was inspected in mid-July 1983 for evidence of production of acidity and metals. The inspection comprised collection of water samples where seepages appeared at the ground surface, and collection of minus #10 mesh soil samples at surveyed points within the quadrilateral-shaped tailings area generally outlined by the pegs P, Q, R and S, shown in Fig. 1. Figures 2-5 show cross-sections of the ground profile at peg locations identified in Fig. 1. (Figures referred to in this report are a set of eleven 18 x 24 in. blueline prints.)

Of the 45 soil samples collected 42 were found to be pyritic as evidenced by gold-colored flecs, particularly on viewing a wet sample of the soil through glass. Soil samples in which pyrite was not observed were those collected at points designated by Pegs S+50, S+100 and S+150, where the soil was brown volcanic ash rather than the gray sand tailings elsewhere. Also, the soil was less pyritic in the area within about 100 ft of Peg R, in which area the soil became rather plastic. Mr. Jan Donato reported that a dozer had once been stuck on attempting to excavate in that area.

Blue or white salt incrustations were generally seen near where seepages surfaced. These incrustations remain after evaporation of saline seepages. Seepages designated by the nearest survey peg numbers were analyzed for pH and copper by atomic adsorption spectrophotometer, with the results listed in Table 5.

TABLE 5: pH and Copper Concentration of Tailings Seepages.

Point	pH	Copper, mg/L
P+50	4.7	13.2
P+75	6.2	10.3
P+100	6.0	8.7
P+306	4.7	15.3
P+356	4.2	16.7
P+406	6.2	1.4
R+50	3.4	27.5
Seepage from clean stream	7.1	0.15
Raw mine water	4.4	17.4

Most seepages varied in degree of contamination up to that for the adit discharge, as for the 'P' series of sampling points along the upslope edge of the flat tailings area. The strongest seepage, from the base of a bank near Dolly Creek near Peg R+50, contained about 60% more copper and roughly ten times as much acidity as raw mine water (by virtue of its pH being one unit lower). The least contaminated seepage from the base of the tailings resulted from diversion of clean stream water into the top of the tailings about 200 ft away. The 200 ft passage of water through the tailings increased the concentration of copper from 0.10 mg/L to 0.15 mg/L, and reduced the pH from 7.8 to 7.1.

Two surface streams about 100 yards east of the mine portal are also lightly contaminated. In July 1983 the nearer stream contained 0.10 mg/L of copper and had a pH of 7.6, and the farther contained 0.35 mg/L of copper and had a pH of 7.8. For comparison, uncontaminated Dolly Creek upstream of the mine workings contained 0.04 mg/L of copper and had a pH of 7.8 at that time.

Although the hydrogeochemistry of the tailings is obscure, a potential for significant seepage from the tailings to Dolly Creek following rainfall or snowmelt is evident. The mean discharge from the mine adit, taken to approximate 0.5 cfs, is about the same rate of flow as that into the tailings into which rain steadily infiltrates at the modest rate of 0.1 in./hr.

<u>Characterization of Soil In Tailings Area:</u> Soil in the tailings area appears to be largely a sandy residual from mineral processing. Table 6 shows the grading of a sample.

TABLE 6: Size Grading Of Walker Mine Tailings Area Soil Sample.

U.S. standard sieve size	Percent passing
200	5.6
100	14.6
65	35.6
50	51.7
20	89.5
16	90.9
12	97.3
4	99.2

The soil is yellowish-grey with some white and a few black particles; golden pyritic flakes could be seen. The particles are subangular on microscopic examination. The voids ratio ranges from a minimum of 0.48 when loose to 0.36 when compacted.

The Unified Soil Classification System defines sands in the following size ranges:

Coarse sand	#4 sieve (4.76 mm)	to	#10 sieve (2.00 mm)
Medium sand	#10 sieve (2.00 mm)	to	#40 sieve (0.42 mm)
Fine sand	#40 sieve (0.42 mm)	to	#200 sieve (.074 mm)

With an effective size (D60) of approximately 0.4 mm, the material at the Walker site would be classified as fine sand.

The permeability of this sand was measured using a falling head permeameter. For a 310 mm deep sample in a 25 mm diameter permeameter tube, the hydraulic head reduced from 947 mm to 485 mm in 45 minutes at 22C. From these experimental data the permeability of the sand is calculated to be 0.0077 cm/sec (8,000 ft/yr). This value is representative for sand. It compares with a range of 0.01-0.015 according to Hazen's emperical equation for loose, uniform sand with a 90% exceedance size of 0.10 mm, interpolated from Table 6.

Soil as permeable as this sample of Walker tailings soil is generally designated as suitable for the pervious sections of dams and dykes. Seepage though the Walker tailings area embankment is likely to result in moistness of the lower one-third or so of the embankment on the Dolly Creek side. Provided this seepage remains steady and slight it is not indicative of incipient piping, and may help in the reestablishment of vegetation on the tailings area. A hydraulic gradient approaching unity is needed to initiate piping heave, while the slope of the downstream face of the dam embankment is limited to flatter than 1:8.

Mine Inspection

Raw mine drainage discharging from the mine portal contained 17.4 mg/L of copper on July 20, 1983 when an inspection visit was made into the mine as far as the Central ore body. One purpose of the inspection was to obtain water samples to characterize the variability of the copper content of mine water.

On travelling up the slightly inclined (1%) rail tunnel into the mine it appeared that most of the water entered the rail tunnel from the shafts that intersected from above. Some wet weather seepage directly through the roof of the rail tunnel near its mouth had been noted the previous year. Water entering the rail tunnel through a fire door between the South and Central ore bodies contained 6.7 mg/L of copper.

It had been suggested that by preventing mine water from entering the inundated mine workings below the level of the rail tunnel, a portion of the load of copper that originates from the inundated workings might be abated. However, this could not be demonstrated by analysis of copper in the subterranean stream that entered the inundated workings in the Central ore body. The stream entering the inundated workings contained 17.7 mg/L of copper, as much copper as water leaving the mine. Drainage from an ore shute into the rail tunnel at the south end of the Central ore body contained 43.6 mg/L of copper.

In-mine source abatement measures for which at least some degree of success has been claimed at other locations include sealing the mine so as to inundate a controlled section of the workings. At Walker the workings are said to extend 500 ft vertically above the adit, and are open to the air at locations other than the adit. There is a convective breeze into the adit during winter and from the adit in summer.

Consequently, a seal that develops a nominal head of a few feet or tens of feet cannot be expected to drastically reduce the loadings of acidity and metals in the mine drainage. If a seal or several seals to develop hundreds of feet of hydraulic head were envisaged, the structural integrity of the rock to safely withstand the resulting pressures would require investigation. Further, the Walker mine is understood to have national strategic significance as an emergency copper mine, a purpose that could not be fulfilled if the mine were sealed.

Lets investigate?

Who says?

Why not?

Another suggested approach is diversion of uncontaminated water within the mine to reduce the hydraulic loading on treatment.

This would extend to the subterranean the concept of the surface diversion ditches already implemented. Because hydraulic design of the processes is largely determined by the need to handle extraordinary flood flows that cannot safely be assumed to diminish with these measures, the major advantage of reducing the hydraulic loading is a reduction in chemical consumption. However, water sources within Walker mine that contain a sufficiently low copper concentration for direct discharge to Dolly Creek remain to be located. Aspirations to stream water quality standards approaching those specified in the Basin Plan may preclude in-mine diversion, for substantial flows containing as little copper as the 0.01 0.1 mg/L specified in the Basin Plan appear unlikely to be found in the mine. Even surface streams near the mine portal contain more copper than 0.1 mg/L.

The following treatment processes are to be employed:

pre-neutralization by crushed limestone;

neutralization by caustic soda or soda ash;

sedimentation of precipitated metals, primarily copper;

gravity thickening of settled sludge; and

natural evaporation of thickened sludge.

Design Overview:

Mine drainage flows from the adit in a 30 inch diameter asbestos-cement pipe, to discharge into the head of a canyon. In the canyon the flow falls 90 ft over its 800 ft passage to the Dolly Creek confluence. From an initial gradient exceeding 30% the mine drainage stream flattens to 3% over its course, with 75 ft of the total fall in the upper 380 ft long section. The upper section of the stream terminates at a point designated on Fig. 1 as Peg P (which is the same as Peg S+300). This peg is the stake identified in Figs. 71-73 of the March 1983 pilot plant operating report as in the vicinity of the neutralization plant site.

Indeed, the site for a crushed limestone barrier for pre-neutralization of the mine drainage is in the steep section of the canyon channel, and for a chemical neutralization plant at Peg S+300. At this point the stream flattens below the 10% gradient tested in the pilot limestone barrier, and may no longer be suitable for a prototype limestone barrier.

Below Peg S+300 the stream of mine drainage skirts the sandy tailings area, the upslope edge of which has been excavated to form a basin up to 15 ft deep. A sedimentation basin constructed in this excavation will receive chemically neutralized mine drainage. Most of the construction cost for the prototype facility is associated with hydraulic structures for removing settled effluent from the sedimentation basin.

The reason for the relatively high cost of sedimentation basin outlet structures is that these structures may possibly have to safely remove from the sedimentation basin extraordinarily high flows associated with extreme precipitation and snowmelt. The magnitude of these extreme events is estimated from theoretically derived rainfall intensities called maximum probable precipitation, that are a commonly recommended basis for designing spillways for Federal dams. Because these extreme events are not estimated from statistical analysis of recorded events, there is no specific return period associated with the extremes. However, the extremes are generally assumed to be the upper limit of historical and projected future events.

Being designed to carry a high flow, the sedimentation basin outlet structure is relatively large. If most of the high flow is diverted around the basin then smaller, less expensive basin outlet structures would carry the reduced peak flow through the basin. However, smaller outlet pipelines are more prone to jamming by logs, displaced mine lumber and similar debris.

Jamming of the outlet would probably lead to topping, breaching and destruction of the dam. The same disastrous effect would result whether the outlet structure trash rack or the outlet pipe became jammed.

To avoid the risk of jamming the spillway an attempt was made to design an overflow-type basin spillway located in the tailings between the excavated basin and Dolly Creek. No safe design for such a spillway was identified, on account of difficult foundation conditions associated with the fine sandy tailings material, aggravated by the high earthquake and ice loadings. Lack of knowledge of subsurface conditions is a major impediment to confidence in design.

Another site exists for an overflow-type spillway, located near Peg Q on Fig. 1. If it were desired to investigate the feasibility of a spillway at this site subsurface exploration and specialist design would be advantageous, because the structure would be rather deep and surface indications of subsurface conditions are meagre. Otherwise, the safer course is to design a pipeline-type basin outlet of sufficient size that the risk of clogging appears minimal.

Water can be drawn into a pipeline-type basin outlet at a drop inlet or "Glory Hole" spillway, comprising an open-topped concrete box set in the reservoir with the rim of the box at the desired overflow level. From the base of the drop inlet water passes through the embankment in a pipeline, alternatively termed a penstock. A stilling basin at the outlet of the penstock dissipates the energy of the flow that would otherwise cause erosion at the outlet.

Figure 6 shows the location of the various treatment units on a contour map of the mine area. The limestone barrier is shown as starting 80 ft downstream of the outlet of the 30 in. pipe from the mine, where the stream gradient has flattened sufficiently to reduce the risk of scour damage to the barrier. At its lower end the 300 ft long barrier abuts the chemical neutralization plant and chemical storage tank structure, where the mine drainage is chemically neutralized and diverted into the sediment basin. A spillway outlet structure decants clarified neutralized water from the basin, that passes through the penstock to the stilling basin before entering Dolly Creek. A manhole permits a change of direction of the penstock, to avoid less stable soils along the straight path.

The removal and dewatering of sludge from the sedimentation basin is likely to be one of the more onerous operating tasks. The best means to remove sludge remains to be developed under the conditions of operation of the prototype facility. However, in the pilot plant sedimentation basin settled sludge consolidated over several months to a dense paste that might be skimmed from the floor of a dewatered basin by skip loader. Sludge readily evaporates to a dry cake in summer air to a dry cake, containing 30% by weight of copper in the case of pilot plant sludge.

Dried sludge may be saleable for its copper content, but more probably would require disposal in the mine slumps at the South or Central ore bodies. Based on a mean flow from the adit of 0.5 cfs of mine drainage containing 15 mg/L of copper, the annual quantity of 30% copper sludge is estimated at 25 tons. Minor construction is needed to provide four wheel drive vehicular access to a slump. Any leachate released from sludge stored in the slump would return to the mine, but because the floors of most slumps have no direct openings to the mine it can be expected that the sludge would not return to the mine.

A key design question is the expected level of performance of the treatment facility with respect to removal of copper, that depends on the degree of chemical neutralization and the efficiency of sedimentation. As explained in the March 1983 pilot plant operating report, for any neutralization pH in the range 6 to 11 there is an ultimate percentage removal of copper attainable only in a quiescent batch sedimentation tank, that represents the upper limit of the degree of removal of copper in real processes with imperfect sedimentation.

The expected removal of copper in a treatment process is the product of the pH-dependent ultimate percentage removal and the efficiency of sedimentation. According to Eq. 6 on page 53 of the March 1983 pilot plant operating report, for the 1.4 acre water surface area of a sedimentation basin with a water level of 6090 feet (Fig. 6), at the peak 1982-83 flow of 1,480 gpm the sedimentation efficiency is 85%. Equation 5 on the same page provides for calculation of ultimate removal for any given pH. Table 7 lists the calculations of ultimate and expected removals of copper, and on the expected concentration of copper in the effluent assuming 15 mg/L in raw mine drainage.

TABLE 7: Effluent Copper Concentration Versus Treatment pH.

Treatment pH	Ultimate removal, percent	Expected removal, percent	Effluent copper, mg/L
6.5	68	58	6.3
7.0	90	77	3.5
8.0	94	80	3.0
9.0	97	82	2.7
10.0	98	83	2.5
11.0	99	84	2.3

Observe in Table 7 how with increasing treatment pH, the expected percentage removal of copper approaches the 85% limiting value dictated by the efficiency of the sedimentation basin. To increase the efficiency of the basin would require a significantly larger basin. For example, with a basin twice as large (2.8 acres) the efficiency of sedimentation increases from 85% to 88%, for a minimum attainable effluent copper concentration as low as 1.8 mg/L. However, a basin this large on the tailings area would require structural walls. No plan involving enlargement of the basin from its present contours is considered in this report, because such a plan would encroach on the existing basin embankments.

The remainder of this report addresses specific aspects of the design, and briefly considers some O&M concerns.

Design Flood Potentially Intercepted By Tailings Area

If a sedimentation basin were constructed in the tailings area as a component of a mine drainage treatment system, hydraulic works are needed to contain any flood flow from the tributary watershed that the basin may intercept. Either ditches may be constructed around the uphill side of the basin to divert runoff into Dolly Creek, or the basin spillway may be designed to carry runoff entering the basin in addition to the flow from the mine.

Neither alternative is likely to prevent scouring of settled copper from the basin into Dolly Creek in the event of a design flood, for under meterological conditions producing a design flood the discharge from the mine tunnel may well increase enough to cause scouring, even if surface runoff were diverted from the basin. No engineering procedure exists for estimating from available or readily obtainable data the design flow from the mine adit as a result of probable maximum precipitation, but a number tentatively representing this flow based on a simple assumption is later presented.

A choice based on economics and safety has to be made between diversion ditches around the basin with smaller basin outlet works versus no ditches and larger basin outlet works. Ditch maintenance needs consideration, without which failure of the ditch and of the basin are likely if a design storm occurs.

So what?

Also, consideration is given to whether Dolly Creek may rise sufficiently during a design storm to threaten the outer embankments of the basin. In this case channel improvements in the creek may be needed. Channel improvements may destroy the beaver habitat in Dolly Creek. Alternatively, rip-rapping might used to protect the fine sand of the embankment from scouring by flood waters in this steep creek, although a considerable number of layers of screened material would be needed for stability, graded gradually from large rock on the outside to sand on the inside.

The first step in these evaluations is to estimate design flood flows tributary to the basin, and in Dolly Creek, and to select a value representing the design flood flow from the mine adit. In these calculations reference is made to figures and tables in the U.S.B.R. publication "Design of Small Dams." Figure 7 shows the watershed tributary to the basin, and Table 8 shows topographic data for this watershed.

Characteristics of Watershed Tributary to Sedimentation Basin

Area drained by sedimentation basin, acres . . .	290
Maximum length of watershed, miles, L	1.13
Elevation range of watershed, feet, H	1,200
Time of concentration, hours (Fig. 13)	0.2

Design Rainfall

Location: Plumas County, California

Maximum probable 6 hr rainfall, in. (Fig 3)	10
Rainfall time distribution zone (Fig. 3)	A
Reduction factor where human life not endangered (Fig. 7)	1.8
Design 6 hr rainfall, in. (10 in./1.8)	5.6

TABLE 8 : Coordinates Of Watershed Boundary.

North coord, feet	East coord, feet	North coord, feet	East coord, feet	North coord, feet	East coord, feet	North coord, feet	East coord, feet
12160	18700	15200	17230	16570	20700	15000	21440
12180	18600	15260	17220	16640	20800	14900	21330
12210	18500	15270	17300	16690	20900	14800	21220
12240	18400	15260	17400	16730	21000	14700	21100
12270	18300	15250	17500	16770	21100	14600	21000
12290	18200	15240	17600	16790	21200	14500	20960
12320	18100	15260	17700	16810	21300	14400	20960
12350	18000	15320	17800	16840	21400	14300	20960
12400	17900	15400	17900	16850	21500	14200	20980
12500	17850	15490	18000	16880	21600	14100	20980
12600	17840	15590	18100	16890	21700	14000	20970
12700	17830	15640	18200	16930	21800	13900	20930
12800	17800	15660	18300	16990	21900	13800	20890
12900	17750	15680	18400	17040	22000	13700	20790
13000	17690	15710	18500	17090	22040	13600	20660
13100	17650	15740	18600	17000	22020	13560	20600
13200	17600	15750	18700	16900	22010	13520	20500
13300	17570	15750	18800	16800	22010	13460	20400
13400	17580	15760	18900	16700	22010	13380	20300
13500	17590	15780	19000	16600	22010	13320	20200
13600	17600	15830	19100	16500	22010	13240	20100
13700	17600	15890	19200	16400	22010	13160	20000
13800	17600	15960	19300	16300	22010	13080	19900
13900	17590	16020	19400	16200	22000	13020	19800
14000	17580	16090	19500	16100	22000	12960	19700
14100	17580	16130	19600	16000	22000	12910	19600
14200	17570	16160	19700	15900	22000	12820	19500
14300	17540	16160	19800	15800	22000	12680	19400
14400	17500	16170	19900	15700	21980	12590	19300
14500	17460	16180	20000	15600	21920	12530	19200
14600	17430	16190	20100	15500	21870	12480	19100
14700	17380	16220	20200	15400	21820	12420	19000
14800	17350	16260	20300	15300	21750	12370	18900
14900	17310	16320	20400	15200	21660	12310	18800
15000	17280	16400	20500	15100	21550	12160	18700
15100	17240	16500	20600				

First and last points listed are lowest point of basin.
Watershed area tributary to sedimentation basin = 289.871 acres.
Elevation of highest point in basin = 7,260 ft.
Elevation of lowest point in basin = 6,060 ft.
Elevation difference from lowest to highest point = 1,200 ft.
Horizontal distance from lowest to highest point = 5,955 ft.

Design Snowmelt

a) Due to warmth of rain
   Meltwater = rainfall x (rain temp F -32F)/144

b) Due to warmth of air
   Meltwater, in./day = 0.02 to 0.13 x (air temp F - 32F)

c) Due to warmth of air and rain
   Assume 60F air, 10 hr melt day
   Total meltwater, in./hr = 0.2 x rainfall, in./hr + 0.2

Design Total Rainfall Plus Snowmelt (Half-hour time intervals)

Interval number	Percent 6 hr rain	Cumulative rain, in.	Incremental rain, in.	Incremental total, in.
1	25	1.40	1.40	1.78
2	40	2.20	0.80	1.06
3	50	2.80	0.60	0.82
4	57	3.20	0.40	0.58
5	63	3.50	0.30	0.46
6	68	3.80	0.30	0.46
7	73	4.10	0.30	0.46
8	79	4.40	0.30	0.46
9	84	4.70	0.30	0.46
10	90	5.00	0.30	0.46
11	95	5.30	0.30	0.46
12	100	5.60	0.30	0.46

Design Runoff

Design flood criterion: Failure would cause only loss of structure, with little additional damage to property and project operation. (p. 43)

Antecedent soil moisture condition: II (p. 43) (Comparable to average conditions at time of annual flood).

Soil type: volcanic ash (Type B, p. 413)

Vegetation: Juniper, good condition (Table A31)

Runoff curve number: 52

Time Distribution of Design Runoff

Arrange increments of rainfall and snowmelt approximately about midpoint of 6 hour design storm interval.

Time, hours	Incremental rain+melt, inches	Cumulative rain+melt, inches	Cumulative runoff, inches	Incremental runoff, inches
0.0-0.5	0.46	0.46	0.00	0.00
0.5-1.0	0.46	0.92	0.00	0.00
1.0-1.5	0.46	1.38	0.00	0.00
1.5-2.0	0.58	1.96	0.00	0.00
2.0-2.5	1.06	3.02	0.13	0.13
2.5-3.0	1.78	4.80	0.72	0.59
3.0-3.5	0.82	5.62	1.10	0.38
3.5-4.0	0.46	6.08	1.33	0.23
4.0-4.5	0.46	6.54	1.58	0.25
4.5-5.0	0.46	7.00	1.85	0.27
5.0-5.5	0.46	7.46	2.12	0.27
5.5-6.0	0.46	7.92	2.41	0.29

Design Runoff Hydrograph
Time from start of rainfall + snowmelt increment to peak runoff
= 0.5 x increment duration + 0.6 x time of concentration
= 0.5 x 0.5 + 0.6 x 0.2
= 0.37 hours

Time from start of increment to end of runoff = 2.67 x peak time
= 2.67 x 0.37
= 0.99 hours

Duration of falling limb = 0.99 - 0.37 = 0.62 hours.

Incremental runoff at time 0.5 hours after incremental peak runoff
= incremental peak runoff x (0.62-0.50)/0.62
= 0.2 x incremental peak runoff

Incremental peak runoff, cfs =

$$\frac{484 \text{ x area in acres x incremental runoff in inches}}{640 \text{ x time to peak runoff, in hours}}$$

= (484 x 290 x incremental runoff, in.)/(640 x 0.37)
= 593 x incremental runoff, in.

Time interval, hours	Incremtal runoff, inches	Time at peak, hours	Incremental peak runoff, cfs This interval	Previous interval	Total runoff
0.0-0.5	0.00	0.37	0	0	0
0.5-1.0	0.00	0.87	0	0	0
1.0-1.5	0.00	1.37	0	0	0
1.5-2.0	0.00	1.87	0	0	0
2.0-2.5	0.13	2.37	77	0	77
2.5-3.0	0.59	2.87	350	15	365
3.0-3.5	0.38	3.37	225	68	293
3.5-4.0	0.23	3.87	136	44	180
4.0-4.5	0.25	4.37	148	26	174
4.5-5.0	0.27	4.87	160	29	189
5.0-5.5	0.27	5.37	160	31	191
5.5-6.0	0.29	5,87	172	31	203

Peak surface runoff into sedimentation basin = 365 cfs

Next, the corresponding flood flow in Dolly Creek at Walker mine is estimated.

Characteristics of Dolly Creek Watershed at Walker Mine.
(Topographic data obtained from USGS 1:25000 Mt Ingalls quadrangle)
Area drained by Dolly Ck at Walker mine, acres . . 860
Maximum length of watershed, miles, L 2.1
Elevation range of watershed, feet, H 1,900
Time of concentration, hours (Fig. 13) 0.3

Because the time of concentration is similar for the two watersheds and is in both cases short compared to the duration of the design storm, the design flood flow in Dolly Creek may be approximately pro-rated from that in the sedimentation basin watershed on the basis of basin area.

Pro-rated flow in Dolly Creek = 365 x 860 / 290 = 1,080 cfs.

On account of the contortion of Dolly Creek by beaver ponds, it is difficult to estimate what depth of flow in Dolly Creek would result from a flow of 1,080 cfs. Assuming a Manning's channel friction coefficient of n=0.1, and using a channel slope of 7.7%, the depth of flow is likely to be in the range from 2 to 7 ft, assuming effective channel widths of 100 ft and 10 ft respectively. This range of depth of flows appears unlikely to breach the sedimentation basin embankment. Flood flows in Dolly Creek do not appear to be of concern.

Finally, we require the peak flow from the mine adit under conditions producing peak runoff from the watershed tributary to the sedimentation basin, and peak flow in Dolly Creek. The only information available to indicate the magnitude of this flow is the plan area of each of the slumps that may deliver water into collapsed stopes in the mine. Contour maps show five slumps in the watershed of the South Branch of Ward Creek and five in the Middle Branch. Contour maps define the limits of the slumps only approximately, but they may be taken to be roughly elliptical in plan, with major and minor diameters and plan areas listed from north to south in Table 9.

TABLE 9: Plan Dimensions Of Slumps

Watershed	Slump no.	Major diam, ft	Minor diam, ft	Plan area, ac.
South	1	80	70	0.2
	2	200	170	1.0
	3	110	80	0.3
	4	100	80	0.2
	5	250	180	1.3
Middle	1	120	100	0.4
	2	160	150	0.7
	3	170	160	0.8
	4	140	120	0.5
	5	140	130	0.5
Total	-	-	-	5.9

Peak intensity of rainfall plus snowmelt = 1.78 in./30 min
Peak discharge from mine adit, based on 1.78 in. for 30 min over an area of 5.9 acres = 2 x 1.78 x 5.9 = 21 cfs.
(c.f. maximum recorded discharge from mine = 3.3 cfs = 1,480 gpm)

Peak flow into sedimentation basin if diversion ditch is not used
= peak watershed runoff + peak flow from mine
= 365 cfs + 21 cfs = 386 cfs, say 400 cfs.

<u>Sizing Of Diversion Ditch To Carry 365 cfs Surface Runoff:</u>

The hillside behind the mine slopes at about 25%, and 3,200 ft of ditch are needed to divert water around the sedimentation basin. About 2,000 ft of ditch would run westward from the mine portal and 1,200 ft of ditch would run south-east, each of these ditches carrying 50% of the total flow, or 183 cfs. Along the longer and shorter ditches the available losses of elevation from the mine adit to Dolly Creek are 150 ft and 100 ft respectively, corresponding to mean gradients of 7.5% and 8.3%; for the purpose of hydraulic design of the ditches a gradient of 5% will be used, because the available loss of head is not uniform along the length of either ditch. In the vicinity of the mine the average hillside slopes is about 20%, and the ditch cross-section would typically be sloped at up to 1:1 on the backslope and 1:6 on the bench. A ditch freeboard of 3 ft is provided for. Then the following ditch geometric parameters apply in terms of the depth of water in the ditch:

ditch cross-sectional area below waterline = 3.5 x depth^2
wetted perimeter of ditch = 7.5 x depth of water^2
hydraulic radius of flow = 0.47 x depth of water^2
Manning's roughness coefficient is taken as 0.06
(The carat symbol, ^, denotes mathematical exponentiation.)

Manning's equation describes the velocity and capacity as:

```
velocity = 1.5/0.06 x 0.47^(2/3) x depth^(2/3) x slope^(1/2)
         = 15 x depth^(2/3) x slope^(1/2) ft/sec
         = 3.4 x depth^(2/3) ft/sec

capacity = 3.5 x depth^2 x velocity
         = 12 x depth^(8/3) cfs
```

These equations indicate that at the outlet end of each ditch where the flow is 183 cfs the depth of flow is 2.8 ft and the velocity 6.8 ft/sec. At the midpoint of each ditch the depth of flow is 2.2 ft and the velocity 5.7 ft/sec.

With 3 ft of freeboard the depths of each ditch at the upper end, midpoint and lower end are respectively 3.0, 5.2 and 5.8 ft. With a ditch bench slope of 1:6, a backslope of 1:1 max, and a country cross-slope of 1:5 (20%) the cross-sectional area of earthworks is 14 x (ditch depth)^2 sq ft. Then the ditch cross-sections at the upper end, midpoint and lower end are 130, 380 and 470 sq ft.

The total volume of excavation for the two ditches totalling 3200 ft long is 3200 x (130 + 4 x 380 + 470) / 6 / 27 = 42,000 cu yd. Based on 7,000 cu yd per month in soil for a D8 model 2U or similar, this would involve about 1,000 hours of dozer work, at a cost of about $70,000 based on rates charged by Jan Donato. The actual cost is likely to be higher on account of much of the work being in rock rather than soil (the above D8 rating applies only for soil) and because cost rates for dozers used by contractors are often higher than those used by Mr. Donato, who is not a contractor. The existing mine access provides a bench to eliminate some of the excavation otherwise required, but most of the work remains to be done. Rates for dozing in rock vary depending on whether the rock can be moved with a blade, or requires ripping, or requires shooting with explosives. In view of limited knowledge of dozing conditions, the cost of the diversion ditch might be appraised in the $150,000 range.

Hydraulic Design Of Sedimentation Basin Outlet Works:

Although the function of the sedimentation basin outlet works is to decant clarified effluent as the final step in the mine drainage treatment process, the design of these works is dictated entirely by requirements for stability of the embankment retaining the water in the basin, and by the capacity of the outlet works to remove flood flows that enter the sedimentation basin without endangering the stability of the embankment.

Type of Spillway: Considerable time was spent investigating the feasibility of a concrete dam outlet structure, but many difficulties arose with this approach.

First, the fine sand in the tailings area has a lower coefficient of friction against concrete (0.30) than the earthquake coefficient of horizontal acceleration used for the Lake Davis dam (0.42), the latter being considered reasonable for the proposed sedimentation basin dam. Consequently, earthquake forces alone would be sufficient to move a gravity dam relative to its foundation.

Second, the fine sand has a low resistance to piping, such that large cutoff walls beneath and to each side of the dam would be required. Vertical cutoffs would need to be 30 to 35 ft deep beneath the base of a dam retaining 10 ft of water, and horizontal cutoffs would have to extend 90 to 105 ft on each side. Even with some relaxation of these dimensions as suggested for very minor structures, the dam itself was a rather insignificant structure compared to its buried cutoff walls.

Third, ice loads on the dam computed by published procedures substantially increased structural requirements for the dam. Ice loads were obtained from published graphs for the 3 ft thickness of ice reported late in the 1982-83 winter at Lake Davis assuming on a temperature rise rate of 5F/hr. Published ice loads for this ice thickness and temperature rise rate were based on an initial ice temperature of -40F. An attempt to reproduce the published loads by finite different solution of the governing equations was unsuccessful, so it was not possible to adjust published loads to an initial temperature more representative of the mine, say 0F.

It became clear that a drop spillway structure discharging through a concrete pipe penstock excavated through the volcanic ash to the east of the tailings area would be cheaper than the overflow dam-hydraulic jump stilling basin-cutoff wall structure.

Design Considerations For Drop Spillway-Penstock Structure:

Depending on whether a ditch is provided to divert surface runoff from the sedimentation basin, the design flow for the basin outlet structure is 20 or 400 cfs. Penstock pipe sizes suitable to carry these flows are respectively about 2 ft or 6 ft.

The drop spillway structure is located near the point of maximum depth of the sedimentation basin, to minimize upwelling at the basin outlet weir that could cause low settlement efficiency. Water spills over the rim of this open box-like structure to enter the penstock. By making the spillway sufficiently massive to resist flotation, structural design requirements are largely satisfied automatically.

Within the basin the penstock requires weighting with concrete to resist flotation, and through the embankment precautions are needed to avoid piping along the outer wall of the pipe. The system of concrete bedding and concrete collars at intervals along the pipe through the embankment serves to weight the pipe and increase the resistance to seepage along the exterior walls of the penstock. The penstock penetrates natural country of consolidated volcanic ash rather than the tailings sand embankment that is more permeable and more prone to piping. A manhole structure is needed at the point of emergence of the penstock from the volcanic ash ridge, to redirect the penstock downstream 150 ft parallel to the embankment, so that the stilling basin may be properly set at stream level.

Design Of Outlet Structures: Design procedures used herein are described in the U.S.B.R. book "Design of Small Dams", to which page numbers, table numbers, figure numbers and quoted text in the following refers, unless stated otherwise.

Drop Inlet Spillway: S.205, pp 311 et seq, esp. Figs. 222-223.

Spillway hydraulics are expressed in terms of the value of the crest coefficient, C = Q/(6.283 R H^1.5), a function of H/R and P/R, where Q = spillway design flow, cfs; R = radius of circular spillway, ft; H = head on spillway, ft; and P = height of spillway weir above the basin floor, ft. Designs as needed for Q = 20 and 400 cfs and P = 8 and 10 ft are summarized in Table 10..

TABLE 10: Design Parameters For Drop Inlet Spillway

Spillway discharge = 20 cfs			Spillway discharge = 400 cfs		
Spillway radius, ft	Hydraulic head, ft Height = 8 ft	Height = 10 ft	Spillway radius ft	Hydraulic head, ft Height = 8 ft	Height = 10 ft
1.25	1.07	1.07	3.50	6.50	6.53
1.50	0.74	0.74	4.00	3.88	3.89
1.75	0.64	0.63	5.00	2.42	2.42
2.00	0.56	0.56	6.00	2.03	2.03
2.25	0.51	0.51	8.00	1.61	1.61

Table 10 shows there is a tradeoff between spillway radius and the hydraulic head on the spillway. A smaller radiused spillway costs less to construct, but a lower hydraulic head reduces the freeboard required on the dam, measured vertically from the crest of the spillway. For a 20 cfs capacity spillway a 4 ft 6 in. diam concrete pipe on end would be suitable, with a radius to the outside wall of approx 2.5 ft; this size is minimal to provide access for construction and for the maintenance task of removing logs and similar trash that has fallen down the spillway and become stuck.

In the case of a 400 cfs capacity spillway it was considered that the cost of increasing the spillway radius from 5 ft to 6 ft warrants the 0.4 ft reduction in hydraulic head, but the cost of increasing the radius from 6 ft to 8 ft would not; therefore a 6 ft radius spillway was selected. Because a 12 ft diam circular structure would be more expensive to construct than a structure 12 ft square in plan a square spillway was selected, that may produce a hydraulic head as much as 0.1 ft lower than the 2.0 ft tabulated.

Clearing trash from a 400 cfs spillway would be easier than from a 20 cfs spillway because less constriction would mean a lesser accumulation of trash (particularly with a trash rack installed on the spillway), and more room to work. More crucially, with any drop inlet spillway there is a risk of a log jam in the spillway that may lead to catastrophic failure of the dam by overtopping. Overtopping may also result from a rise in basin water level following clogging of the trash rack by debris. The 400 cfs spillway appears unlikely to clog, and is the only alternative presented in detail in this report. Strengthening of the dam to resist overtopping poses problems previously alluded to with respect to design of an overflow dam in this material.

Height Of Drop Inlet Spillway: After pouring over the spillway crest the energy of the water is broken in a pool within the spillway structure that feeds the penstock. The depth of the pool will adjust to the sum of the diameter of the penstock plus the velocity head in the penstock plus the entrance loss into the penstock. In order for the pool not to interfere with the hydraulics of flow over the crest of the spillway, the pool level must not be higher than the level of the crest of the spillway. The limiting condition in which the pool level is at the level of the crest of the spillway fixes the minimum height of a drop inlet spillway.

For a flow of 20 cfs in a 2 ft diam penstock the velocity is 6.4

ft/sec and the velocity head 0.63 ft. Even if the entrance of the penstock projects into the spillway (loss coefficient = 1.0), the pool at the base of the spillway would not be deeper than 3.3 ft. In this case, the 8 - 10 ft height of the spillway structure as needed for adequate performance of the sedimentation basin will also provide sufficient head to get water into the penstock.

For 400 cfs in a 6 ft penstock the velocity is 14.2 ft/sec and the velocity head 3.1 ft. A 12.2 ft deep pool would be needed with a projecting penstock entrance, an 11.3 ft deep pool with a square-cornered entrance, a 10.0 ft deep pool with a rounded entrance (1 ft radius), and a 9.4 ft deep pool with a bellmouth entrance. A rounded entrance was selected as not excessively difficult to construct, but allowing a significantly shallower and less expensive outlet structure than the more simply shaped alternatives. A 10 ft height was selected for the 400 cfs drop inlet spillway.

Design Of Drop Inlet Spillway Against Flotation: For a 20 cfs spillway constructed from an 8 ft long 3 ft ID concrete pipe with a minimum wall thickness of 2 in. the weight is 1,900 lb and the buoyancy 4,350 lb. In order to obtain a factor of safety of 1.2 against flotation the pipe must be secured to a base containing a volume of concrete calculated as (4,350x1.2-1,900)/(150-62.4) = 38 cf = 4 x 4 x 2.4 ft or equivalent. For a 400 cfs spillway with 12 ft OD long walls 10 ft high and 1 ft thick the weight is 66,000 lb and the buoyancy 90,000 lb. The thickness of base needed to provide a safety factor of 1.2 against flotation is given by (90,000x1.2-66,000)/(150-62.4)/144 = 3.3 ft. This spillway contains 35 cu yd of concrete.

Design of Concrete Penstocks Against Flotation: Penstocks are sized 2 ft ID for 20 cfs or 6 ft ID for 400 cfs. Based on S.235 and Appendix C of "Design of Small Dams" concrete bedding and cutoff collars were sized as follows for a 6 ft concrete penstock. The 7 ft OD pipe is bedded on a 10 ft wide 20 in. thick reinforced concrete pad that is poured during or after placing of the penstock pipes. Cutoff collars are 1 ft thick and spaced at 8 ft centers, projecting 2 ft from the pipe walls so as to key into the walls of the 10 ft wide trench. Backfill is hand compacted in layers to above the level of the pipe, thereafter machine compacted in layers. The total submerged weight of pipe, bedding and cutoffs of 2,660 lb/ft resists the buoyancy of the pipe interior of 1,760 lb/ft.

Manhole to Change Direction of Penstock: This structure is similar to the spillway except that the top of the box is roofed with a concrete slab, and the walls are slightly higher, extending 12 ft from the base to the level of the crest of the dam. One velocity head is allowed for hydraulic losses, i.e. 0.6 ft for a manhole joining 2 ft pipe segments carrying 20 cfs, or 3 ft for 400 cfs in 6 ft pipes. A manhole cover and step irons provide access to the interior. A standard manhole is used with a 20 cfs outlet. For for the 400 cfs outlet a concrete box is constructed, 10 ft square in plan and 11 ft high inside, with 1 ft thick walls and a 1.25 ft thick floor. Structural design would be so as to resist a 10 ft deep exterior loading of saturated soil equivalent to a 130 lb/cu ft fluid. The 400 cfs manhole requires 32 cu yd of concrete.

Impact Stilling Basin: Ref. S.202, pp. 305-307. The force needed to break the momentum of 400 cfs of water in a 6 ft pipe is 1.94 x 400 x 14.1 = 11,000 lb, a load carried by the baffle in the impact stilling basin structure. From the structure, the force of water is transmitted to the subsoil by a key in the base of the basin that provides 40 sq ft of bearing area against the native country at a loading of 275 lb/sq ft. Standard design graphs show that a 400 cfs stilling basin is 20 ft x 26 ft in plan with a 15.5 ft high rear wall, the entire structure containing 67 cu yd of concrete. Design graphs for impact stilling basins show that key dimensions vary with the 0.4 power of hydraulic capacity, so that a 20 cfs unit would contain approx 20 cu yd of concrete. An 18 in. thick layer of riprap on the channel downstream of the stilling basin is specified, approx. 1,200 sq ft for the 400 cfs basin or 120 sq ft for the 20 cfs basin.

Preliminary Estimates Of Cost For Treatment Structures:

Figures 8-11 are preliminary engineering sketches of the chemical neutralization plant, spillway, manhole and stilling basin. Tables 11-16 are itemized preliminary estimates of cost for construction of the limestone barrier, chemical neutralization plant, spillway, penstock, manhole and stilling basin.

TABLE 11: Preliminary Cost Estimate For Limestone Barrier.

Component	Item	Unit	Quantity	Rate	Amount
channel	excavation	cu yd	366	4.00	1466
limestone	1-1/2" size	ton	450	50.00	22500
	place stone	ton	450	30.00	13500
					37466

TABLE 12: Preliminary Cost Estimate For Chemical Neutralization Plant.

Component	Item	Unit	Quantity	Rate	Amount
sitework	excavation	cu yd	782	4.00	3129
	subbase	sq yd	160	3.00	480
	backfill	cu yd	497	3.00	1491
	resurface	sq yd	171	3.00	513
north wall	formwork	sq ft	1189	8.00	9512
	rebar	lb	3631	0.55	1997
	concrete	cu yd	24.21	140	3389
	screeding	sq ft	12	0.70	8
	trowelling	sq ft	12	2.00	24
east wall	formwork	sq ft	952	8.00	7620
	rebar	lb	2639	0.55	1452
	concrete	cu yd	17.59	140	2464
south wall	formwork	sq ft	1355	8.00	10845
	rebar	lb	3567	0.55	1962
	concrete	cu yd	23.78	140	3329
	screeding	sq ft	12	0.70	8
west wall	formwork	sq ft	988	8.00	7911
	rebar	lb	2746	0.55	1511
	concrete	cu yd	18.31	140	2564
	screeding	sq ft	22	0.70	15
floor	rebar	lb	5226	0.55	2874
	concrete	cu yd	34.84	140	4878
	screeding	sq ft	636	0.70	445
	trowelling	sq ft	240	2.00	480
roof	formwork	sq ft	454	8.00	3632
	rebar	lb	2590	0.55	1425
	concrete	cu yd	17.27	140	2418
	screeding	sq ft	534	0.70	374
divider	formwork	sq ft	900	8.00	7200
	rebar	lb	2500	0.55	1375
	concrete	cu yd	16.66	140	2333
stairs	formwork	sq ft	69	8.00	560
	rebar	lb	190	0.55	105
	concrete	cu yd	1.26	140	178
	screeding	sq ft	48	0.70	34
	trowelling	sq ft	48	2.00	96
	nosing	lin ft	42	12.00	504
	handrails	lin ft	14	18.00	252
channel	formwork	sq ft	525	8.00	4204
	rebar	lb	1197	0.55	658
	concrete	cu yd	7.98	140	1117
	screeding	sq ft	128	0.70	90
	trowelling	sq ft	128	2.00	256
	6"C.I.pipe	lin ft	5	15.00	75
	PVC pipes	each	5	35.00	175
column	formwork	sq ft	174	8.00	1392
	rebar	lb	533	0.55	293
	concrete	cu yd	3.55	140	498

bar screen	galv steel	lb	4406	3.50	15422
roof grill	galv steel	sq ft	116	7.00	812
floorgrill	galv steel	sq ft	105	7.00	735
door	mild steel	lb	802	2.50	2006
waterwheel	mild steel	lb	1946	2.50	4867
wheelframe	mild steel	lb	474	2.50	1185
chem pump	peristaltic	each	6	450	2700
manhole	cast iron	each	1	700	700
step irons	SS, chem tank	each	27	30.00	810
pipe	SS, 2 in.	lin ft	130	18.00	2340
valve	SS, 2 in.	each	7	130	910
bend,90deg	SS, 2 in.	each	8	30.00	240
pipe	SS,1/2 in.	lin ft	15	5.00	75
valve	SS,1/2 in.	each	10	40.00	400
hose,chem	armored,5'	each	12	20.00	240
pipe,drain	PVC,12 in.	lin ft	120	40.00	4800
coupling	PVC,12 in.	each	5	150	750
gantry	1/2ton cap	each	2	220	440
Total					137611

TABLE 13: Prelininary Cost Estimate For Spillway.

Component	Item	Unit	Quantity	Rate	Amount
sitework	excavatn	cu yd	53	4.00	215
	subbase	sq yd	53	3.00	161
north wall	formwork	sq ft	260	8.00	2080
	rebar	lb	618	0.55	339
	concrete	cu yd	4.12	140	576
	screeding	sq ft	11	0.70	7
east wall	formwork	sq ft	287	8.00	2296
	bellmouth	sq ft	35	40.00	1401
	rebar	lb	750	0.55	412
	concrete	cu yd	5.00	140	700
	screeding	sq ft	17	0.70	12
south wall	formwork	sq ft	260	8.00	2080
	rebar	lb	618	0.55	339
	concrete	cu yd	4.12	140	576
	screeding	sq ft	11	0.70	7
west wall	formwork	sq ft	260	8.00	2080
	rebar	lb	618	0.55	339
	concrete	cu yd	4.12	140	576
	screeding	sq ft	11	0.70	7
floor	rebar	lb	800	0.55	440
	concrete	cu yd	17.76	140	2486
	screeding	sq ft	81	0.70	56
trash rack	supports	lb	510	2.20	1122
	rails	lb	1632	2.20	3590
	bars	lb	3264	2.20	7180
pipework	6"d. short	ea	2	80.00	160
	6"d. valve	ea	2	300.00	600
	12"d short	ea	1	150.00	150
	12" flange	ea	1	90.00	90
Total					30091

TABLE 14: Preliminary Cost Estimate For Penstock.

Component	Item	Unit	Quantity	Rate	Amount
sitework	excavatn	cu yd	1277	4.00	5111
	subbase	sq yd	333	3.00	999
	backfill	cu yd	626	25.00	15655
	resurface	sq yd	333	3.00	999
RC pipe	72", laid	lin ft	300	130	39000
conc bedding	rebar	lb	8561	0.55	4709
	concrete	cu yd	147.71	140	20679
	screeding	sq ft	1530	0.70	1071
cutoffs	formwork	sq ft	4116	8.00	32932
	rebar	lb	4418	0.55	2430
	concrete	cu yd	76.23	140	10672
	screeding	sq ft	375	0.70	262
					134525

TABLE 15: Preliminary Cost Estimate For Manhole.

Component	Item	Unit	Quantity	Rate	Amount
sitework	excavation	cu yd	254	4.00	1016
	subbase	sq yd	53	3.00	161
	backfill	cu yd	178	3.00	535
	resurface	sq yd	37	3.00	113
north wall	formwork	sq ft	280	8.00	2240
	rebar	lb	782	0.55	430
	concrete	cu yd	5.21	140	730
east wall	formwork	sq ft	280	8.00	2240
	rebar	lb	782	0.55	430
	concrete	cu yd	5.21	140	730
south wall	formwork	sq ft	307	8.00	2456
	bellmouth	sq ft	35	40.00	1401
	rebar	lb	887	0.55	488
	concrete	cu yd	6.12	140	857
	screeding	sq ft	7	0.70	5
west wall	formwork	sq ft	307	8.00	2456
	rebar	lb	918	0.55	505
	concrete	cu yd	6.12	140	857
	screeding	sq ft	7	0.70	5
floor	rebar	lb	1470	0.55	808
	concrete	cu yd	9.80	140	1372
	screeding	sq ft	81	0.70	56
roof	formwork	sq ft	100	8.00	800
	rebar	lb	677	0.55	372
	concrete	cu yd	4.51	140	632
	screeding	sq ft	144	0.70	100
	trowelling	sq ft	144	2.00	288
manhole	frame/cover	each	1	700	700
	stepirons	each	14	15.00	210
Total					23002

TABLE 16: Preliminary Cost Estimate For Stilling Basin.

Component	Item	Unit	Quantity	Rate	Amount
sitework	excavation	cu yd	576	4.00	2304
	subbase	sq yd	144	3.00	432
	backfill	cu yd	364	3.00	1092
	resurface	sq yd	86	3.00	258
north wall	formwork	sq ft	518	8.00	4144
	rebar	lb	1662	0.55	914
	concrete	cu yd	11.08	140	1552
east wall	formwork	sq ft	595	8.00	4765
	rebar	lb	1677	0.55	922
	concrete	cu yd	11.18	140	1565
	screeding	sq ft	15	0.70	10
	trowelling	sq ft	15	2.00	30
south wall	formwork	sq ft	375	8.00	3000
	rebar	lb	1061	0.55	583
	concrete	cu yd	7.07	140	990
	screeding	sq ft	32	0.70	22
	trowelling	sq ft	32	2.00	64
west wall	formwork	sq ft	595	8.00	4765
	rebar	lb	1677	0.55	922
	concrete	cu yd	11.18	140	1565
	screeding	sq ft	15	0.70	10
	trowelling	sq ft	15	2.00	30
floor	rebar	lb	2769	0.55	1523
	concrete	cu yd	18.46	140	2584
	screeding	sq ft	391	0.70	273
splashguard	formwork	sq ft	68	8.00	551
	rebar	lb	250	0.55	137
	concrete	cu yd	1.67	140	233
	screeding	sq ft	90	0.70	63
	trowelling	sq ft	90	2.00	180
baffle	formwork	sq ft	357	8.00	2861
	rebar	lb	961	0.55	528
	concrete	cu yd	6.41	140	897
	screeding	sq ft	72	0.70	50
outfall	excavation	cu yd	325	4.00	1300
	bedding	sq yd	130	15.00	1954
	riprap	cu yd	60	90.00	5477
Total					48567

Table 17 is a preliminary estimate of the total cost of construction of the Walker mine drainage treatment facility, based as for the above cost estimates on assumed concrete wall thicknesses and other items fixed by structural design.

TABLE 17: Preliminary Estimate Of Total Cost Of Construction.

Limestone barrier	37,466
Neutralization plant	137,611
Spillway	30,091
Penstock	134,525
Manhole	23,002
Stilling basin	48,567
Sedimentation basin and sludge lagoon earthworks	30,000
Contractor's establishment, supervision, O&P (20%)	88,252
Total	529,514
	say $530,000

The precision with which cost estimates are presented is not intended to suggest the degree of accuracy of the preliminary cost estimates. Conventional methods of costing construction based on unit rates prevailing in cities have less validity to the costing of construction in remote areas such as Walker mine.

Local conditions such as the remoteness of the site, the lack of electric power, the loss of winter access and perhaps the infeasibility of inspecting the site prior to bidding may lead to a wide range of bids. Prospective contractors' evaluations of these factors properly become matters of judgement rather than being ruled by the arithmetic of the schedule of quantities and lists of standard unit rates. The schedule of unit rates and costs that contractors are required to complete and submit with their bids may poorly represent the way they view the job.

Operating and Maintenance Supplies and Equipment: In addition to the treatment plant itself, certain supplies and equipment are needed for operation. A partial list of these items follows:

Records	Stationery and supplies for keeping records of work, including time cards, O&M records, reports of accidents and other events, and required reporting forms.
Safety	Protective clothing and safety and emergency equipment, first aid supplies, and CB radio.
Tools	Hand tools including shovels, brooms, picks, ladders; mechanics, plumbers, carpenters and gas welding and cutting equipment; ropes, winches and tackle for clearing trash and manipulating mobile installed equipment; lubrication equipment; supplies and spare parts.
Machines	Electrical generator for charging radio batteries and use with hand power tools; chemical pump (gas powered) to pump concentrated caustic soda from tanker to chemical storage tank and for mixing with dilution water in storage tank; sludge pump (gas powered) for desludging sedimentation basin.
Analyses	Rudimentary analytical test equipment and supplies, including pH paper, Imhoff cones and colorimetric test kit for copper.
Building	Office and lab space, lunch room, first aid station, restroom and storage facility.
Vehicles	Pickup truck (or arrangements to use private vehicle), four wheel drive; small boat for working in sedimentation basin.
Chemical	Stock of neutralization chemical, renewed annually.

Labor Requirements For Operation and Maintenance of Plant:

Table 18 estimates the annual number of manhours for operation and maintenance of the Walker plant as 3,926. Based on an average of 1,500 effective hours per person, the number of people needed to run the plant is 3,926/1,500 = 2.6. If due to loss of winter

TABLE 18

Staff Estimate For Municipal Wastewater Treatment Facility
According to EPA O&M Procedure Contract 68-01-0328.
Walker Mine Neutralization/Sedimentation Plant
Design flow 2.2 mgd.

I. Local Conditions for Which Estimate Was Produced:

Plant layout - extended
Process - non-std. equipment; different manufacturers
Level of treatment - primary
Removal requirements - amount of waste in effluent
Industrial waste - none or constant
Labor productivity - low
Climate - moderate winters
Training - neither certification nor cont. education
Automatic monitoring - none
Automatic sampling - none
Off-plant lab work - none
Off-plant maintenance - none
Age and condition of equipment - new and/or well cared for
Staffing pattern - smaller than normal night/weekend staff

II. Annual Operating and Maintenance Manhours By Process:

	Operation	Maintenance
Screening or grinding..................	150	30
Gravity thickening.....................	177	227
Sludge drying beds.....................	371	0
Lime or ferric coag. and settling.......	804	294

III. Annual Manhours By Labor Category:

Operation..............................	1,502 hours/year
Maintenance............................	551 hours/year
Supervision............................	499 hours/year
Clerical...............................	65 hours/year
Laboratory.............................	330 hours/year
Yardwork...............................	978 hours/year
TOTAL..................................	3,926 HOURS/YEAR

access to the plant it is only possible to work there for six months each year, then twice that many people are needed for half a year, or 5.2 persons.

The basis of Table 18 is a procedure that was prepared for the purpose of estimating operating and maintenance manpower requirements for municipal wastewater treatment plants. Municipal wastewater treatment processes that most closely resembled the processes employed in the Walker mine drainage treatment plant were evaluated for manpower requirements, and these estimates were assumed to apply for the Walker plant.

However, certain features of the Walker plant are likely to require more labor than would similar processes in a municipal wastewater treatment plant. In municipal waste treatment plants single purpose items of installed electric-powered equipment make for greater convenience than portable gasoline-powered equipment. At Walker the task of delivering neutralization chemicals to the storage tank and dissolving or diluting these chemicals may be laborious, because delivery vehicles may not be able to approach closer than about 500 ft to the storage tank, and it is difficult to adapt conventional mixing equipment to the Walker situation. Finally, the task of removing sludge from the Walker sedimentation basin, and drying and transporting this sludge to the disposal site is likely to require considerably more labor than the largely automated operations at municipal wastewater treatment plants. A five person summer season operating and maintenance team may be quite busy.

CALIFOR! REGIONAL WATER QUALITY CONTRO OARD
CENTRAL VALLEY REGION

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ORDER NO. 83-148

REQUEST TO ABATE POLLUTION
FROM
WALKER MINE, ROBERT R. BARRY, AND CALICOPIA CORPORATION
PLUMAS COUNTY

The California Regional Water Quality Control Board, Central Valley Region, (hereafter Board) finds that:

1. A condition of pollution exists which has resulted from a nonoperating copper mine owned by Robert R. Barry and Calicopia Corporation (hereafter Discharger) in central Plumas County, about twenty miles (32 km) east of Quincy, in Sections 19, 29, 30, 31, and 32, T25N, R12E, and Sections 5, 6, 7, and 8, T24N, R12E, MDB&M, and situated within the jurisdiction of this Board.

2. The Board, on 30 May 1980, adopted Waste Discharge Requirements Order No. 80-058, NPDES No. CA0080110; Order No. 80-071, Referral to the Attorney General; and Order No. 80-070, Cleanup and Abatement Order against the Discharger.

3. The Discharger has violated and continues to violate Waste Discharge Requirements established in Order No. 80-058 and Cleanup and Abatement Order No. 80-070. No significant progress has been made by the Discharger towards reduction of the toxic acid mine drainage. The discharge flows to Dolly Creek, tributary to Little Grizzly Creek, which is tributary to Indian Creek, thence the East Branch North Fork Feather River, waters of the United States.

4. Unless certain abatement measures are initiated, the mine will continue to discharge acid water containing metals toxic to fish and other aquatic life.

5. Measures to abate the toxic discharges include either sealing the mine tunnel or providing treatment such as a limestone barrier, neutralization plant, and sedimentation basin.

6. Pursuant to Section 13305 of the California Water Code, the Board may "...request the city, county, or other public agency in which the conditions of pollution...exists to abate it." "The owner of the property on which the conditions exist...is liable for all reasonable costs incurred ...in abating the condition." And, "...the cost for abating the condition...shall constitute a lien upon the property...upon recordation,..."

7. Issuance of this Order is exempt from the provisions of the California Environmental Quality Act in accordance with Section 15321(a)(2), Title 14, Chapter 3, California Administrative Code.

8. On 9 December 1983, in Sacramento, after due notice of Finding No. 1 to the Discharger and all affected persons, in accordance with Section 13305, California Water Code, the Board conducted a public hearing and considered all objections and protests to the proposed correction of the condition.

IT IS HEREBY ORDERED, that pursuant to Section 13305 of the California Water Code:

1. The Board requests Plumas County, the U.S. Forest Service, and all other appropriate public agencies to abate the condition of pollution resulting from the nonoperating mine owned by Robert R. Barry and Calicopia Corporation.

2. In the event that the agencies listed above do not abate the condition of pollution resulting from the Walker Mine within a reasonable time, the Board shall take all steps necessary to abate the condition.

3. In the event that Robert R. Barry and Calicopia Corporation present to the Board a plan for abatement of the condition of pollution on or before 1 February 1984, the Board shall evaluate such plan prior to conducting any abatement work at the mine site.

I, WILLIAM H. CROOKS, Executive Officer, do hereby certify the foregoing is a full, true, and correct copy of an Order adopted by the California Regional Water Quality Control Board, Central Valley Region, on 9 December 1983.

William H. Crooks

WILLIAM H. CROOKS, Executive Officer

Amended 12/9/83

RECORD OF DECISION

FOR REMEDIATION OF THE WALKER MINE TAILINGS

BECKWOURTH RANGER DISTRICT, PLUMAS NATIONAL FOREST

April, 1994

RECORD OF DECISION

FOR REMEDIATION OF THE WALKER MINE TAILINGS

PLUMAS NATIONAL FOREST

PREPARED BY:

TERRY A. BENOIT
Forest Hydrologist

Date

RECOMMENDED BY:

JEFF WITHROE
Acting District Ranger
Beckwourth Ranger District

Date

RECOMMENDED BY:

H. WAYNE THORNTON
Forest Supervisor

Date

APPROVED BY:

MELROY H. TEIGEN
Acting Director, Engineering
Pacific Southwest Region

Date

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WALKER MINE TAILINGS RECORD OF DECISION

Figure 3 - Walker Mine Tailings Site Map

APPENDIX

Summary of Significant Comments Received During the Public Comment Period

DECLARATION FOR THE RECORD OF DECISION

WALKER MINE TAILINGS

PLUMAS NATIONAL FOREST

PLUMAS COUNTY, CALIFORNIA

Site Name and Location

The Walker Mine Tailings are located on National Forest System land approximately 15 miles east of the Plumas County community of Quincy in Section 12, T24N, R11E; and Sections 7 and 18, T24N, R12E; Mt. Diablo Baseline and Meridian. The 100-acre tailings area is downstream of Walker Mine, which is located on patented land, and at the confluence of Dolly Creek and Little Grizzly Creek.

Statement of Basis and Purpose

This decision document represents the selected remedial action for the treatment of the Walker Mine Tailings area developed in accordance with the Comprehensive Environmental Response, Compensation, and Liability Act of 1980 (CERCLA), as amended, and the National Contingency Plan (NCP). This decision is based on the administrative record for this site. The State of California, Plumas County, most of the public, and Atlantic Richfield Company (a Potential Responsible Party) is in agreement with the selected remedy.

Assessment of the Site

Actual or threatened release of hazardous substances from the site, if not addressed by implementing the response action selected in this Record of Decision (ROD), will present an imminent and substantial endangerment to public health, welfare or the environment.

Description of the Selected Remedy

This ROD for the Walker Mine Tailings includes the following actions to address existing and future contamination:

- Treat the tailings material on-site. Removal of all or part of the material is not proposed.

- Reconstruct 1500 feet of Dolly Creek channel to a stable geometry and revegetate its banks, including the larger gully banks.

- Construct a 15-acre passive water treatment system (wetland) in the lower portion of Dolly Creek. This will include raising the sediment dam approximately one foot.

- Construct wind barriers on 50 acres of the tailings surface.

- Neutralize 10 acres of low pH material with crushed limestone.

- Revegetate 60 acres of tailings area with grasses, shrubs, and trees.

- Close the site to public access where needed to protect treatment features.

- Monitor for success and compliance with Applicable, Relevant and Appropriate Requirements (ARARs).

Declaration

The selected remedy is protective of human health and the environment, meets Federal and State requirements that are applicable, relevant and appropriate to this remedial action and is cost-effective. The remedy satisfies the statutory preferences for remedies that employ treatment that reduces toxicity, mobility or volume as a principal element and utilizes permanent solutions to the maximum extent practicable. The remedy meets requirements provided by the State of California.

MELROY H. TEIGEN
Acting Director, Engineering
Pacific Southwest Region

Date

DECISION SUMMARY

I. *Site Name and Location*

The Walker Mine Tailings are located on National Forest land approximately 15 miles east of the Plumas County community of Quincy in Section 12, T24N, R11E; and Sections 7 and 18, T24N, R12E; Mt. Diablo Baseline and Meridian (Figure 1).

At an elevation of 5750 feet mean sea level, the tailings area is at the confluence of Dolly Creek and Little Grizzly Creek, tributary to Indian Creek, then the East Branch North Fork Feather River. Dolly Creek flows from northeast to southwest from near the Walker Mine and across the tailings area. Little Grizzly Creek flows along the southern edge of the tailings area from southeast to northwest (Figures 2 and 3).

II. *Site Description, History and CERCLA Response Actions*

The Walker Mine, located on patented lands, produced copper and minor quantities of gold and silver from 1915 through 1941. The 1941 operation was shut down and has since remained idle except for occasional exploration activities.

The tailings area is located in a natural basin three-quarters of a mile southwest and downstream of the Walker Mine on Dolly Creek at its confluence with Little Grizzly Creek. The tailings were produced as a slurry at the mill located at the mine site. This slurry flowed by gravity to the tailings site where it was impounded by a small dam on Dolly Creek. Much of the free water from the milling process evaporated, leaving behind a well distributed pile of fine-grained, sandy, silty, and clay-like tailings material covering an area of approximately 100 acres to an average depth of 28 feet (based on borings made in 1992).

The Walker Mine has a long history of pollution, acid mine drainage, heavy metals contamination, and noncompliance with Waste Discharge Requirements (WDRs) established by the California Regional Water Quality Control Board, Central Valley Region (CVRWQCB). In 1987, the CVRWQCB retained an engineering contractor to design and install a concrete seal in the mine tunnel to minimize acid mine drainage and discharge of heavy metals into waters from the mine. The seal appears to be effective in reducing mine discharge into the nearest receiving water, Dolly Creek, then Little Grizzly Creek. See Figure 2 for a summary of the current affectiveness of the mine seal.

The Walker Mine Tailings also adversely affect the water quality of Dolly Creek and Little Grizzly Creek. Dolly Creek, and any remaining drainage from the Walker Mine, flow from northwest to southwest along the northern portion of the tailings, picking up leachate water and resulting in release of tailings, heavy metals (copper, iron, and zinc), and turbid water to the receiving waters. In 1958 the CVRWQCB adopted Resolution No. 58-181 prescribing discharge requirements for the tailings, and named the USFS and the owners of the Walker Mine as the dischargers. In 1986 the CVRWQCB rescinded Resolution No. 58-181 and issued WDRs Order No. 86-073, naming the USFS as the sole discharger. New WDRs were issued in 1991 and Resolution No. 91-017 was adopted. Maximum receiving water quality criteria for the compliance station on Little Grizzly Creek, downstream of the Walker Mine Tailings were established.

The Walker Mine tailings site was placed on the Federal Agency Hazardous Waste Compliance Docket ("the docket"), pursuant to the Comprehensive Environmental Response, Compensation and Liability Act (CERC-LA, 42 USC 9620 (c)) by the U.S. Environmental Protection Agency (EPA) in 1991.

A site investigation was started in 1990 and completed in 1992 with the installation of monitoring wells and a waste characterization program. the 1990-1991 investigation focused on the release and transport of copper and sediment from the tailings and the development of alternatives for stabilizing and reclaiming the tailings area. Included in the study was an investigation and preliminary assessment of health risks to forest users and workers at the site.

Other contamination pathways, such as groundwater, were studied and determined to be insignificant or non-existent, although questions still remain because of increased concentrations of copper detected in Little Grizzly Creek between the confluence with Dolly Creek and the Brown's Cabin monitoring site.

III. *Community Relations*

Community relations were initiated in 1989 when the East Branch North Fork Feather River Coordinated Resource Management group (EBNFFR CRM) added the treatment of the Walker Mine Tailings into their water quality improvement program. The EBNFFR CRM is a formal partnership that includes 19 local, state and federal agencies plus private land owners and the Pacific Gas and Electric Company. The primary goal of the EBNFFR CRM is water quality improvement in the East Branch North Fork Feather River.

A formal public involvement plan was initiated in September 1991, to facilitate public involvement with the proposed project. The public includes the EBNFFR CRM, local, State and Federal agencies, interested and affected individuals and groups, and Potential Responsible Parties (PRPs). Communications included direct mailings, newspaper notices, news releases, and public meetings. Interested parties also became informed and involved through personal communications.

Public support for the project has been positive, except for a few people who use the site as a "playground" with their off-highway vehicles (OHV). Support from the various government agencies has also been positive.

The primary support agency has been the CVRWQCB. United States Forest Service (USFS) personnel and water quality engineers for the State agency have worked closely to analyze the site and develop treatment alternatives.

The PRPs have been identified and requested to participate in the planning process. Little response has been received until lately, when the Atlantic Richfield Company (ARCO) was identified in 1993. ARCO responded immediately and positively (See Appendix).

Copies of all relevant documents have been sent to interested parties, the CVRWQCB, and PRPs. Comments on the draft documents were solicited. The Proposed Plan for remediation of the site was also handled in this way.

Very little public interest has been demonstrated. Homeowners in Genessee Valley, downstream from the tailings area have informally expressed their support of the proposed treatments, as have other interested parties. Recreation users of the site, as mentioned above, have informally expressed their desire to leave the site as it is and allow them to continue to use the area for off-highway vehicle use.

Mr. Leroy Pedersen of Four Hills Mining Company has made numerous contacts with the USFS regarding the treatment of the tailings material. He is working with a patented process to treat tailings material containing high amounts of silica, removing the metals and the silica. Further testing of the process is required before it can be evaluated and approved for use. If this or any process is found to be a desirable remedy for the site in the future, there is nothing in the proposed treatment that will preclude their use and effectiveness.

No response has been received from Mr. Henry R. Barry, CEO, Calicopia Corporation, owners of the Walker Mine and a Potential Responsible Party (PRP) for remediation of the tailings area. The latest mailing to Mr. Barry resulted in a return mailing and no forwarding address. Efforts to locate him suggest that he is no longer in the country and that Calicopia Corporation no longer exists.

Three PRPs hold mining claims on the tailings area. No work has been performed by them, except for a minimal amount of exploratory work. Contact was made with one of the claimants, Mr. Archie Sparkman, who spoke for all of the claimants. They would like to dissolve all interest in the site. They have not paid taxes on the claim for the past three years. Mr. Sparkman said they fully support the treatments that are proposed for the site.

Recently, another PRP has surfaced as a result of research conducted for the USFS by TechLaw Inc. TechLaw has established a fairly solid link between the Walker Mining Company and Anaconda Company. Additionally, TechLaw has substantiated Anaconda Company's relationship to Atlantic Richfield Company (ARCO). The USFS notified ARCO of their potential responsibility and received a positive response with a willingness to participate in remediation efforts to the limit of their liabilities, which still needs to be determined. They have also responded in support of the proposed treatments for the site, stating that they believe them to be very practical and reasonable.

IV. *Site Characteristics*

Where Dolly Creek flows across the tailings area, the upper channel section has incised 20 feet through the tailings material to native soil. It is here where most of the sediment enters Dolly Creek for transport downstream. Water is the primary agent eroding the tailings material to the streams, although wind drives a significant amount of tailings material from the surface of the tailings to the gully banks, where it is then picked up by flowing water. Below this incised section, Dolly Creek becomes braided and is dominated by alluviation and continuous bed movement. Some natural wetland development is occurring in this area. The base level is controlled by a sediment retention dam constructed originally by the operators of the Walker Mine and then reconstructed in 1979 by the USFS.

The beneficial uses of the water from Dolly Creek and Little Grizzly Creek are:

1. Agricultural water supply.
2. Recreation.
3. Aesthetic enjoyment.
4. Preservation and enhancement of fish, wildlife, and other aquatic resources.

Downstream beneficial uses of the Feather River include:

1. Municipal water supply.
2. Industrial water supply.
3. Ground water recharge.
4. Hydroelectric power generation.

The mean annual precipitation for the area is about 40 inches, with a significant portion in the form of snow. The mean minimum temperatures at the site for the months of January and July are 16 degrees Fahrenheit and 42 degrees Fahrenheit, respectively. Surface runoff usually results from snowmelt, but fall and spring rains and summer thundershowers are also common.

Vegetation in the vicinity of the mine and tailings area consists largely of mixed conifer forest. The tailings area is mostly nonvegetated but does support locally vegetated areas containing rushes in low-lying areas, islands of pines and shrubs, and islands of sedges along Dolly Creek. Because of this general lack of vegetation, moisture levels in the tailings material rarely drops below field capacity even during the summer months. Only the top three to six inches completely dries out.

Groundwater in the surrounding area is found in seasonal shallow or perched aquifers (decomposed granite) mantling bedrock surfaces or fractured-rock aquifers formed by the interconnected joints and fractures in the bedrock. Ground water in the tailings area is controlled primarily by the elevation of the sediment dam, but does reflect moisture conditions during winter and summer months. During the monitoring well installation in October, 1992, water elevations averaged 5.73 feet below the surface of the tailings material, ranging from 0.40 feet to 17.23 feet below the tailings surface.

The tailings aquifer is recharged by snow and rain falling onto the tailings area, by several springs surrounding the site and possibly buried by the tailings material, by conveyance along the original Little Grizzly Creek channel (now buried by tailings material), and directly by Dolly Creek as it flows across the tailings area. Discharge occurs by evaporation from the surface, by seepage along the base of the levee separating Little Grizzly Creek from the tailings material, by surface and seepage flow over and through the sediment retention dam, and, possibly, by seepage through rock fractures and the original Little Grizzly Creek channel.

V. *Risk Assessment Summary*

Copper, iron and zinc are continually released into Dolly Creek and Little Grizzly Creek through a variety of pathways, exposing aquatic organisms to lethal or otherwise stressful concentrations of these metals. These organisms have been shown to be either killed outright or their life cycles affected to such a degree that they cannot maintain viable and productive populations. Approximately 3800 feet of Dolly Creek and about one mile of Little Grizzly Creek are affected by the contaminants released from the tailings. Within that one-mile section of Little Grizzly Creek, dilution and biological uptake reduce contaminant concentrations to near background levels.

Human health is potentially affected when dust emanating from the tailings area is inhaled. The respirable free silica is crystalline in form and can cause silicosis and lung cancer, especially under occupational exposure for several years. Concentrations of metals in the tailings material known to cause human health problems have been identified, but are at levels in the surface material that is indistinguishable from soils at background sites. Table 1 displays metals found in the tailings material at well sites and bore holes. Table 2 displays metals released into the waters of Dolly Creek (Station R1, above the tails; and Station R2, below the tails) and Little Grizzly Creek (Station R3, above the tails; Station R4 below the tails; and Station R5, the compliance station located below the confluence with Dolly Creek). Station R6 is an overflow pipe located near the middle of the tailings area and next to Little Grizzly Creek. Refer to Figure 4.

Metals found in the tailings material, but not released into the environment in amounts detrimental to human health or the environment include:

Arsenic	Barium	Cobalt	Chromium	
Iron		Lead	Mercury	Nickel
Silver	Thorium		Vanadium	

The primary land and resource uses in the area include timber harvesting, mining and recreation. Downstream uses include recreation, fishing, and irrigation of pasture land at the mouth of Little Grizzly Creek. There are no known diversions of water for domestic purposes.

Human exposure to dust is limited to recreational use of the site and to workers in and around the site. Recreation on the site is primarily OHV use. This activity causes large amounts of the tailings material to become airborne, especially where these vehicles are concentrated. Wind also causes large amounts of the tailings material to become airborne, often making it difficult to see and breath.

In addition, wind erosion affects the surface of the tailings area on a daily basis during the growing season. Plants emerging on the site are sheared, buried, or eroded away. The lack of nutrients for plant growth makes it difficult for all but the hardiest plants, usually pioneering varieties, to emerge in the first place.

Towards the end of the mining and milling operations at Walker Mine, ore was incompletely processed then discharged into Dolly Creek to flow freely downstream to the tailings dump. The areas of the tailings covered by this material are distinctly different from the rest of the tailings area. These areas are limiting plant growth due to acidic conditions, increased solubility of metal ions, elevated levels of iron, and deficiency of sulphur, calcium, and molybdenum. Molybdenum is required by many pioneer species, especially legumes which typically will not grow without sufficient molybdenum for nitrogen fixation.

Most of the tailings material is affected by a lack of similar nutrient chemistry. This includes both macronutrients (nitrogen, phosphorous, potassium, sulfur, calcium, and magnesium) and micronutrients (manganese, boron, and molybdenum). There is a general low level of nitrogen, phosphorous, iron, and molybdenum. The obvious lack of organic matter, necessary for cation exchange, limits the uptake of nutrients. For the purposes of plant growth, all of the tailings are deficient in all of the major plant nutrient cations (potassium, calcium, and magnesium).

Since treatment of the tailings is proposed on-site and none of it removed, there is a risk that treatments may not be fully successful and release of contaminants could continue above levels described in section VII, Remedial Action Goals and Objectives.

VI. *Applicable or Relevant and Appropriate Requirements (ARARs) Analysis*

Any alternative should comply with applicable or relevant and appropriate requirements (ARARs). The Environmental Protection Agency (EPA) determined that this site does not warrant placement on the National Priorities List (NPL) by evaluating its hazards and vicinity to human habitations. As a consequence, the site falls under the jurisdiction of California's Environmental Protection Agency and their mandated clean-up standards.

Requirements applicable or relevant and appropriate to the site have been identified through formal communication and consultation with the California State Attorney General, and the CVRWQCB, plus other relevant State and local agencies. None of the ARARs listed have been waived.

Identified ARARs are as follows:

1. **State Water Board Resolution 68-16 (anti-degradation policy):**

This resolution satisfies the Federal Clean Water Act anti-degradation policy requirement.

It requires the continued maintenance of high quality waters of the State even where that quality is better than needed to protect beneficial uses, unless specific findings are made.

Water quality may not be allowed to be degraded below what is necessary to protect beneficial uses in any case.

2. **Order No. 91-017. Waste Discharge Requirements (WDR) for the U.S. Department of Agriculture, Forest Service, Plumas National Forest, Walker Mine Tailings, Plumas County:**

A. Discharge specifications (water over the dam and from the culvert):

1. Neither the treatment nor the discharge shall cause a pollution or nuisance as defined in Section 13050 of the California Water Code.

2. The discharge shall not cause degradation of any water supply.

3. The discharge shall not have a pH less than 6.5 nor greater than 8.5.

4. The discharge shall not contain more than 0.2 ml/l settleable solids.

B. Sludge and Solid Waste Disposal:

1. Sludge and/or solid wastes generated by remediation activities shall only be discharged to a waste management unit which is in compliance with the requirements of Title 23, Division 3, Chapter 15, California Code of Regulations (CCR), or to a site(s) which has been approved by the Executive Officer.

2. The Discharger may propose alternative sludge or solid waste disposal alternatives if the waste is to be treated. Disposal of treated waste must comply with Chapter 15 requirements and be approved by the Executive Officer.

C. Receiving Water Limitations:

1. The discharge(s) shall not cause concentrations in Little Grizzly Creek, at a point immediately above Road 25N42 and above the west side spring discharge (R-5) to exceed the following limits:

Constituents	*Units*	*Limitation**
Aluminum	ug/l	750.00
Cadmium	ug/l	1.80
Copper	ug/l	9.22
Iron	ug/l	1000.00
Lead	ug/l	33.80
Mercury	ug/l	2.40
Zinc	ug/l	65.00

* **[Copper and zinc are the only constituents presently detected at the water monitoring stations. Copper and zinc are synergetic in their effects to aquatic organisms. The current goal of remedial actions at the site is to reduce the release of copper and zinc (Cu + Zn) to 10 mg/l or less, at hardness of 50 mg/L $CaCO_3$.** See Figure 2, Browns Cabin Station.]

Receiving water limitations for cadmium, copper, lead, and zinc are adjusted for hardness at the Little Grizzly Creek upstream station (R-3), according to equations established in the Waste Discharge Requirements Order.

2. The discharge shall not cause visible oil, grease, scum, foam, floating or suspended material in the receiving waters or watercourses.

3. The discharge shall not cause concentrations of any materials in the receiving waters which are deleterious to human, animal, aquatic, or plant life.

4. The discharge shall not cause aesthetically undesirable discoloration of the receiving waters.

5. The discharge shall not cause bottom deposits in the receiving waters.

6. The discharge shall not cause fungus, slimes, or other objectionable growths in the receiving waters.

7. The discharge shall not increase the turbidity of the receiving waters by more than 20% over background levels.

8. The discharge shall not alter the normal ambient pH of the receiving water more than 0.5 units.

3. Crystalline silica dust presents the highest public health concern at the tailings. The Safe Drinking Water and Toxic Enforcement Act of 1986 identifies airborne particles of respirable size, crystalline silica (Chemical Abstracts Services Registry date: October 1, 1988) as known to the State to cause cancer. Although listed, the State of California, Environmental Protection Agency, Department of Toxic Substances Control did not identify any specific air quality ARARs for the site. The Plumas County Department of Environmental Health has provided general comments that it will enforce exposure restrictions upon frequent users and workers at the site by requiring restricted access and/or use of proper respiratory equipment.

VII. *Remedial Action Goals and Objectives*

GOALS. Protection of the beneficial uses of Little Grizzly Creek from the release of contaminants to the environment (receiving waters) from the tailings area.

Protection of the health of users and workers at the site from the exposure to tailings dust.

OBJECTIVES. To reduce the release of contaminants from the tailings area to Dolly Creek and Little Grizzly Creek by meeting the requirements for receiving water as stated in State Water Board Resolution No. 68-16 (the antidegradation policy requirement), or, if not feasible, the requirements in Waste Discharge Requirements Order No. 91-017 within five (5) years of completion of remediation work.

To eliminate the inhalation of fugitive dust by humans using and working at the site within five (5) years of completion of remediation work.

VIII. *Description of Remedial Alternatives*

The no action alternative serves as a baseline for comparison of the other alternatives. No action means that no remedial activities will be conducted to reduce or cleanup the hazards associated with the generation and release of contaminants from the tailings material. Surface and perched groundwater monitoring would be conducted as part of this alternative; however, to quantify the impact associated with a no remedial response action. The site conditions would be re-evaluated periodically to determine whether there have been any changes regarding risk to human health and the environment.

The following is a brief summary of the alternatives considered:

The tailings have been divided into two areas for treatment; (1) Dolly Creek and (2) the remainder of the tailings. The Dolly Creek area includes the active stream channel and the area extending out to, and including, the gully banks.

Treatment alternatives considered, but dropped from the analysis include:

> Alternative 6: Covering the tailings area with impermeable material to reduce the amount of oxygen and water that contact sulfide materials. This would be very costly and impractical for this site.
>
> Alternative 7: Actively treating water leaving the site to remove contaminants. This also would be very costly and impractical for this site.
>
> Alternative 8: Use of bactericides to stop the ferric to ferrous transfer. The bacteria to be treated would be found in the upper layers of the tailings material. These bacteria have been found to be, for all practical purposes, non-existent in this area.

Any of these treatments could be revisited if the proposed treatments are found to be ineffective on the site or if new information about the site or these treatments arises.

There are two proposed alternatives, plus the no action alternative, for each of the two areas. The four alternatives considered in detail are summarized below.

Area 1, the Dolly Creek area, would be treated by either Alternative 2 or 3.

Alternative 2: Channel Erosion Control and Development of a Wetland for Passive Water Treatment.

Under this alternative, Dolly Creek would be stabilized by reconstructing the natural geometry of the channel and revegetating all banks in the upstream portion of the channel and by constructing a wetland in the lower portion. The wetland would not only stabilize the lower portion of Dolly Creek, but it would serve to passively treat contaminated water leaching through the tailings material to Dolly Creek before it flows to Little Grizzly Creek.

Alternative 3: Diversion of Dolly Creek Around the Tailings Area, Stabilization of Dolly Creek Below the Diversion and Passive Water Treatment.

Alternative 3 would include the treatments described above in Alternative 2 plus the diversion of Dolly Creek around the tailings area to Little Grizzly Creek. This would separate the "good" water from the "bad" water. Water from rain and snow melt plus spring and other groundwater flows would still leach metals from the tailings material to Dolly Creek. Flood flows from the upper watershed area would still pass through the existing Dolly Creek channel on the tailings.

Area 2, the remainder of the tailings area, would be treated by either Alternative 4 or 5.

Alternative 4: Revegetation and Wind Erosion Control.

Alternative 4 would involve modest, low-cost efforts to revegetate the area plus provide wind erosion control measures. The surface of the tailings area is constantly blowing around, inhibiting natural revegetation from occurring. Wind on the area also causes large dust clouds to form, creating a health hazard because it contains large amounts of very fine grained, crystalline silica.

Revegetating the surface of the tailings area is expected to not only eliminate the wind problems over the long-term, but to eventually reduce oxygen in the acid producing, aerated upper layer of the tailings material (the vadose zone), thus reducing the release of contaminating metals to Dolly Creek, and the wetland.

This alternative would use plants that are known to survive conditions existing at the site. Fertilizers would also be used where needed. Mixing plant species such as lodgepole pine and legumes is expected to enhance plant survival. lodgepole pine would provide one of the major tree components and legumes would provide a long-term nitrogen supply to the trees. The underlying principle for successful revegetation of the site is the maximization of plant diversity utilizing plants of known tolerance to the site. This should provide a stable plant community that would require little to no long-term maintenance.

Alternative 5: Vegetated Soil Islands and Wind Erosion Control.

Alternative 5 would employ the same wind erosion control measures as in Alternative 4, but instead of immediately revegetating the entire area, islands of imported soil would be constructed and vegetated. Because covering the entire tailings area with soil was determined to be impractical and too costly, this alternative was developed. The vegetation on these islands would be expected to migrate into unvegetated areas; areas containing no imported soils.

None of the above described treatment alternatives would preclude future treatments that employ improved technologies, providing that they meet treatment objectives and site requirements. Potentially, technologies that would result in total removal and treatment of the tailings material would provide a more permanent solution than the alternatives considered, if cost effective and environmentally acceptable.

IX. *Comparative Analysis of Alternatives*

Discussion. Each alternative was evaluated using the nine criteria outlined in 40 CFR 300.430, paragraph (e) (9) (iii). These evaluation criteria are as follows: overall protection of human health and the environment; compliance with ARAR's; long-term effectiveness and permanence; reduction of toxicity, mobility, or volume through treatment; short-term effectiveness; implementability; cost; State acceptance; and community acceptance.

Upon completion of the the detailed analysis of each alternative against each of the nine evaluation criteria, a comparative analysis was conducted that focused on the relative performance of each alternative against those criteria. A preferred treatment was selected and a proposed plan developed and presented for review and comment to the public, State agencies involved with the project, and identified Potential Responsible Parties (PRPs). Two public meetings were held to discuss the proposed plan, one in Portola and one in Taylorsville. Comments were reviewed in consultation with the State in order to determine if the proposed plan is the most appropriate treatment for the site. Changes to the proposed plan are discussed in the following section.

Analysis. There are two areas to be treated, Dolly Creek and the remainder of the tailings area. Alternatives should be combined to provide total site remediation. Alternatives 2 and 3 treat Dolly Creek and its riparian areas and banks. Alternatives 4 and 5 treat the remainder of the tailings area. For this reason, only Alternative 2 and 3 can be compared together and Alternative 4 and 5 compared together. Each alternative and its treatment area are as follows:

	Alternative	*Treatment Area*
1	No Action	N/A
2	Channel Erosion Control and Developed Wetland	Dolly Creek
3	Alternative 2 plus Diversion of Dolly Creek	Dolly Creek
4	Revegetation and Wind Erosion Control	Remainder of Tails
5	Vegetated Soil Islands and Wind Erosion Control	Remainder of Tails

The following summarizes the comparative analysis using the nine evaluation criteria listed above.

Overall Protection of Human Health and the Environment

The implementation of either Alternative 2 or 3 alone would not provide protection of the health of humans using or working at the site because they are strictly designed to treat the problems associated with the flow of Dolly Creek on the tailings area and contaminants that have leached into Dolly Creek.
The control of wind and water erosion and dust containing respirable crystalline silica would require the implementation of either Alternative 4 or 5. Long-term institutional controls, similar in all alternatives, would provide immediate protection of human health.

All alternatives, except the No Action alternative, reduce contaminant release to some level. Alternatives 2 and 3 would passively treat the waters of Dolly Creek in a wetland environment before it enters Little Grizzly Creek. Alternatives 4 and 5 would reduce oxygen in the vadose zone of the tailings area, thereby reducing contaminant concentrations in the leachate water flowing to Dolly Creek.

The implementation of Alternative 2 or 3 would also stabilize the Dolly Creek channel and gully walls, reducing erosion and sedimentation. Alternative 3 provides exactly the same treatment as Alternative 2 with the addition of a diversion on Dolly Creek upstream of the tailings area and routed around the site to Little Grizzly Creek. This would reduce the amount of water flowing in the Dolly Creek channel located on the tailings area. Water would still flow in the abandoned channel, but at a much reduced rate, along with the leachate water from the tailings material. Passive water treatment would still be relied upon.

An unknown problem would be the reduction of the water table in the tailings material if Dolly Creek is diverted around the tailings area. It is unknown whether or not springs and seeps in the area would maintain the existing water level alone. It is important that the tailings water table be kept as high as possible to limit the amount of tailings material that is exposed to water and oxygen.

Alternatives 4 and 5 would stabilize the remainder of the tailings area. Alternative 4 would result in the immediate revegetation of the site through use of special plant material adapted to the site, fertilizers, some organic material, and wind erosion control. Total vegetation coverage of the site from the implementation of Alternative 4 is expected to occur in approximately 10 years.

Alternative 5 would import soil to form islands to be revegetated. Importing soil to the site would increase costs considerably. It is expected that over time (30 years) this vegetation would spread into the inter-island areas, where wind erosion control measures would be used. Wind erosion control measures would utilize logs, straw, forest debris and "brush trench packs," vegetation, and wind fences. Water erosion would also be minimized by these measures.

Compliance with ARARs

Since Waste Discharge Requirements are not currently being met, the no action alternative cannot meet ARARs. All other alternatives would be expected to meet the specific ARARs they are designed to address.

The implementation of Alternative 2 alone (no upstream diversion) is expected to meet water quality ARARs. The success of the treatments would be evaluated at five year intervals. If water quality improvements are occurring, no further actions would be taken except monitoring. If water quality is not improving, or doesn't appear to be able to meet ARARs, further remedial actions would be considered, including the diversion of Dolly Creek around the tailings area (Alternative 3). Alternative 3 would be expected to reduce the amount of contaminants entering Little Grizzly Creek from Dolly Creek, but water treatment would still be required to reduce metal concentrations in the leachate water from the tailings material. Alternative 3 would reduce the amount of contaminated water flowing to Little Grizzly Creek, but may not reduce the amount of contaminants released from the site to Little Grizzly Creek without the wetland water treatment system.

Alternatives 4 and 5 are expected to help reduce acid generation and the release of contaminants to leachate water. By themselves they would not meet ARARs, but do address the human health hazards caused by inhalation of dust from the site. It is expected that Alternative 4 or 5 would begin reducing acid generation in less than ten years.

The evaluation of the ability of the alternatives to comply with ARARs includes a review of chemical and physical specific ARARs plus action items to prevent human exposures. These were presented earlier in this report. There are no known location-specific ARARs for this site.

Long-term Effectiveness and Permanence

The treatment of Dolly Creek with either Alternative 2 or 3, PLUS the treatment of the remainder of tailings area with either Alternative 4 or 5 provides the highest degree of long-term effectiveness and permanence, treating all known contaminant pathways plus the generation of contamination over the entire site. If either Alternative 2 or 3 is implemented alone, only partial treatment would be provided, leaving natural mechanisms to treat the remainder of the site. The implementation of either Alternative 4 or 5 alone would not meet water quality goals, no matter how long they are in place.

Long-term protection of human health would best be achieved by institutional controls if either Alternative 2 or 3 is implemented alone. Institutional controls could be terminated after site stabilization if either Alternative 4 or 5 is implemented along with Alternative 2 or 3.

There is no evidence that there is any long-term advantages between Alternatives 2 and 3 at this time. Monitoring water quality is expected to give the evidence needed to consider the installation of the diversion structures in Alternative 3.

It is expected that both Alternative 4 and 5 would meet project goals, although it is estimated that Alternative 5 would require at least 30 years to become fully effective. Acid generation and mobility of contaminants would be reduced by site stabilization and reduced oxygen in the vadose zone. Passive treatment of water leaving the site would eliminate release of contaminants leaching to Dolly Creek, or, at least, reduce them to acceptable levels.

The difference between Alternatives 4 and 5 is the time of effectiveness and probability of success. Alternative 4 would address the entire treatment area at once, but would not use any soil amendments. It would rely solely on the use of proper vegetation and planting techniques. Alternative 5 creates islands of soil where revegetation is expected to flourish, then it relies on the spread of that vegetation between the islands, finally covering the entire site. Since wind erosion would be controlled, vegetation spread is expected to occur, but slowly. Revegetation of the entire site would probably not be as thorough as in Alternative 4 and, therefore, less effective in the long-term. Both alternatives are expected to be permanent, requiring little maintenance after final vegetation establishment. Institutional control of public access to the site would be required to protect rehabilitation features and plants until the site has become fully rehabilitated.

The stabilization of Dolly Creek would be permanent, but would require 5-10 years of maintenance. The wetland would require long-term (greater than 30 years) maintenance to facilitate its effectiveness. Monitoring water quality would also occur as a long-term element to ensure that all treatments are functioning properly and ARARs continue to be met.

Reduction of Toxicity, Mobility, or Volume Through Treatment

TOXICITY: Copper and zinc toxicity in Dolly Creek and Little Grizzly Creek is expected to be reduced to levels required by the Central Valley Regional Water Quality Control Board by reducing the amount of copper and zinc released into these streams. All alternatives, except Alternative 1 (No Action), would reduce the release of copper, but in different ways.

Alternatives 2 and 3 would reduce the transport of copper that is attached to sediment particles by stabilizing the Dolly Creek channel and its gully. Both alternatives would then treat Dolly Creek water and the tailings leachate by passing the water through a constructed wetland. In addition, Alternative 3 would divert the lesser contaminated water of Dolly Creek around the tailings area, discharging it into Little Grizzly Creek. Leachate water flowing from the tailings into Dolly Creek below the diversion would be treated by the constructed wetland. Without the full flow of Dolly Creek, the wetland size would be much smaller than needed for full treatment of leachate water, and the level of the aquifer now maintained at near the level of the sediment dam may drop during the drier season of the year, exposing more tailings material to oxygen and acid generation.

Alternatives 4 and 5 would reduce the release of copper to Dolly Creek by reducing the generation of acid within the tailings vadose zone. Much of the oxygen needed for the production of acid would be consumed by decomposing organic debris. The difference between these alternatives is the length of time for this process to become fully effective. Alternative 4 is expected to take much less time to become fully effective (approximately 10 years) than Alternative 5 (approximately 30 years).

Blowing sand and dust (containing crystalline silica particles) would be reduced or eliminated by implementing either Alternatives 4 or 5. Both alternatives would reduce or eliminate dust emanating from the site, but again, Alternative 4 would be expected to become fully effective much sooner than Alternative 5. Wind erosion control features would be installed with the implementation of either alternative. These devices are expected to reduce the transport of sand and the generation of dust to very low levels, but need to be replaced by plants for long-term success. Alternative 4 would require maintenance of these devices for approximately 10 years, while Alternative 5 would require approximately 30 years.

MOBILITY: The constituents of concern are sediment, blowing sand and dust, and metals in solution (copper and zinc). As discussed above, Alternatives 2 and 4 are expected to best control the release and transport of these constituents.

VOLUME: None of the alternatives reduce the volume of tailings material. All material would be treated on-site.

GENERAL DISCUSSION: As mentioned in the previous section, both Alternative 4 and 5 would reduce wind erosion and airborne contaminants. Vegetation growing over the tailings area is expected to reduce oxygen in the vadose zone of the tailings material by normal plant respiration processes as roots and other organic matter decomposes, thereby reducing the generation of acid and mobilization of copper and zinc, the primary contaminants released from the site.

The wetland would be relied upon to extract soluble copper and zinc (plus other metals if released), transforming them into inert precipitates. Some of the metal contaminants would be taken up by the plants. The effectiveness of the wetland is expected to vary with the seasons and the amount of water required to be treated. Raising the elevation of the tailings dam about one foot may be needed to facilitate wetland establishment and size.

Stabilizing Dolly Creek is expected to reduce sediment production to acceptable levels or lower. This would reduce the release of copper and zinc from sediment to downstream areas.

Remediation of Air Quality. Concentrations of total crystalline silica are present in the tailings dust at levels of 19-23 percent. Silicosis, lung cancer, and secondary respiratory infections could result from repeated exposure to the dust. It is not known what the lower level of human exposure is, although respiratory effects are usually documented after occupational exposure to silica concentrations for several years. Expected results of implementing either Alternative 4 or 5 is the near total reduction of dust generated at the site. The near total reduction of fugitive dust at the site is expected to take approximately 10 years if Alternative 4 is implemented and 30 or more years with Alternative 5.

Remediation of Water Quality. Recent concentrations of copper and zinc at the compliance station for water quality (located downstream from the confluence of Dolly Creek with Little Grizzly Creek) ranged from 0.036 mg/L to 0.14 mg/L for copper and 0.0044 mg/L to 0.013 mg/L for zinc. The synergistic affect of copper and zinc on aquatic biota is well documented. For this reason, the water quality goal at the compliance station has been established for copper *plus* zinc at a concentration not to exceed 0.01mg/L. Examining the recent concentrations of copper and zinc, copper plus zinc has ranged from 0.040 mg/L to 0.15 mg/L. These concentrations are lowest during the high runoff and winter (cold) months and highest during mid-summer months.

Even though copper is required in animal metabolism, concentrations in fresh water above 0.01 mg/L (dependent on the alkalinity of the water) can have adverse effects, especially to the young or juvenile forms of aquatic animals.

Alternatives 2 and 3 include water treatment using a basic compost wetland, which is expected to remove copper and zinc from Dolly Creek to near background levels if properly maintained. Walker Mine, the primary source of copper to Dolly Creek and Little Grizzly Creek for many years, was sealed in November, 1987, reducing copper and zinc levels in Dolly Creek above the tailings area to near background levels during most of the year. Some copper is still released from the site; not from the sealed tunnel, but rather the waste rock and contaminated soil areas at the mine and milling sites. This problem is currently being addressed by the CVRWQCB and is expected to be remediated in the near future, possibly by 1995. The existing source of copper and zinc is leachate water that moves from the tailings material into Dolly Creek as it flows across the tailings area.

Since the primary source of copper, the mine portal, has apparently been successfully treated, only the small amount of copper and zinc released from the tailings material and the mine site remains. The mine site will soon be treated. Passive water treatment using a wetland should successfully remove the remaining copper and zinc without specialized wetland treatment technology. Periodic maintenance will require removing and treating contaminated soil, compost, and plant material and rejuvenating the wetland to its proper size and replacing lost compost and plant material. Structures designed to slow water movement will have to be replaced periodically, but should last longer than 30 years. Since iron is usually below the water quality objective of 1.0 mg/L and pH values are always near neutral, the use of an anoxic limestone drain for iron removal and neutralization is not warranted.

Proper wetland functioning also relies on active plant and bacterial metabolism, which is highest during the active growing season. This is also when the concentrations of copper and zinc in the receiving water are highest. Winter months will result in lower wetland activity and lower copper and zinc concentrations, because of dilution and lower activity of the mechanisms that cause release of the metals in the first place.

Revegetation of the tailings area will not only reduce wind erosion and the generation of fugitive dust, but it will also reduce the release of copper and zinc (and any other metals that could become mobilized over the years) by reducing the amount of oxygen in the vadose zone (the oxygenated zone between the top of the water table and the top of the tailings). This will reduce the release of copper and zinc to Dolly Creek and the amount of these metals to be removed by the wetland. An estimated reduction of metal mobility has not been made, but monitoring the several wells already installed in the tailings should give some indication of the relative changes in metal mobility achieved.

Short-Term Effectiveness

The implementation of Alternative 2 plus 4 is expected to have the greatest short-term effectiveness by treating all pathways and providing immediate reduction of respirable silica dust. Some particulate emissions is anticipated during the implementation of all alternatives, however, and proper respirators would be required to be worn by all workers whenever dust conditions warrant.

Implementability

Alternative 3 treatments are the same as Alternative 2 with the addition of the diversion works. This is an additional construction and maintenance complication.

Alternative 4 and 5 require similar wind erosion control features and installation requirements. Alternative 4 revegetation would be the simplest to conduct. Alternative 5 would require importing soil and construction of islands, mulch, and vegetation. The location of these islands would be critical for aiding the spread of plants to adjacent areas.

All alternatives use proven techniques and readily available services and materials.

The implementation of Alternative 3 with Alternative 5 would be the most complex to construct and maintain. The simplest treatment would be the implementation of Alternative 2 alone with institutional controls.

Cost

Alternative 2 alone has the lowest capital cost and Operation and Maintenance (O&M), but doesn't provide full site treatment and long-term effectiveness. The implementation of either Alternative 4 or 5 with either

Alternative 2 or 3 would provide full treatment of the site. Mixing Alternative 2 with Alternative 4 would require a lower capital cost than mixing Alternative 2 with Alternative 5. The use of Alternative 3 would greatly increase the cost of treating the site, both in its capital cost and O&M cost. Additional work and expense could be required if revegetation doesn't meet expectations, increasing O&M costs over the estimates.

Combining Alternatives 2 and 4, provides the best overall effectiveness proportional to costs. The following table compares values and costs of each alternative. Refer to the *Feasibility Study* for a more detailed discussion.

ALTERNATIVE	*30-YEAR NET VALUE*	*CAPITAL COST*	*O&M COST*
1	$0	$0	$8,000
2	**$81,000**	**$240,000**	**$8,400**
3	-$21,000	$1,544,000	$20,400
4	**$63,000**	**$180,000**	**$4,200**
5	$42,000	$330,600	$1,400

State Acceptance

The State does not accept the No Action alternative. No "cease-and-desist order" for the site has been imposed on the Forest Service, but has been mentioned. Through conversations with State personnel, the CVRWQCB favors those alternatives that more completely treat the site and as quickly as possible. They favor most the proposed plan, discussed in section X, below.

Community Acceptance

Very few responses were received from the public. Of the responses received, most were informal and favored implementation of the proposed plan. No formal response was received from those who oppose work at the site. Through informal channels, it was learned that several people who use the site for off-highway vehicle recreation would prefer that the site remain as it is and that it remain open for their use.

Table 3 summarizes the advantages and disadvantages of each alternative.

X. *The Proposed Treatment Plan and Modifications*

The assembled remedial action alternatives represent a range of distinct waste management strategies which address human health and environmental concerns associated with the site. They build on one another, enhancing each other, except the no action alternative. The ability of each alternative to meet ARARs and the other evaluation criteria, discussed in the previous section, was evaluated.

Alternative 2 was selected in combination with Alternative 4 (*Channel Erosion Control and Development of a Wetland for Passive Water Treatment + Revegetation and Wind Erosion Control*) as the "preferred treatment". By analyzing the alternatives using the evaluation criteria discussed in the previous section, Alternative 2 plus Alternative 4 were determined to permanently treat the entire site and best meet the remediation goals and objectives discussed in Section VIII in a timely and cost-effective manner. These alternatives also have the support of the State agencies overseeing these matters, the local communities, and most PRPs.

Because little rejection of the proposed treatment plan was received and no new information was introduced, no modifications to the proposed plan are made.

Because hazardous substances will remain at the site at levels above that allowed for unlimited use and unrestricted exposure, the Forest Service, in cooperation with the CVRWQCB, will review the remedial action no less often than every five years after initiation of the selected remedial action [(40 CFR 300.430, paragraph (f)(4)(ii) and (f)(5)(iii)(C)].

TABLES

FIGURES

APPENDIX

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SUPERIOR COURT OF THE STATE OF CALIFORNIA

FOR THE COUNTY OF PLUMAS

THE PEOPLE OF THE STATE OF CALIFORNIA, Plaintiff, v. ROBERT R. BARRY, CALICOPIA CORPORATION, and DOES I through XXX, exclusive, Defendants.	NO. 11901 DECLARATION OF WILLIAM J. MARSHALL

I, William J. Marshall, declare under penalty of perjury if called as a witness in the above-captioned matter, I would testify as follows:

That for the past sixteen months I have been, and now am a Senior Water Resources Control Engineer for the Regional Water Quality Control Board, Central Valley Region (Regional Board). I have a Bachelor of Arts degree in geology from Rutgers University, a Bachelor of Science degree in civil engineering from Newark College of Engineering, and a Masters

Degree in civil engineering from California State University, Sacramento. I am a registered engineer in the State of California. Prior to my employment with the Regional Board, I worked for the State Water Resources Control Board as a senior engineer in the area of water rights adjudication.

As Senior Water Resources Control Engineer my duties involve supervising and approving the actions of area engineers and setting policy for the enforcement of regulations. I make enforcement decisions for water quality violations occurring within my assigned region. I am responsible for the Regional Board's activities within several counties including Plumas County. All documents regarding Walker Mine which come to the Regional Board office are directed to me. I am familiar with the Regional Board's official file on Walker Mine, and I know the history of the Regional Board's involvement with Walker Mine from personal knowledge and from business records in the official file maintained by the Regional Board.

Walker Mine is an inactive copper mine located in east central Plumas County about twenty miles east of Quincy. Walker Mine discharges acid mine drainage to Dollie Creek and Little Grizzly Creek, upper tributaries of the East Branch of the North Fork Feather River. Above the mine these creeks are of excellent quality and contain abundant levels of aquatic insects and fish. However, below the mine the condition of the waters of Dollie Creek and Little Grizzly Creek is such that aquatic organisms cannot survive. Approximately ten miles of watercourses are toxic to aquatic organisms due to the acid mine drainage. Only through the dilution by other tributaries at the confluence of Little Grizzly Creek with Indian Creek is the quality of these waters improved sufficiently for aquatic habitat.

Dollie Creek and Little Grizzly Creek below its confluence with Dollie Creek are grossly polluted by the discharge from the Walker Mine. The discharge originates from the mine adit, flows down and across the mine workings, and into Dollie Creek. The quality of this discharge is acidic and mineralized, having a pH as low as 3.3 and copper content as high as 69 miligrams per liter. The affected creek waters contain high concentrations of copper, zinc, iron, sulphates, and other mineral compounds and toxic materials, making them unfit for aquatic habitat. Below its confluence with Indian Creek the waters of Little Grizzly Creek are diluted enough to support aquatic life. However, even in Indian Creek periodic flows containing copper from the Walker Mine cause concentrations above tolerance limits for many aquatic organisms.

Walker Mine was discovered in 1904 and actively mined from 1916 to 1932, and from 1935 to 1941. Since the mid-1940's, the mine has discharged acid water containing metals toxic to fish. The Plumas County Assessor's Office indicates that Robert R. Barry received the Walker Mine property on 24 September 1948 from Coleman Burke by way of a quitclaim deed. Apparently, this was not recorded until 19 November 1965 at which time it was also deeded to Calicopia Corporation, a Robert R. Barry family-owned corporation (in New York State). A recent check with

New York indicates that Calicopia Corporation was dissolved by proclamation on 20 December 1977.

Waste discharge requirements were first adopted in 1958. The mine has continuously violated these and subsequent requirements, except for short periods in extremely dry years when the discharge has ceased. The following chonology provides a brief history of Regional Board actions relating to Walker Mine.

24 Apr 58	Waste Discharge Requirements issued.
8 Apr 59	Cleanup and Abatement Order issued.
18 Jul 63	Cease and Desist Order issued.
26 Oct 70	Abatement Order (Section 13305 of the California Water Code) issued.
8 Sep 71	Cleanup and Abatement Order issued.
23 May 75	Water Discharge Requirements Order No. 75-119 issued.
1978	Regional Board hires D'Appolonia Consulting Engineers with federal 208 funds to prepare report on Walker Mine abatement. Report recommends surface water diversion and wastewater treatment.
30 May 80	Waste Discharge Requirements Order 80-58 adopted; Cleanup and Abatement Order No. 80-70 adopted.
Jul 80	Surface water diversion ditches constructed under staff's direction at owner's expense.

Jul 81	Pearson and Associates Consulting Engineers proceed with State Clean Water Bond monies to evaluate treatment alternatives and construct a pilot project on-site.
Sep 83	Pearson and Associates complete draft "Pilot Plant Operation, December 1982 to July 1983, and Design Report".
9 Dec 83	Request to Abate Pollution, Order No. 83-148 adopted (Section 13305 of the California Water Code).
Feb 84	Regional Board sends out Request For Proposals to design and construct mine seal.
Jun 84	Steffen, Robertson and Kirsten, Consulting Engineers, selected to design and construct mine seal for Walker Mine.

D'Appolonia Consulting Engineers were hired by the Regional Board in 1978 to prepare a report on feasible methods of abating the acid mine drainage at Walker Mine. D'Appolonia submitted the final report to the Regional Board in 1979 and therein recommended that the Walker Mine pollution problem be treated by construction of a limestone barrier, neutralization plant, and sedimentation basins. The Regional Board then sent out Request For Proposals and subsequently awarded a contract to Pearson and Associates, Consulting Engineers, to construct an on-site pilot project and evaluate treatment alternatives. The Pearson draft design report (September 1983) estimated that the diversion/treatment process of handling the mine drainage would entail a capital cost of more than $500,000 and additional significant operation and maintenance costs. The Regional Board then determined that this treatment method was no longer feasible for financial reasons and that the alternative solution of a mine seal should be investigated. Requests For Proposal were sent out and in February of 1984 Steffen, Robertson and Kirsten Consulting Engineers from Lakewood, Colorado were selected to design and construct a mine seal at a cost of $100,000. State Clean Water Bond Funds are being used to finance the project. Defendants have failed to comply with the Regional Board's orders to abate the pollution from Walker Mine (Waste Discharge Requirements Order No. 80-58 is attached hereto as Exhibit 1; Cleanup and Abatement Order No. 80-70 is attached hereto as Exhibit 2).

The Regional Board requested access to the Walker Mine tunnel (which is blocked by a locked metal door) and property by letter dated January 19, 1984 (attached hereto as Exhibit 3), and again by letter dated March 14, 1984 (attached hereto as Exhibit 4). Counsel for the Regional Board requested access by letter dated June 7, 1984 (attached hereto as Exhibit 5). Counsel for defendants refused such access by letter dated June 29, 1984 (attached hereto as Exhibit 6). The Regional Board again requested access by letter dated July 6, 1984 (attached hereto as Exhibit 7). Defendants have failed to respond.

Now that a contractor has been selected it is necessary that access to the mine be provided to enable the engineering firm hired to design the mine seal to conduct the necessary on site investigation. Furthermore, the main portal of Walker Mine is 6180 feet in elevation and early snows in the Walker Mine area could make access difficult. Typically the mine is inaccessible due to snow from October through May but occasionally snow occurs in September. To avoid potential weather problems access needs to occur as soon as possible.

I declare under penalty of perjury that the foregoing is true and correct and that this declaration was executed on July 15, 1984, at Sacramento, California.

William J. Marshall

WILLIAM J. MARSHALL

For:
From: J. Prochnau
Re: Walker Mine, Plumas County, California
Preliminary Evaluation

October 21, 1984

Two days, 4 & 5 October, were spent at the Salt Lake City office of Standard Bullion reviewing data on their Walker Mine, Plumas Co., California. A third day, 16 October, was spent on the property with Paul Spor and Jan Donato, Standard's geologist and local watchman/caretaker. The field examination was cut short by a heavy, unseasonable snowfall and we were unable to carry out the surface sampling necessary to assess the potential bulk tonnage oxide gold deposit touted by Standard.

The wide, lenticular copper gold orebodies mined by Anaconda prior to closure of the mine in 1942 remain open down dip. According to estimates at shut-down there are remaining blocked underground reserves on the order of 1,175,000 t @ 1.5% Cu, 0.70 opt Ag and 0.035 opt Au, and perhaps an additional 2000000 t of probable ore. Potential low grade halo mineralization, amenable to large scale bulk mining (target ±30 mt @ 0.5 Cu to 200'), was investigated by Noranda in 1969-70 without success. Neither base metal target is attractive at present or forseeable conditions.

The possibility of significant low grade gold mineralization (10 mt @ 0.05 Au) in the oxidized upper portion of the Walker vein system has been proposed by their consultant, Galen Hansen. However, there is little hard data to support this idea and validation of the potential target will require sampling of surface cuts and glory holes. This will be carried out when/if the weather improves and the early snow melts.

Location

South end of Plumas Copper Belt, some 27 mi. northwest of Portola and 15 mi. southeast of Taylorsville, Plumas County, California (Fig. 1)

Sections 5-8 incl. T24N, R12E

Section 12 T24N, R11E

Sections 7,8,17-20 incl., 29-32 incl. T25N, R12E

Sections 11-14 incl., 23-26 incl., 35-36 T25N, R11E MDM

Terrain is typical of the Sierra with elevations ranging from 4000-7000 feet. Property lies in a heavy snow belt and is effectively inaccessible between early December and May.

Property & Ownership

34 unpatented mining claims	687 acres
Mill & townsite patents	108 acres
347 unpatented lode claims	7169 acres

The Standard Bullion Company, Inc.
3445 South Main Street, Suite 107
Salt Lake City, Utah 84115
Tel (801) ~~486-0873~~
265-0759

Background - Mining History

1905	Discovery
1910	Initial shaft sinking by Walker Mining Company
1916-1920	Control acquired by International Smelting (an Anaconda subsidiary), 75 tpd mill built, 700 level adit X-C driven and the principal orebodies discovered and developed
1923	Modern 500 tpd flotation concentrator built
1923-1942	Principal operating period. Closure in 1942 due to increased costs & wartime labor shortages
Production	5,300,000 t @ 1.55 Cu, 0.7 Ag, 0.04 Au

Background - Post Production History

1949	Acquired by Pet. Barry
1969-70	Noranda, Surface mapping, geochem, geophysics, 11 core holes testing low grade disseminated

	halo mineralization and geophysical anomalies on extensions and parallel zones.
1976-77	Amax (No documentation available) geology, geochem, 3 core holes
1979-1981	Conoco, geology, geophysics, geochem. Re-interpreted Walker as volcanogenic and drilled 11 core holes testing for extension of the Walker zone north under volcanic cover, FW "exhalite" zones, "exhalite" zones in north part of property.

Attached Map 3 shows distribution of surface holes and survey work by Noranda, Amax and Conoco.

Mine Development & Facilities

The Walker is developed by a 3600 foot adit x-cut and 8000 foot drift at the 700 level, and an internal shaft to 1200 feet. The x-cut and drift are open and accessible by tram to the vicinity of the Central Orebody and internal shaft (Maps 4 & 5, Long. Projections). Rest of the 700 level to the North, 712 and Piute orebodies is accessible on foot. Parts of the mine above 700 are accessible through raises and open stopes. The mine is flooded below the 700 level. Ground at the 700 level holds well with a minimum of support although there are reports describing difficult ground conditions in some working areas at time of closure.

Mill has been dismantled and sold years ago. Surface buildings and equipment are in good shape. All of the drill core is neatly stored on site.

General Geology

The Walker property is underlain by a five mile long strip of Jurassic metasediments and metavolcanics overthrust by Paleozoic sediments along its west boundary, intruded by Nevadan granite on the north and south and capped by Tertiary volcanics to the east (Figure 2 and Map 3).

The units uniformly trend NNW'ly and dip steeply east.

Mine Geology and Mineralization

The Walker copper gold deposits are lenticular "veins" consisting of massive chalcopyrite seams and stringers in a quartz gangue with locally abundant magnetite. The veins are essentially conformable with the host hornfelsic sediments or volcanics and strike NNW'ly with 60°E dips. Ore shoots rake directly down dip.

The mineralized zone has an overall strike length of 8000 feet (about half making ore in six "shoots"), developed slope depth of 1200 feet and overall width of ±200 feet. Ore grades within this broad zone occurred over thicknesses of 5-60 feet.

Following is a tabulation of ore shoot dimensions (See Maps 4 & 5).

Deposit	Length (ft)	Thickness	Slope Length
South	250	20	300
South HW	400	6	200
Central	800	30	700
North	1200	40	700
712	200	35	600
Piute	800	60	500

These dimensions, and historical production, suggest an overall ore incidence somewhere between 7000 and 14000 tpvf, certainly impressive for this style of mineralization.

Conoco worked hard to make Walker volcanogenic and certainly some characteristics support that possibility (general geologic environment and conformability of mineralized zones with enclosing rocks, remarkable continuity on strike & dip, quartz-magnetite-sulfide association). However, whether the deposit is epigenetic or synvolcanic has little bearing on our assessment of the principal remaining targets. I saw too little of the deposit during my single visit to generate an opinion.

Potential Targets

Following is a brief summary of potential ore targets remaining at Walker:

Underground Ore

Reserves Blocked By Anaconda at Closure 1175000 t @ 1.5 Cu 0.035 Au

Much of the blocked reserve is tied up in pillars and on the fringes of old stopes and, after 40 years, is unlikely to be recoverable.

There is nothing to indicate the known ore shoots have bottomed and, considering the historical ore incidence of about 7000 tpvf, I think one can reasonably assume an additional 1-1.5 mt to 200 feet below the existing workings. apart from two absurdly deep holes by Conoco (respectively 1500 and 1000 feet below the 1200 level) I'm not aware of any drilling below the developed ore shoots. Anaconda, by the way, used an overall figure of 3 mt for blocked, probable and possible reserves at the time of closure.

Prognosis - At historical grades not attractive in existing or foreseeable circumstances. Maybe a reserve for the future.

Low Grade Halo Mineralization

One of Noranda's targets was low grade halo mineralization adjacent to the Walker veins. Although one hole, WM-1, intercepted a core length of 170' @ 0.46% Cu, several others testing this target were blank.

Prognosis - Irregardless of Noranda's results the theoretical target, assuming the entire 8000 foot Walker strike length is mineralized across a width of ±200', could only generate some 30 million tons to a depth of 200 feet. This is hardly realistic, either as a technical target at any time, or certainly as an economic objective in present conditions.

New Zones

Potential strike extensions of the Walker zone under volcanic cover, sub parallel footwall zones, and geologically similar showings in the Copper King area at the north end of the property were investigated successively by Noranda, Amax and Conoco (See Map 3). Despite these programs, and expenditures certainly in the +$1m range, there has not been an ore intercept outside the original mine area.

Prognosis - Although long odds targets may remain outside the mine area, if one is going to achieve near term success at Walker I believe it's going to be on top of, or below, the existing deposits. Moving farther afield, especially under deep volcanic cover, is not, in my opinion, a justifiable exercise.

Oxide Gold Target

Our interest in Walker was initially stimulated by Galen Hansen's proposed oxide gold target of ±10 mt at 0.05 opt gold. Review of available data indicates there is no hard data to support this idea but no overriding reason to discredit it either. Apparently no one has ever bothered to sample for gold at surface.

The size potential is there although a 10 mt target may be expecting too much continuity throughout the entire mineralized zone (say half the 8000x200 foot surface dimensions to 100 feet). My concerns lie more with grade and depth extent of oxide material.

To get a better feel for near surface values I quickly averaged available gold assays from upper level of the mine. These averages are as follows:

Deposit	Nr. Assays	Est. Width	Strike Length	Ave. Au
North	41	±20'	1050'	0.034
South	28	---	1450'	0.044
Piute	82	±40'	1080'	0.050
712	38	---	820'	0.022

The grades are modest at best, represent mineralization within the mined orebodies (not halo material) and are likely within the sulfide zone.

Although not particularly encouraging I continue to feel the surface cuts and glory holes should be sampled, and the depth to sulfides measured in glory holes, connecting underground workings, and shallow drill holes, before firm opinion is drawn. This should take two days. and, with assays, cost $1200.

nac

WATER ABATEMENT PROPOSAL

WALKER MINE

Plumas County, California

October, 1984
Revised January, 1985

Submitted
to

Mr. Robert Barry and Calicopia Corporation

by

The Standard Bullion Company, Inc.
Paul C. Spor
Geologist

Copy
ARR 3/2/89

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SCOPE:

The purpose of this report is to evaluate the data available and determine the magnitude of the inferred contamination of discharged water from the Walker Mine portal and what methods of abatement might be considered in resolving this question.

BACKGROUND:

The Walker Mine has the potential of becoming a major metal producer in California, producing gold and silver with copper being a by-product. During the first half of this century the mine produced significant amounts of copper with gold and silver being the by-products. By developing the known, and defining the potential ore reserves the Walker Mine could again produce significant amounts of metals.

In the 1800's a small shaft was sunk by prospectors in search of gold and this ushered in the beginning of mining in this remote area of Plumas County. In the spring of 1905 Mr. J. R. Walker on a visit to Plumas County from Salt Lake City, Utah purchased two claims and located twenty additional lode claims; they are now known as the Consolidated group. Later that summer, Mr. Walker became aware of two other claims, on the site of the original old prospectors shaft, he purchased the two claims and located ten more. The small shaft was sunk deeper and at a depth of 30 feet encountered the copper sulfide zone. This area became the heart of the Walker Mine. In October of 1909, Mr. Walker organized the Walker Mining Company of Utah. This was the beginning of an incredible part of California history. The mine produced five plus millions of tons of ore that contained approximately 156,410,000 pounds of copper, 4,036,640 ounces of silver and 201,832 ounces of gold. At today's values these metals would have a gross value of $196,540,000. According to the final combined reserve figures of Anaconda Mining Company when the mine was closed in October of 1941, there were 5,436,639 tons of ore remaining in the mine, with grades of 1.37 percent copper, 0.75 ounces per ton silver and 0.035 ounces per ton gold. At today's metal prices the gross value of these metals would be $173,900,000. Additional potential tonnages have been inferred and the total mineralized zone indicates that as much as 64,000,000 tons of material could be available. The grades of this mineralized zone have not been confirmed but the total gross value would have a significant inpact on the economy of Plumas County and the State of California. The Standard Bullion Company, Inc. intends to actively pursue the development of the Walker Mine and confirm the aforementioned resources.

PROPOSAL:

According to the California Regional Water Quality Board, Central Valley Region, the Walker Mine has been discharging water that has had an adverse affect on the environment, since the mine was closed in October of 1941. The Regional Board engaged several different consulting firms to evaluate the effluent from the mine and to recommend/propose methods of abatement. D' Appolonia Consulting Engineers and Pearson and Associates have done perhaps the most comprehensive studies to date. The following Tables (Tables 1 & 2) summarize D' Appolonia and Pearsons findings and Table 3 lists the water discharge requirements of Plumas County, Order No. 75-119 and the NPDES No. CA0080110, California Regional Water Quality Control Board, Central Valley Region.

TABLE 1

Summary of Water Quality Data - Walker Mine and Vicinity

D' Appolonia Consulting Engineers

WATER QUALITY PARAMETER	UNITS	WALKER MINE (1) PORTAL MEAN (RANGE)	WALKER MINE PORTAL MEAN	DOLLY CREEK ABOVE MINING PORTAL MEAN (RANGE)	DOLLY CREEK BELOW MINE PORTAL MEAN (RANGE)
Sampling Record	Years	1957-1978	1978	1957-1970	1957-1978
Field Measurements:					
Estimated Flow	gpm	89.8 (0-224.4)	49.4	166.1(89.8-336.6)	488.8 (244.4-2444.2)
Temperature	°C	7.8 (6.1-12.2)	4.5	(6.7-24.4)	(4.0-18.3)
pH	pH units	4.8 (4.3-6.6)	4.5	7.6 (6.7-7.9)	7.6 (5.5-8.2)
Specific Conduatance	µmhos/cm @ 25°C	303 (95-420)	180	126 (55-241)	138 (99-208)
Laboratory Measurements:					
Acidity	mg/LCaCO$_3$	105 (50-115)	75	-	66 (79-108)
Alkalinity	mg/LCaO$_3$	- (2)	3	-	-
Bicarbonate	mg/L	1.2 (0-2.4)	3.7	-	-
Sulfate	mg/L	120 (8-205)	125	1 (0-8.6)	27 (1-60)
Chloride	mg/L	1 (1-5)	2	-	< 1
Fluoride	mg/L	< .1	-	-	< 0.1
Nitrate	mg/L	4.5	-	-	-
Nitrite	mg/L	0.0	-	-	-
Total Suspended Solids @ 105°C	mg/L	109	-	-	-
Metals:					
Calcium	mg/L	32 (9.8-36)	23.9	-	23 (20-29)
Magnesium	mg/L	6 (6-7)	6.5	-	7 (6-8)
Potassium	mg/L	1.3 (0.92-1.6)	1.4	-	-
Sodium	mg/L	2.6	3.1	-	-
Copper	mg/L	11 (0.23-69)	16.1	0.01 (0-0.16)	0.92 (.1-12)
Zinc	mg/L	0.78 (0.09-3.2)	.70	< 0.001 (0-0.14)	0.10 (0-0.48)
Iron	mg/L	0.8 (0.01-1.4)	1.08	0.32 (.13-.52)	0.3 (0.1-1.7)
Manganese	mg/L	-	3.1	0.0	-
Aluminum	mg/L	4.8 (0.9-12)	-	<0.1 (0-0.4)	0.3 (<0.2-0.4)
Lead	mg/L	<50	<0.08	-	<50
Molydenum	mg/L	<2 (1-<200)	-	-	<2 (<1-2)
Nickel	mg/L	<25	<0.05	-	<25
Silver	mg/L	<10	-	-	<10
Arsenic	mg/L	<0.01	0.02	-	<0.01
Cadmium	mg/L	10	-	-	<10
Chromium	mg/L	-	0.13	-	-

(1) Summary of data provided by the California Water Quality Control Board, Central Valley Region, Conoco and Amax.
(2) - indicates parameter not determined.

TABLE 2

Summary of Water Quality Data - Walker Mine and Vicinity

Pearson and Associates

DESCRIPTION	gpm	pH	Cu mg/L	Zn mg/L	Mn mg/L	Fe mg/L
Mean mine effluent	85.7	4.9	15.1	0.67	2.62	0.39
Dolly Creek above mine	-(1)	7.9	<0.1	0.005	0.007	0.008
Dolly Creek below mine	586	7.4	2.8	0.15	0.71	.52

(1) Indicates parameter not determined

TABLE 3

WATER DISCHARGE REQUIREMENTS
WALKER MINE PORTAL

WATER QUALITY PARAMETER	UNITS	REGIONAL BOARD WALKER MINE DISCHARGE LIMITATIONS (1)	REGIONAL BOARD RECEIVING WATER (DOLLY CREEK) LIMITATIONS (1) (2)		WALKER MINE DISCHARGE LIMITATIONS TO MEET RECEIVING WATER STANDARDS (3)	EPA NATIONAL DRINKING WATER REGULATIONS (4)
			30-DAY AVERAGE	DAILY MAXIMUM		
pH	pH units	6.5 to 8.5	-(5)	-	6.5 to 8.5	6.5 to 8.5
Settleable Solids	mL/L	0.2	-	-	0.2	-
Copper	mg/L	-	0.02	0.05	0.028	1.0
Zinc	mg/L	-	0.1	0.2	0.18	5.0
Aluminum	mg/L	-	0.2	0.4	0.28	-
Iron	mg/L	-	0.2	0.4	0.20	0.3
Manganese	mg/L	-	-	-	-	0.05

(1) Waste discharge requirements for Walker Mine, Plumas County, Order No. 75-119, NPDES No. CA0080110, California Regional Water Quality Control Board, Central Valley Region.
(2) Walker Mine discharge cannot cause concentrations of the listed constituents in Dolly Creek to exceed the stated limits. Additionaly, Walker Mine discharge cannot cause the following in Little Grizzly Creek: aesthetically undesirable discoloration; bottom deposits; floating or suspended materials; concentration of any materials which are deleterious to human, animal, aquatic, or plant life; and violation of any applicable water quality standard for receiving water adopted by the California Regional Water Quality Control Board of the State Water Resources Control Board as required by the Federal Water Pollution Control Act.
(3) Based upon a 200-gallon per minute (0.45 efs) treated mine discharge, median metal levels measured in Dolly Creek above mine portal and calculated low flow (0.33 efs) of Dolly Creek above Walker Mine portal.
(4) Title 40, Code of Federal Regulations, Part 141 and Federal Register, Vol. 42, No. 62, National Secondary Drinking Water Regulation, March 31, 1977.
(5) - indicates no standard given.

Pearson and Associates field tested several methods of removing metals from the Walker Mine effluent, their most successful abatement pilot plant demonstrated that 97 percent of the total copper could be removed from the effluent in a chemical neutralization-sedimentation plant operating at a pH of 10.2. Another study indicated that a limestone barrier produced a stable effluent of pH 6.5 from pH 4.9 influent and precipitated up to 90 percent of total copper in the mine drainage. Pearson and Associates concluded that it would cost from $600,000 to $1,300,000 to construct a plant to produce an effluent containing 0.2 mg/L of total copper and have a pH of 10.2. This plant though producing an effluent that would meet the EPA's National Drinking Water Regulation of 1.0 mg/L of copper would not meet the Walker Mine Discharge Limitions imposed by the California Regional Water Quality Control Board, Central Valley Region, NPDES No. CA0080110 of 0.02 mg/L 30 day average or 0.05 mg/L daily maximum of total copper in the effluent. The standards that are being imposed by the Water Quality Board will be difficult if not impossible to comply with and the cost to implement a plant that would require labor 24 hours per day and constant maintenance is not realistic.

It is Standard Bullions recommendation that a more prudent practical method of abatement be considered. A very old but commonly used practice of removing copper from effluent is to precipitate copper by leaching on iron. This method of removing the copper from water was used in the Walker Mine years ago and is being used today in many parts of the United States and throughout the world and would remove from 90 to 95% of the total copper from the water before it left the mine portal. Although this method would not meet the standard imposed by the Water Quality Board, it would be a major step in the right direction.

To implement this program a complete evaluation would need to be conducted on the underground workings of the mine to identify the areas that may be the major contributors to the alleged contamination and to determine the flow rates from each of these areas on a monthly basis. Once the areas are isolated, the discharge can be carried by flumes, to location, where the contaminated water can be run through launders filled with detinned iron shavings. After this system has been implemented, additional studies would be required to determine how the remaining .55 to 1.61 mg/L of total copper can be removed.

The effluent from the mine should be diverted into the settling pond below the mine in such a way that it would not become recontaminated by the mine dump. Further settling of metals would take place here.

The pH can be controlled by a simple, low maintenance system to mix and aerate water in a pipeline. It consists of a jet pump, which entrains air by Venturi action, and a static mixer, which induces turbulent flow. Neutralization and aeration can be combined into a single step by injecting sodium hydroxide (NaOH) into the port of the jet pump. This method of in-line aeration and treatment of acid mine drainage has been developed by the Bureau of Mines and has been field tested by the bureau as reported in the Bureau of Mines Report of Investigations 8868 by T. E. Ackman, Mining Engineer and R. L. P. Kleinmann, Supervisory Geologist, Pittsburgh Research Center, Bureau of Mines, Pittsburgh, PA, in a report entitled In-Line Aeration and Treatment of Acid Mine Drainage. This would bring the pH up to approximately 6.5. The 6.5 pH would meet the lower end of the standard that the California Regional Water Quality Control Board requires.

The above proposal is a more realistic, practical approach to abating the contaminating nature of the Walker Mine effluent than the proposed mine seal which would render the mine useless and/or the $600,000 to $1,300,000 water treatment plant proposed by Pearson and Associates. Upon the completion of the evaluation and the implementation of the launders, additional studies could commence to determine supplemental feasible means of further reducing the metal content of the effluents and to raise the pH to that of Dolly Creek. See Table 4 - Eighteen Month Walker Mine Abatement time schedule.

EIGHTEEN MONTH WALKER MINE

ABATEMENT TIME SCHEDULE

Table 4

DESCRIPTION	MAY 1985	JUNE	JULY	AUG	SEPT	OCT	NOV	DEC	JAN 1986	FEB	MAR	APR	MAY	JUNE	JULY	AUG	SEPT	OCT
Clean and stabilize main adit level (700 level)	■	■	■	■	■	■												
Test work - (Identify areas of alleged contamination)					■	■	■	■										
Construction of flume and launder system							■	■	■	■								
Monitor effluent - (At mine portal and at designated points)										■	■	■	■	■	■	■	■	■
Test work - (Additional studies to further reduce contamination)											■	■	■	■				
Design and construction of (BAT - Best available technology economically achievable system)															■	■	■	■
Monitor effluent - (At mine portals and at designated points)																		■

CALIFORNIA REGIONAL WATER QUALITY CONTROL BOARD
CENTRAL VALLEY REGION

ORDER NO. 85-033

NPDES NO. CA0080110

WASTE DISCHARGE REQUIREMENTS
FOR
WALKER MINE
ROBERT R. BARRY
CALICOPIA CORPORATION
AND THE STANDARD BULLION COMPANY, INC.
PLUMAS COUNTY

The California Regional Water Quality Control Board, Central Valley Region, (hereafter Board) finds that:

1. The Walker Mine, owned by the Calicopia Corporation and Robert R. Barry (hereafter Discharger), is a non-operating copper mine in east central Plumas County about twenty miles (32 km) east of Quincy; T24N, R4E, MDB&M.

2. A Report of Waste Discharge (RWD) was filed on 2 November 1984. The RWD indicates that the mine operator is The Standard Bullion Company, Inc., (hereafter Discharger).

3. Mining operations ceased in 1941, but acid mine drainage continues to discharge to Dollie Creek near its confluence with Little Grizzly Creek, which is tributary to Indian Creek, thence the East Branch North Fork Feather River, waters of the United States.

4. Available data indicates the water quality of the discharge to be as follows:

Constituents	Median	Range	Units
Flow	0.2	0.0 - 0.5	cfs
pH	4.8	4.4 - 6.6	-
Copper	11.0	0.23 - 69	mg/l
Zinc	0.78	0.09 - 3.2	mg/l
Aluminum	4.8	0.9 - 12	mg/l
Iron	0.8	0.01 - 1.4	mg/l

5. The Board, on 25 July 1975, adopted a Water Quality Control Plan for the Sacramento River Basin (5A) which contains water quality objectives for all waters of the Basin. These requirements are consistent with that Plan.

6. The beneficial uses of the Feather River and its tributaries are municipal, industrial, and agricultural supply; recreation; esthetic enjoyment; navigation; ground water recharge, fresh water replenishment; hydroelectric power generation; and preservation and enhancement of fish, wildlife and other aquatic resources. The aquatic resources of much of Little Grizzly Creek have been eliminated by the discharge from Walker Mine.

7. On 21 December 1983, the Regional Board filed a complaint in the Plumas County Superior Court for preliminary and permanent injunction and civil penalties against Robert R. Barry and Calicopia Corporation and Does I through XXX, inclusive. This matter has not been brought to trial.

8. Effluent limitations, and toxic and pretreatment effluent standards established pursuant to Sections 208(b), 301, 302, 304, and 307 of the Clean Water Act and amendments thereto are applicable to the discharge.

9. The discharge is presently governed by waste discharge requirements Order No. 80-058 adopted by the Board on 30 May 1980 which expires 1 May 1985.

10. The action to adopt an NPDES permit is exempt from the provisions of the California Environmental Quality Act (Public Resources Code Section 21000, et seq.), in accordance with Section 13389 of the California Water Code.

11. The Board has notified the Discharger and interested agencies and persons of its intent to prescribe waste discharge requirements for this discharge and has provided them with an opportunity for a public hearing and an opportunity to submit their written views and recommendations.

12. The Board, in a public meeting, heard and considered all comments pertaining to the discharge.

13. This Order shall serve as a NPDES permit pursuant to Section 402 of the Clean Water Act, or amendments thereto, and shall take effect ten days from the date of hearing, provided EPA has no objections.

IT IS HEREBY ORDERED that the Calicopia Corporation, and The Standard Bullion Company, Inc., and Robert R. Barry, in order to meet the provisions contained in Division 7 of the California Water Code and regulations adopted thereunder, and the provisions of the Clean Water Act and regulations and guidelines adopted thereunder, shall comply with the following:

A. Effluent Limitations:

1. The discharge shall not have a pH less than 6.5 nor greater than 8.5.

2. The discharge shall not contain more than 0.2 ml/l settleable solids.

B. Sludge and Solid Waste Disposal:

1. Sludge and/or solid wastes generated by treatment facilities or during mining exploration shall only be disposed at sites which have been approved by the Executive Officer.

C. Receiving Water Limitations:

1. The discharge shall not cause concentrations of constituents in the receiving waters to exceed the following limits:

Constituents	Units	30-Day Average	Daily Maximum
Copper	mg/l	0.02	0.05
Zinc	mg/l	0.10	0.20
Aluminum	mg/l	0.20	0.40
Iron	mg/l	0.20	0.40

2. The discharge shall not cause visible oil, grease, scum, foam, floating or suspended material in the receiving waters or watercourses.

3. The discharge shall not cause concentrations of any materials in the receiving waters which are deleterious to human, animal, aquatic, or plant life.

4. The discharge shall not cause esthetically undesirable discoloration of the receiving waters.

5. The discharge shall not cause fungus, slimes, or other objectionable growths in the receiving waters.

6. The discharge shall not cause bottom deposits in the receiving waters.

7. The discharge shall not increase the turbidity of the receiving waters by more than 20% over background levels.

8 The discharge shall not alter the normal ambient pH of the receiving water more than 0.5 units.

9. The discharge shall not cause a violation of any applicable water quality standard for receiving waters adopted by the Board or the State Water Resources Control Board as required by the Clean Water Act and regulations adopted thereunder. If more stringent applicable water quality standards are approved pursuant to Section 303 of the Clean Water Act, or amendments thereto, the Board will revise and modify this Order in accordance with such more stringent standards.

E. Provisions:

1. Neither the discharge nor its treatment shall create a nuisance or pollution as defined in Section 13050 of the California Water Code.

2. The requirements prescribed by this Order amend the requirements prescribed by Order No. 75-119, which expired on 1 March 1980.

3. The Discharger shall comply with the Standard Provisions and Reporting Requirements dated 1 October 1984 which are part of this Order.

4. The Discharger shall comply with the attached Monitoring and Reporting Program No. 85-033 as ordered by the Executive Officer.

5. This Order expires on 1 February 1990 and the Discharger must file a Report of Waste Discharge in accordance with Title 23, California Administrative Code, not later than 180 days in advance of such date as application for issuance of new waste discharge requirements.

6. In the event of any change in control or ownership of land or waste discharge facilities presently owned or controlled by the Discharger, the Discharger shall notify the succeeding owner or operator of the existence of this Order by letter, a copy of which shall be forwarded to this office.

I, WILLIAM H. CROOKS, Executive Officer, do hereby certify the foregoing is a full, true, and correct copy of an Order adopted by the California Regional Water Quality Control Board, Central Valley Region, on 25 January 1985.

William H Crooks

WILLIAM H. CROOKS, Executive Officer

12/24/84:EZC:gs

Attachments