Some Practical Aspects of Open-Pit Dewatering at Pine Point

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Abstract

Open-pit dewatering at Pine Point Mines Limited has progressed satisfactorily to date, but it will be a fairly complex problem and major expense in the future due to varying ground-water conditions and the increase in deeper dewatering requirements.

A general description and history of all facets of open-pit dewatering is included, and possible future problems in scheduling dewatering, deep dewatering design and mining are discussed.

The economics of open-pit dewatering are discussed and it is concluded that the system now used will continue to be satisfactory if: (1) groups of proximal orebodies are mined and dewatered simultaneously; and (2) the optional use of sump pumps continues.

Because of high dewatering costs and limited power supply, it is concluded that, in future years, mining schedules and forecasts may be determined entirely by open-pit dewatering requirements.

Introduction

LOCATION AND HISTORY

PINE POINT MINES LIMITED is located in the Northwest Territories of Canada, approximately 700 road miles north of Edmonton, Alberta, 6 miles south of Great Slave Lake and 60 miles east of Hay River.

Ore production began in 1964. Due to the fact that several projected and operating pits had ore occurrences beneath the water table, perimeter dewatering systems were initiated by Pine Point Mines Limited on the advice of Leggette, Brashears and Graham of New York. Perimeter dewatering systems have proved to be successful at all ore occurrences mined to date.

TOPOGRAPHY

The land is flat and swampy, with a gentle northwesterly slope toward the southern shore of Great Slave Lake. The elevation at the townsite is approximately 700 feet above mean sea level (amsl) and Great Slave Lake is ±515 feet amsl. This vertical drop takes place over approximately 6 miles.

Swamp, muskeg and low gravel ridges (representing old lake beaches) are the main topographic features.

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After three years with Cominco Ltd. as geologist in charge of open-pit dewatering at Pine Point Mines, he worked for three years in Western Australia on various groundwater projects with the State Government and a hydrogeological consulting firm.

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General Geology and Hydrogeology

GENERAL GEOLOGY

The ore deposits of the Pine Point area are closely associated with a Devonian barrier complex. The main rock types are medium to coarse-grained recrystallized dolomites, but they also include limestones, shales, clays, and mud and sand seams. "Due to a gently southwest-dipping strata, the barrier continues at depth west of the Buffalo River and can be followed southwestward into northern British Columbia"[10].

The barrier complex consists of various facies, representing the following environments: immediate forereef; reef core; lagoonal; and tidal flat. In lithology, the barrier varies from a coarse crystalline dolomite which transcends the above facies to a medium-grained, sandy-textured dolomite. The occasional limestone "window" permits an environmental interpretation. The associated forereef and basinal facies vary lithologically from argillaceous, sandy-textured dolomites, fine-grained limestone and dolomites to shales. The facies behind the barrier are fine- to medium-grained dolomites and evaporites.

The main geological structures, other than the gentle western dip of the strata, are the minor folds and major amounts of fracturing that have occurred. Folding is generally associated with differential compaction, gentle flexing and differences in rates of subsidence in the original sediments. Faulting and fracturing in the Devonian are related to tectonic movements in the basement and follow two main directions — northeast to southwest and east to west.

Large, sand-filled slump and solution sinkholes are prominent throughout the area, and are considered to be of two distinct ages: Devonian and Pleistocene. Devonian sinkholes were formed as part of a karstic environment when the barrier lithologies were exposed. They are generally filled with a fine, spherical silica sand. Pleistocene sinkholes are thought to be a result of solution and erosion of limestones and dolomites by glacial outwash, and are generally filled with granite, limestone and dolomite boulders in sand, gravel and clay.

A detailed description of the geology of the Pine Point district is given by Skall (1975)[10].

GENERAL HYDROGEOLOGY

The varied lithology of the Pine Point complex makes the distinguishing of separate aquifers very difficult. It is felt that, in specific areas, most saturated rock units contribute to ground-water withdrawals, but one or two units can be distinguished as the major aquifers. Linked porosity is very low, with the result that ground water flows mainly through fractures, faults and other secondary structures. Fracturing is so extensive in most rock units that they respond as isotropic aquifers to ground-water withdrawals.

Prior to pit dewatering, ground-water movement was to the northeast toward Great Slave Lake. Since pumping began, elongated troughs have been formed in the water table of the central portion of the area, as shown in Figure 1.

Table 1 is a summary of drawdowns to date, with average withdrawal rates.

Precipitation is very low in the area (13 in. annually at Fort Smith) and, due to the thick overburden covering the glacial till, less than 25% of the precipitation is thought to recharge the aquifers[4,5]. Most of the recharge probably takes place during spring thaws and summer storms. The presence of numerous sinkholes in the southern portion of the property probably represents one major area of recharge.

Aquifer conditions range from unconfined (water-table conditions) to confined (flowing artesian conditions) (Fig. 1). In the central and southern portions, aquifer coefficients and water-level data indicate the presence of unconfined conditions. To the north, ground water becomes confined and many areas of artesian flow are present. Leaky artesian conditions also exist in large areas of the northern portion of the property, mainly because of the presence of interfingering clay, shale and mud units (Watt Mountain and Buffalo River shales). Water is contained in and above these low-permeability units, and when pumping begins water slowly leaks into the main aquifer, influencing projected drawdowns. Confined conditions appear to be a result of:

(1) drop in elevation of land surfaces;
(2) confining effect of thick, frozen, clayey overburden and increased shale units in the northern half of the property;
(3) higher-elevation recharge area (i.e. southern half of property).

Transmissivities (transmissivity, T, is defined as flow in gallons per day through a vertical strip of aquifer 1 foot wide, extending the full saturated height of the aquifer, under a hydraulic gradient of 1 foot per foot) are generally wtihin the range of 30,000-70,000 USgpd/ft, but notable exceptions are the townsite (17,000), the millsite area (90,000-100,000) and the K-62 pit (90,000). Coefficients of storage (S) (defined as the volume of water an aquifer will release from or take into storage per unit surface area of aquifer per unit change in head perpendicular to the aquifer surface) confirm varying aquifer conditions in the areas mentioned above. Ground-water conditions, aquifer characteristics, and values of T and S are given for several pits in Table 2.

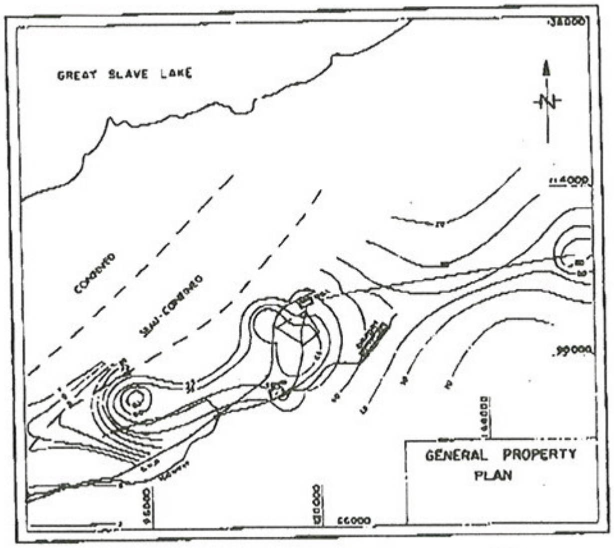

FIGURE 1 — A regional drawdown contour map (ft).

TABLE 1 — Summary of Dewatering Results (December 1975)

Pit	Transmissivity (USgpd/ft)	Average Pumping Rate (USgpm)	Approximate Total Days Pumping	Maximum Drawdown at Pit Center	Remarks
J-44	60,000	6,280	1,370	98 ft	Completed
O-42	72,000	3,880	1,370	85 ft	Completed
X-15	35,000	3,780	1,980	123 ft	No pumping at present; 27 ft regional drawdown from W-17
K-57	41,000	5,300	2,190	152 ft	Completed
A-70	65,000	1,000	45	19 ft	Regional drawdown from K-62
W-17	69,000	14,270	1,217	176 ft	27 ft regional drawdown from X-15
K-62	69,000	4,830	150	115 ft	80 ft regional drawdown from K-57
R-61	37,000	1,220	305	45 ft	6 ft regional drawdown
M-40	—	1,220	180	65 ft	No pumping at present

USgpd = U.S. gallons per day
USgpm = U.S. gallons per minute

Open-Pit Dewatering

AQUIFER TEST ANALYSIS

General methods of aquifer test analysis and the scheduling of the dewatering of open pits at Pine Point have previously been described by Calver[7] and Brashears and Slayback[5]. They will be dealt with very briefly here.

Methods used for analysis have involved the "straight-line method of Cooper and Jacob"[8] (a modification of the non-equilibrium formula by Theis[18]), the distance drawdown method originated by Thiem[17], the method of matching to modified Theis-type curves[20], and the Leaky Artesian method of Jacob[12], and Hantush and Jacob[11].

(1) Cooper and Jacob

To apply this method, drawdown in specific test holes is plotted against time (logarithmic scale) on semi-logarithmic paper (Fig. 2). After a certain time, the plotted data will form a straight-line trend. This trend is classified as a "storage depletion" trend, and can be used to predict long-term drawdowns at varying distances from the pumping well, and thus at pit centers. The general equations used are:

TABLE 2 — Summary of General Hydrogeology, Pine Point, Alberta

Pit	T (USgpd/ft)	S	Main Aquifer	Ground-Water Conditions
X-15	35,000	0.006	Fine-med., well-fractured, porous dolomite	Water table to confined locally
J-44	60,000	0.025	Coarse, well-fractured dolomite (Presqu'ile) Fine-med., porous fractured dolomite (Pine Point Formation)	Water table
O-42	72,000	0.035	"	"
N-42	54,000	0.016	"	"
K-57	41,000	0.001	"	Semi-confined
A-70	65,000	0.002	Fine to med.-grained, porous fractured dolomite (Pine Point Formation) Well-fractured limestone & dolomite (Presqu'ile)	Leak artesian (confined)
W-17	69,000	0.05	Fine to med., fractured dolomite (Pine Point Formation)	Water table
K-62	69,000	0.05	Coarse, well-fractured dolomite (Presqu'ile) Fine to med., fractured dolomite (Pine Point Formation)	Water table
Hinge Zone	90,000	0.03	"	Water table
R-61	37,000	0.001	"	Semi-confined
J-70	70,000	0.002	"	Semi-confined

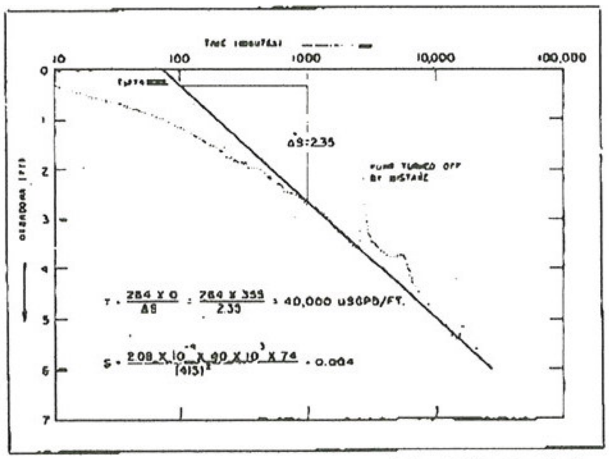

FIGURE 2 — Values of T and S obtained for WH3-K57 during the K-57 pump test.

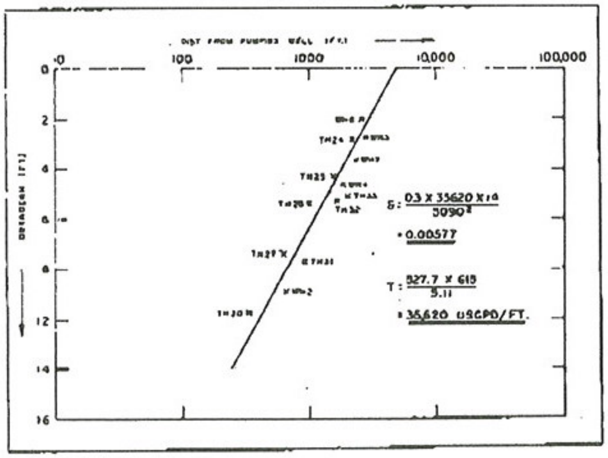

FIGURE 3 — Values of T and S obtained from the distance-drawdown plot of test holes in the X-15 pump test (after 14 days). (After Calver.)

$$T = \frac{264Q}{\Delta s} \quad \ldots\ldots\ldots\ldots\ldots \quad (1)$$

$$S = \frac{2.08 \times 10^{-4}\, T t_o}{r^2} \quad \ldots\ldots \quad (2)$$

where

T = coefficient of transmissivity in U.S. gallons per day per foot.

Q = total discharge (U.S. gallons/min.) of pumping well(s).

Δs = slope of straight-line trend over one log cycle of time.

S = coefficient of storage, as a decimal fraction.

t_o = intersection of straight-line slope with zero drawdown axis, in minutes.

r = distance of observation hole to pumping well in feet.

(2) Thiem

A method used concurrently with the cooper and Jacob method is the distance-drawdown method of Thiem. After the beginning of a storage depletion trend, simultaneous drawdown measurements are taken in all test holes available and drawdown versus distance from pumping well (logarithmic scale) is plotted on semi-logarithmic paper (Fig. 3). If the aquifer is ideal, these points should form a straight line and the following equations can be applied:

$$T = \frac{528Q}{\Delta s} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (3)$$

$$S = \frac{0.3\, Tt}{r_o^2} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (4)$$

where

t = time of simultaneous measurements (after pump started) in days.

r_o = intersection of straight line with zero drawdown axis (in feet).

All other symbols are as previously described.

(3) Modified Theis Curve

In this method, drawdown is plotted against time for each test hole on log-log paper and is matched to a theoretical type curve. This is a modification of the original Theis type curve derived from the Theis equation of:

$$s = \frac{Q}{4\pi T} \int_u^\infty \frac{e^{-u}\, du}{u} \quad \ldots\ldots\ldots\ldots\ldots \quad (5)$$

where

$$u = \frac{r^2 S}{4Tt} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (6)$$

e = base of the Naperian logarithm
and W(u) is the integral expression in equation (5) and is known as the well function (tables are available).

The original Theis type curve was a plot of W(u) vs u.

Using time in minutes and U.S. unit notation, equations (5) and (6) reduce to:

$$s = \frac{114.6Q}{T}\, W(u) \quad \ldots\ldots\ldots\ldots\ldots\ldots \quad (7)$$

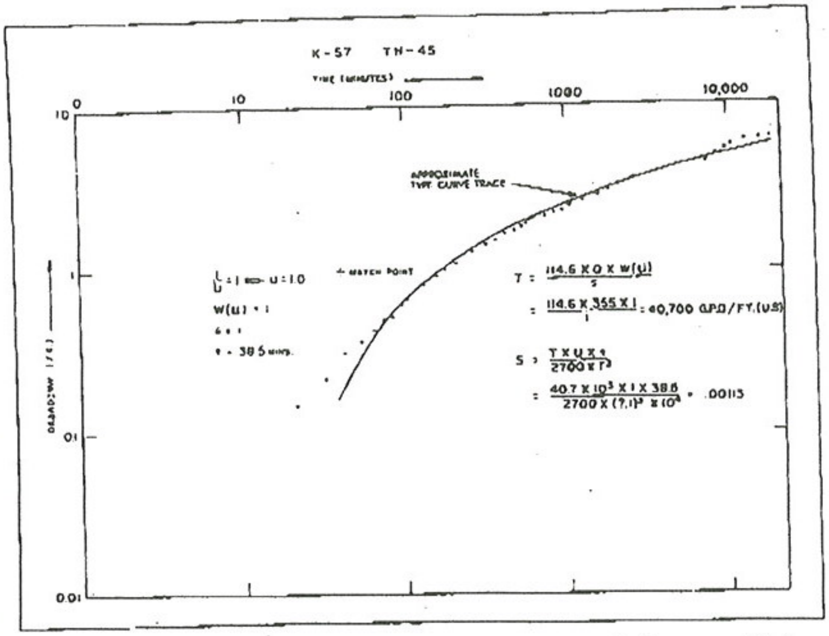

FIGURE 4 — Field data from the K-57 pump test (TH45) matched to a modified Theis type curve.

$$S = \frac{T u t}{2700 r^2} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (8)$$

where t = time in minutes.

Walton plotted W(u) versus 1/u and obtained a type curve which was a mirror image of the Theis type curve. Field values of s and t can be plotted directly on log-log paper and a match point obtained. From the match-point values of W(u), 1/u, s and t can be inserted in equations (7) and (8) and values of T and S obtained. Figure 4 shows values obtained for a test hole during the K-57 pump test.

(4) Leaky Artesian Conditions

An aquifer test, conducted in pit A-70 in September of 1970, indicated that, in the confined area of the property, possible leaky artesian conditions do exist, probably related to the large number of semi-permeable shale, clay and mud units in the north (Fig. 5).

The method consists of plotting drawdown vs time on logarithmic paper. After a certain time, the field data plot will deviate from the type curve. Walton drew a set of curves indicating varying amounts of leakage.

Hantush and Jacob derived the following equations to calculate aquifer constants:

$$s = \frac{114.6Q}{T}\, W\left(u, \frac{r}{B}\right) \quad \ldots\ldots\ldots\ldots\ldots \quad (9)$$

$$u = \frac{2700\, r^2 s}{Tt} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (10)$$

$$\frac{r}{B} = \frac{r}{\sqrt{\dfrac{T}{\dfrac{K^1}{b^1}}}}$$

where:

$W\left(u, \frac{r}{B}\right)$ = the well function for leaky aquifers.

B = the leakage factor.

K^1 = vertical hydraulic conductivity of leaky beds (USgpd/ft²).

b^1 = thickness of confining strata (ft).

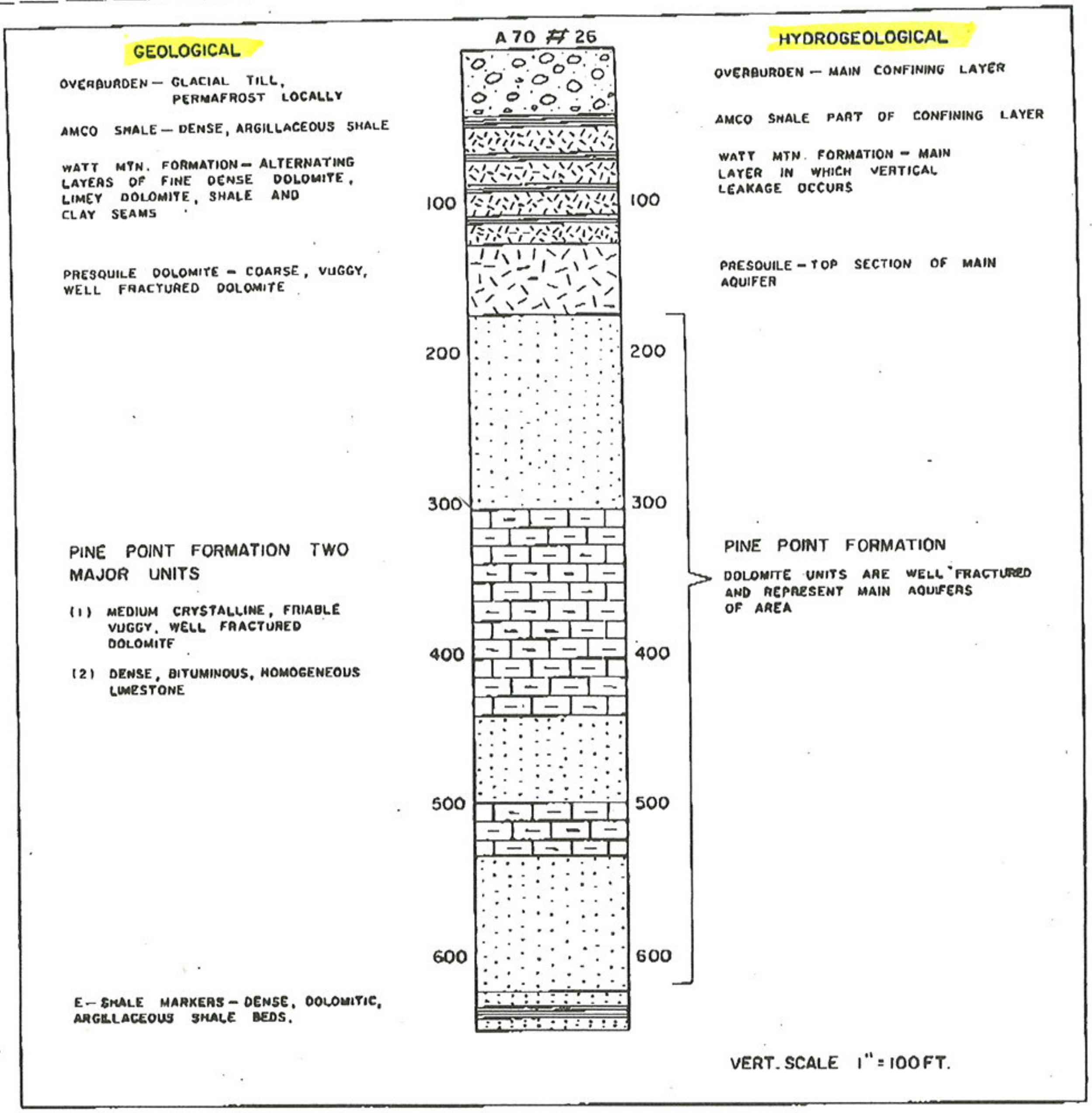

FIGURE 5 — Lithographic log showing geological and hydrogeological descriptions for the A-70 pit area — a leaky artesian area. Dewatering is required to the 200-ft level.

Matching the plotted field data as closely as possible to one of the type curves, a match point is chosen which will give values of s and t (from the field data coordinates) and W(u, r/B) and 1/u (from the type curve coordinates).

These values are then inserted into equations (9), (10) and (11) and values for T, S, K^1, and K^1/b^1 (leakage coefficient) can be obtained.

Figure 6 shows a typical calculation from the A-70 pump test.

SCHEDULING DEWATERING

A dewatering schedule must be drawn up in close conjunction with the mining schedule and possible pit planning changes. Calver, and Brashears and Slayback have described general methods used at Pine Point. A brief review will be presented here.

At present, all the wells required to dewater a pit are drilled at one time; this dewaters the top benches far in advance of their mining. As the rate of drawdown decreases, due to a greater volume in the cone of depression in the water table around the pit, the mining rate increases relative to the dewatering rate and ideally the bottom pit bench will be mined shortly after it is dewatered. Originally, wells were added every year (i.e. the maximum number required were not drilled initially) to maintain a dewatering lead over the mining schedule.

Two basic types of methods can be used for calculating the number of pumps required to dewater certain benches of a pit:

(a) graphical (using Thiem and Cooper and Jacob); and

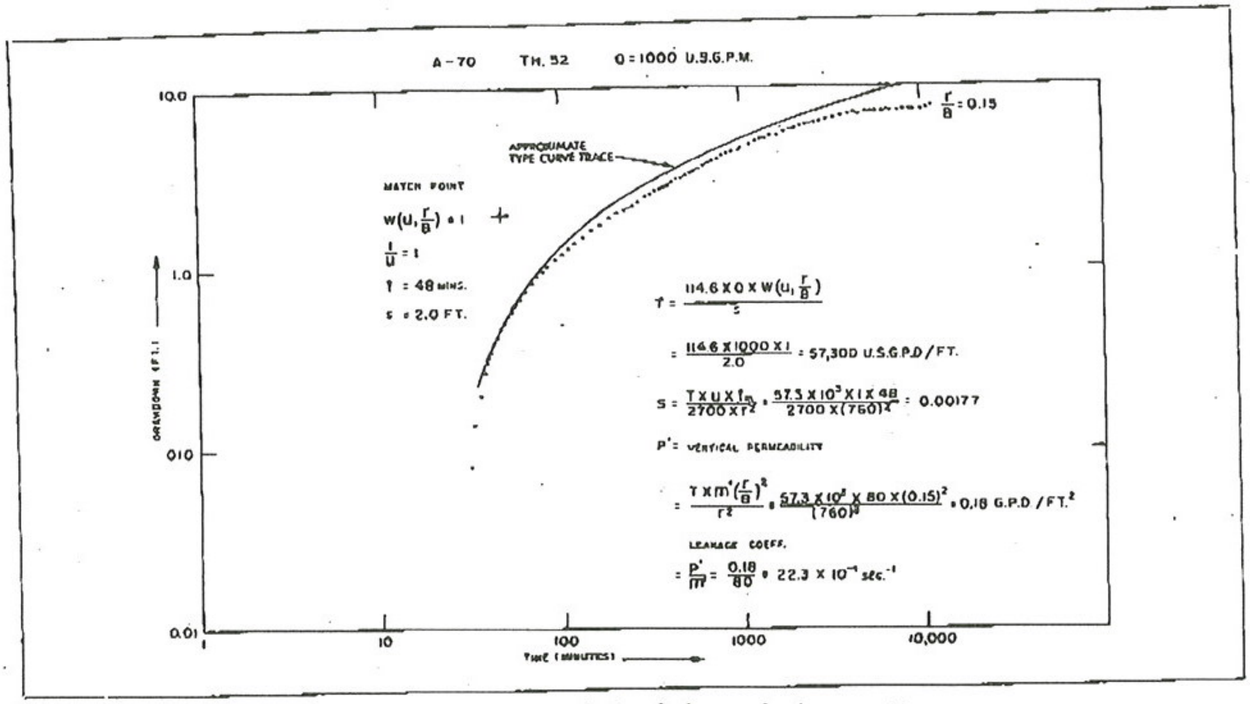

FIGURE 6 — Field data for the TH52-A70 pump test matched to leaky artesian type curves.

(b) by desk computer (using the basic Jacob modification for steady-state conditions).

(a) Graphical Methods

A combination of the Thiem (distance-drawdown) and the "storage depletion trend" time-drawdown of Cooper and Jacob is used.

The storage depletion trend graph shows drawdown versus time; however, a certain length of time is required after pumping begins to establish a straight-line relationship. At Pine Point, a steady-state condition is assumed to exist after 10-14 days. The distance drawdown method will therefore give amounts of drawdown at varying distances from the pumping well(s) after a given length of time. Rather than calculate the drawdown for each length of time required, the data are transferred to the drawdown-time graph, and the drawdown at any time can be read directly.

The first step is to calculate a family of straight lines representing different values of discharge (Fig. 7).

By rearranging equations (3) and (4), we see that:

$$\Delta s = \frac{528Q}{T} \qquad (12)$$

$$r_a^2 = \frac{0.3\ Tt}{S} \qquad (13)$$

where t = elapsed time since pumping began in days and all other symbols are as previously described.

If we then choose three points in the pit center that represent the level of deepest dewatering required, the total drawdown at each point can be calculated by totalling the individual drawdowns from each well after the same length of time (at least 10 days).

Each of the drawdown values (either total or individual) can then be transferred to a time drawdown graph, all points being plotted on the same time line.

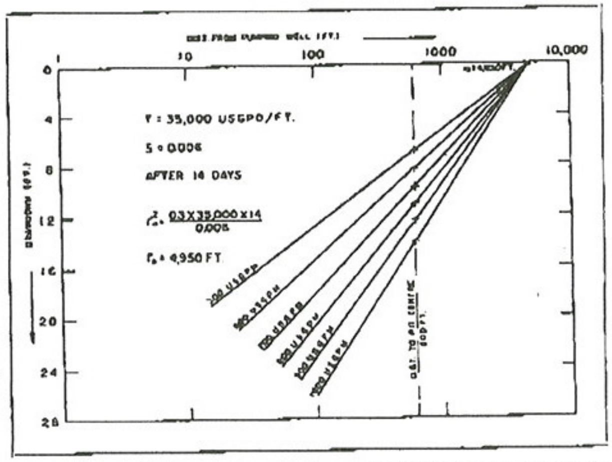

FIGURE 7 — Distance-drawdown graph at variable discharge.

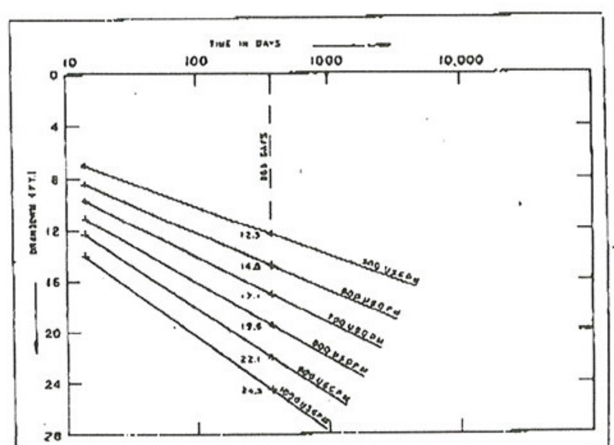

FIGURE 8 — Time-drawdown graph at the pit center for variable discharge.

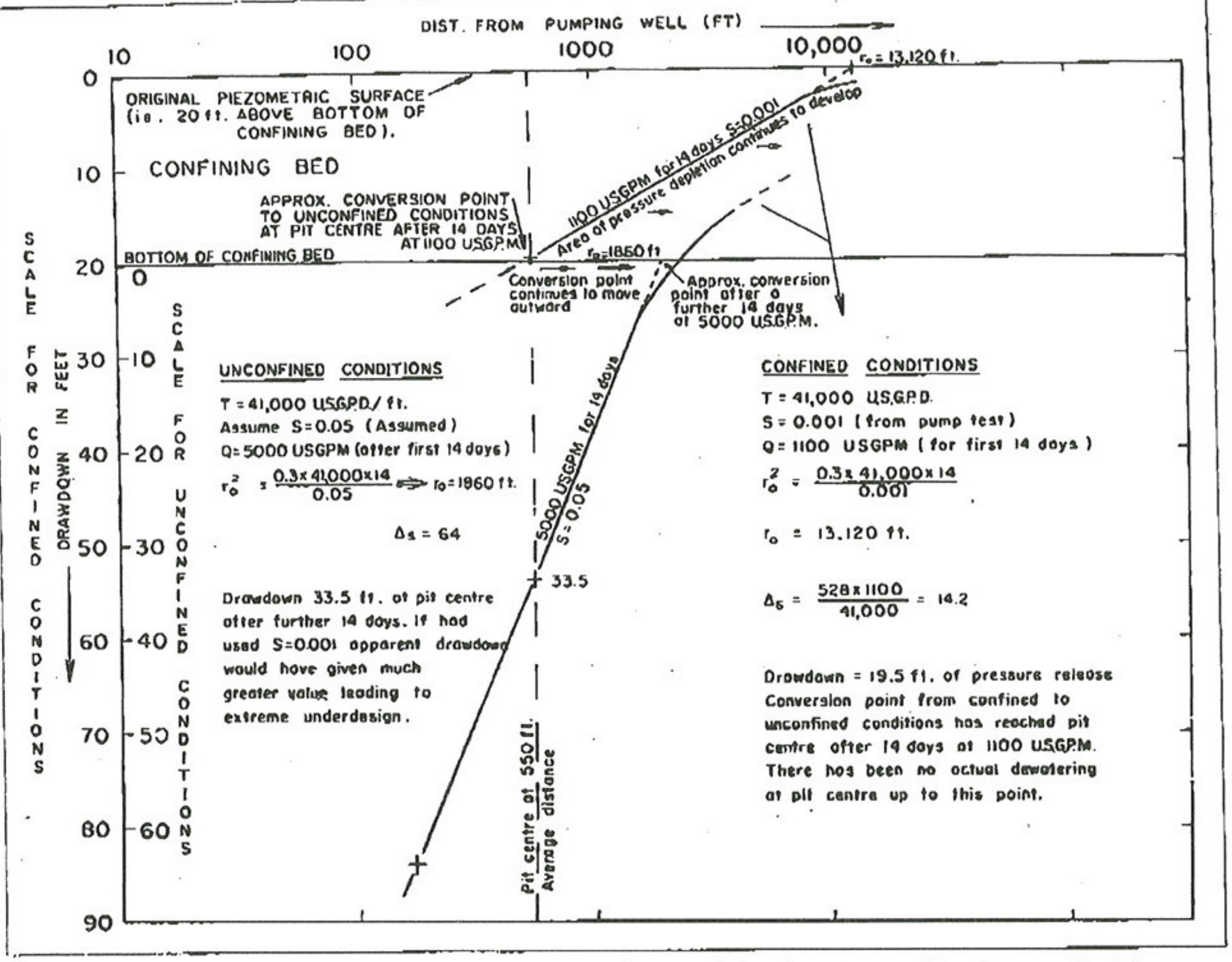

FIGURE 9 — Distance-drawdown calculation for estimating theoretical drawdown in transition from confined to unconfined conditions when the unconfined coefficient of storage is not known. Values are transferred to the time-drawdown plot for long-term predictions.

From equation (1):

$$\Delta s = \frac{264Q}{T} \qquad (14)$$

and therefore the slope of the time-drawdown lines can be calculated (it is ½ the slope of the distance-drawdown lines). Each line representing a different Q is then drawn and values of drawdown after any given length of time can be read (Fig. 8).

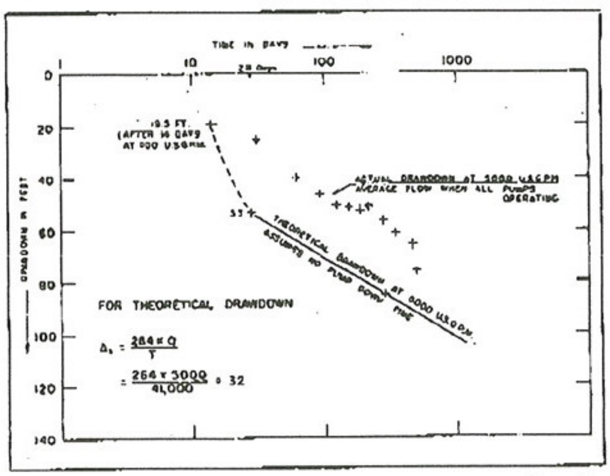

FIGURE 10 — Theoretical storage-depletion trend for the K-57 pit versus actual drawdown at the pit center.

A certain amount of trial and error is involved before the correct discharge (i.e. number of pumps) is calculated to meet all mining schedule requirements.

(b) Computer Method

A computer program has been developed using the Jacob equation:

$$s = \frac{264Q}{T} \log_{10} \frac{0.3\,Tt}{r^2S} \qquad (15)$$

where:

Q = discharge in USgpm.
T = transmissivity in USgpd/ft.
t = elapsed pumping time in days.
r = distance from pumping well in feet.

The program has proved invaluable for economic studies concerning dewatering, and it is used in open-pit dewatering scheduling where long laborious graphical work has been involved. The method is accurate only for calculating drawdowns after at least 10-14 days of pumping.

THEORETICAL AND ACTUAL RESULTS

In applying aquifer analysis methods, certain assumptions have to be made concerning the hydraulic

TABLE 3 — Comparison of Well Drilling Rates and Costs

Year	Method Used	Hole Diameter	Total Footage in Program	Average Cost/Ft ($)	Average Drilling Rate (time on bottom)
1968	Mud	12¼″	10,895 ft	19.40	9.5 ft/hr
1969	Mud, air & ground water	12¼″	10,200 ft	19.15	14 ft/hr
1970	Surface water, air & ground water	12¼″	7,755 ft	16.80	21 ft/hr
1971	Surface water & air	12″¼ 15″	2,910 ft 935 ft	17.10 17.50	28 ft/hr 22 ft/hr
1972	"	15″ (17″ at top)	6,790 ft	N.A.	17 ft/hr
1973	"	15″	6,055 ft	N.A.	19.5 ft/hr
1974	Surface water & air + foam	15″	7,350 ft	Approx. $25	12 ft/hr
1975	"	15″	8,165 ft	N.A.	14 ft/hr

N.A. — not available.

and geologic properties of the aquifer (i.e. the aquifer is of infinite extent, homogeneous, isotropic, etc.). These properties do not entirely apply in most cases, but, as faulting and fracturing are very extensive, ground-water movement and drawdowns can be estimated using the previously described methods. Variations in these assumptions, together with the more real problems of power failures, pump failures, blast damage and general maintenance, have led to a safety design of one or two additional pumps over normal requirements.

An example is shown of the K-57 open pit (an originally semi-confined aquifer) in Figures 9 and 10. After pumping for one year, actual drawdowns were approximately 20 feet less than predicted drawdowns. This was mainly due to power failures, pump repairs and general "down" time for pit electrical work, and also to the variation of the storage coefficient from the assumed value of 0.1 (the K-57 pump test did not manage to lower water levels below the confining layer). Two additional pumps would have increased drawdowns closer to theoretical values, and thus allowed both for pump "down" time and variations within the groundwater regime.

Two methods will be mentioned later in this report which allow a more accurate calculation of a water table storage coefficient.

GENERAL MONITORING OF PIT DEWATERING

A continuous dewatering record is kept for each area with pump installations. The record consists of weekly observation-well water levels, pumping rates and pump performance data. Evaluation of the records provides a check on dewatering progress, and a basis for changes in capacity and design.

Numerous test holes located within and around the pits can be used to compile water-contour maps in the immediate area of the pit. The maps show the status of dewatering levels and serve as guides for planning and design. Regional contour maps, as shown in Figure 1, are used to predict water levels in areas where no test holes exist and also show regional effects between areas that are being dewatered.

Well Drilling

Two main sizes of wells have been drilled at Pine Point — 12¼-in. and 15-in. The 15-in. wells have been drilled as a result of increased problems with pumps in crooked wells, and also to allow for the installation of larger-capacity pumps. Table 3 summarizes drilling costs, rates and methods.

There have been four main methods used in drilling wells, each being an improvement over the previous method.

1. Drilling with Mud

This was the original method used at Pine Point, but it has many disadvantages in water-well drilling. Drilling mud can penetrate and thus seal many water-bearing fractures and channels. This type of drilling can produce wells of poor yields, and damage to the water-bearing zones of the well may be permanent. Well development is usually costly in this case.

2. Drilling with Mud, Air and Ground Water

Mud is used to drill the well into the water table. The mud line is then connected to a source of compressed air, and a mixture of air and ground water is used to bring up the cuttings.

Drilling rates for this method average the same as with straight mud, but it has the advantage that the well is cleaned as it is drilled.

Originally, three 100-psi, 600-cfm compressors were used to drill with air. It was found that 100 psi had limitations to a depth of approximately 400 feet (depending on depth to water table). In the 1970 well drilling program, a 250-psi, 1200-cfm trailer-mounted compressor was used and the over-all footage cost was reduced (Table 3).

3. Drilling with Surface Water (from a drilling sump) Initially and Then Using Air and Ground Water

Near the end of the 1970 drilling program, it was discovered that in most cases (except sinkholes, unconsolidated formations, etc.), mud was not needed to drill the upper section of the well. At most locations, the overburden will mix with straight water from the sump and form a very thin mud. In the bedrock part of the upper section of the well, the contractor's pump (350 gpm) had enough velocity to lift the drill cuttings from depths below the water table. Compressed air and ground water were used to complete the well.

The economic advantages of this method are as follows: no expense for mud materials; and little development time is required, as the well is reasonably clean.

4. Drilling with Surface Water and Then Using Surface Water and Air to the Total Depth of the Well

This method was developed in the 1971 well drilling program. Surface water (from the sump) is used to drill approximately 150-200 feet, and then air and surface water are used at the same time. Drilling into the water table before switching on the air prevents a continuous loss of sump water volume. When the water table is reached, the compressor is put into operation.

Generally, faster drilling rates exist with this method, as cuttings are removed from the face of the bit faster using air and water rather than straight air. Using conventional air drilling, it is probable that air is removing the cuttings with little water to aid in the lifting until the cuttings are forced above the bit. Using air and surface water, there is always some water (200 gpm when the pump is idling) mixed with the air; this increases the rate and volume of cuttings removed. A further development in this method is the injection of drilling foam to further enhance cuttings removal.

After reaching the chosen depth (usually 150 feet), the drill-mud pump is reduced to idling (200 gpm) and the air compressor is operated at full capacity (250 psi, 1200 cfm). The valve connecting the two lines is left fully open. The mud pump cannot be operated at full volume (350 gpm), as this causes the compressor to overheat. On completion of the hole, the pump is turned off and the well is surged until clean water is being produced.

WELL HOLE DEVELOPMENT

Other than using primary development methods, such as airlifting ("blowing") wells after completion, very little secondary development to increase well yields has been attempted. The secondary methods attempted include:

(1) polyphosphate treatment to remove drilling mud;
(2) explosive development; and
(3) scouring by acidization.

Scouring by acidization has been successful in one well in the X-15 pit (increasing the well yield from 50 to 200 USgpm).

Explosive fracturing development increased the yield from 50 to 400 USgpm in a W-17 pit well. The charge used involved 100 lbs of slurry explosive and two strings of detonating cord.

Deep Well Pump Problems

Experimentation with 15-in. well drilling developed mainly as a result of deep well pump maintenance problems. Originally, well surveying was used to determine the type of pump (submergible or lineshaft) to be installed in a well. Lineshaft pumps were used in the straighter wells; submergible pumps were used in wells that were very crooked.

The initial type of pump used for dewatering was the 3600-rpm lineshaft. These pumps have had moderate success and are now being phased out of operation. Any misalignment problems are magnified at this high speed and considerable maintenance problems have been encountered with bearing, oil-column and lineshaft components.

In an attempt to overcome these problems, submergible pumps were used. Submergible pump usage has varied greatly since 1969, but it has generally decreased in the last three years. Applications in crooked holes have not been entirely successful, mainly because the wells are completed "open hole" and damage to the electric power cable is frequent. Submergible pumps are susceptible to water-level surging and cavitation, causing leaking seals (because of uneven thrust conditions) and damaged motors. In wells where the majority of the flow is entering above the pump, the motor may not be properly cooled and overheating problems can develop. Submergible pump motors are also sensitive to power fluctuations, and expensive to repair. At present, 3600-rpm submergible pumps are being phased out of operation at Pine Point because of high maintenance costs.

In 1971-1972, two important events overcame the majority of the above-mentioned problems. The drilling of 15-in. wells was very successful and 1800-rpm lineshaft pumps were introduced in the dewatering system. With increasing drilling experience, it has been possible to drill 15-in. wells at footage rates and prices nearly comparable to those of the 12¼-in. well holes (Table 3). In addition, a shock-absorbing device ("shock sub") installed in the drilling pipe string has eliminated crooked wells. The 1800-rpm lineshaft pumps seem very suitable for pit dewatering, as they are heavier duty, slower speed and able to pump larger volumes of water than a 3600-rpm pump of equivalent horsepower.

The major types of pumps being used in open-pit dewatering are:

(1) 1800-rpm lineshaft — oil lubricated (125, 150 and 200 hp) — 63% of total;
(2) 3600-rpm lineshaft — oil lubricated (60 and 100 hp) — 23% of total;
(3) 3600-rpm submergible (75 and 100 hp) — 14% of total.

Reducing the use of 3600-rpm lineshaft pumps has been an important step in lowering pump maintenance costs. This, together with successful 1800-rpm lineshaft use, has reduced pump problems. For deep dewatering requirements (greater than 150 ft of drawdown), experimenting with larger-capacity pumps (200 hp, 2000 USgpm) has been only moderately successful. Pumps of this size represent such a large proportion of groundwater withdrawals in a pit area that pump failure causes a major reduction in dewatering rates. An optimum pump size for Pine Point requirements is in the range of 100-125 hp, because most wells will efficiently yield 1000-1200 USgpm.

Due to freight costs and high repair costs for submergible and lineshaft pumps, rebuilding of pump components and motors has been attempted at Pine Point. For lineshaft pumps, a good technology has been developed and most repairs (including rewinding motors) are completed on site. Submergible pumps which have more specialized components are impossible to repair on site, and repair costs continue to be excessive.

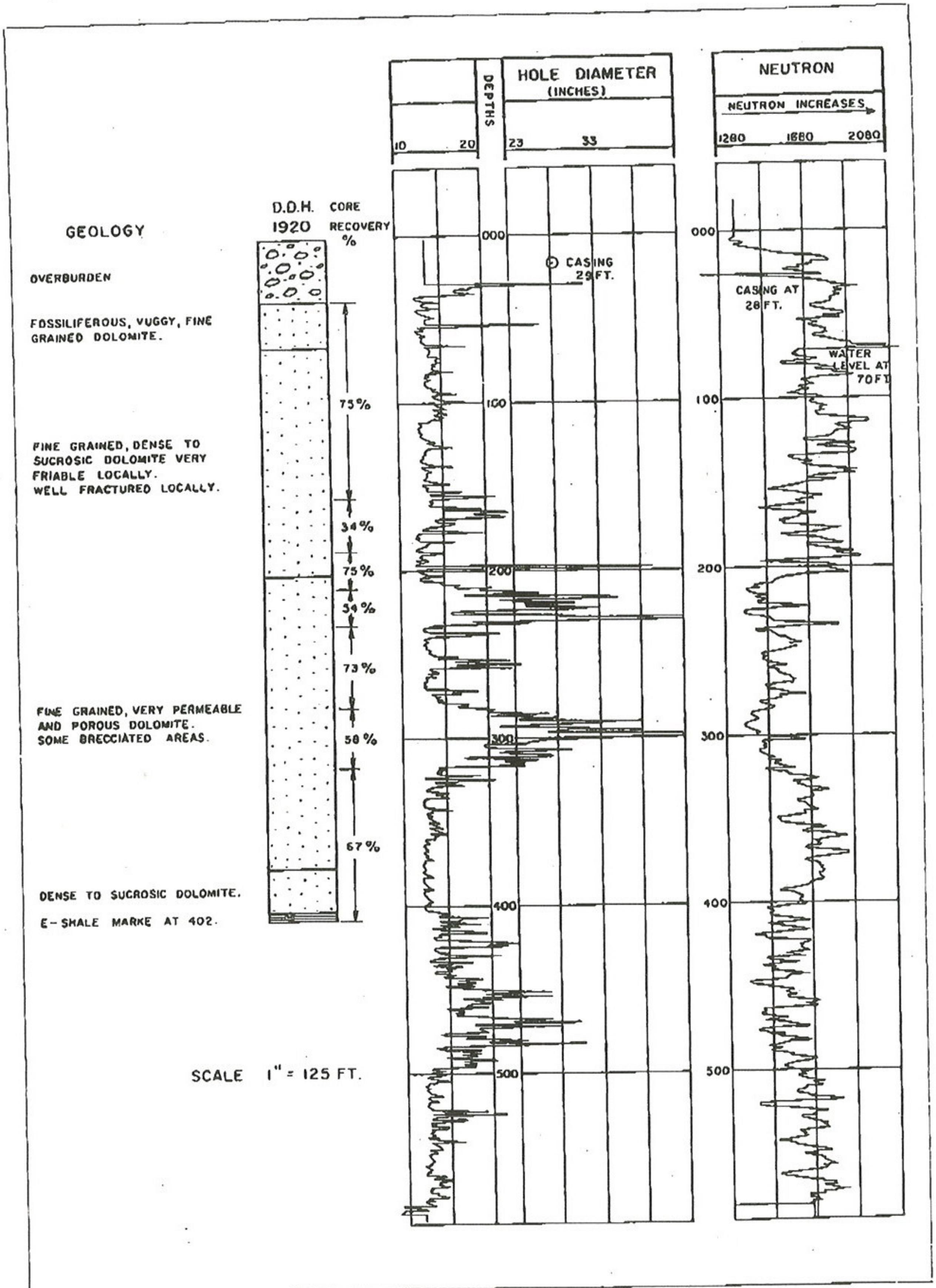

FIGURE 11 — Comparison between the caliper and neutron log for WH5-W17 and the geological log of DDH 1920 (1000 ft north).

Miscellaneous Problems

LOW-YIELD AREAS

Generally, pump size selection for individual wells has not posed a problem, mainly because 80% of the wells at Pine Point will yield 800 USgpm or greater. The remaining 20% of low-yield wells, however, tend to group together in specific pit areas (e.g. X-15 and K-62), and thus threaten effective dewatering because of low withdrawal rates. Figure 11, a caliper and neutron log of the WH5-W-17 pit, shows an interesting comparison in a region of fairly homogeneous fine-grained dolomite (where it is particularly difficult to pick out water-bearing zones while drilling). In areas of low neutron activity, high amounts of caving are indicated on the caliper log and, from a diamond drill hole nearby, areas of low core recovery are indicated in these same areas. As this caving is directly related to the amount of fracturing and friability, these zones are probably large-volume flow zones. The neutron log also indicates increased porosity in these areas. In order to maintain the most continuous flow to the intake of the pump (particularly important in the case of submergible pumps), it can be sited next to one of these zones. The choice of the zone depends on depth dewatering requirements.

In low-yield areas, therefore, these logs show areas of possible yield and favourable intervals for development attempts where little geological data is available because of sinkhole areas, etc. Also, caliper-neutron logs will show wells with no major aquifer zones and, therefore, lead to the installation of a small-capacity pump.

AREAS OF DEEP DEWATERING

Open pits that require deep dewatering (as much as 350 ft of drawdown) pose special problems. It is especially important in these areas that the dewatering system be as efficient as possible to keep substantial operating and maintenance costs within economic limits. In a small but very deep pit (carrot-shaped), for example, an overcrowding of wells around the pit perimeter may cause excessive drawdown interference between wells and reduce the effectiveness of the system.

Two important solutions here are:

(1) the drilling of wells closer to the pit center; and
(2) the use of sump pumps.

As is obvious from the preceding graphs, the closer the wells are to the pit center the more drawdown they produce at that point. Drilling wells on the berms of either the first or second bench of an open pit will:

(1) slow drilling rates as bedrock is encountered at surface — in order to install well casing in a reasonable time, a blasthole drill is used to complete the top section of the well;
(2) require a continuous supply of water (because of lost circulation) until the depressed water table is reached;
(3) possibly lead to a change in standard pit design to allow wider berms for drilling-rig access.

Pumps installed within the pit perimeter require:

(1) extensive protection from blasts;
(2) power from the perimeter power line; and
(3) pump discharge lines which drain quickly in winter.

A sump pump was used to successfully dewater the bottom bench of the K-57 open pit. The lowest ore level of this pit was becoming uneconomic due to excessive dewatering costs, and the use of a sump pump allowed approximately half of the perimeter pumps to be removed.

In an area with such severe climate and highly transmissive rock, a sump pump must be used with caution. Extensive power outages can be disastrous, especially in winter, and safety factors such as wall stability and pit flooding must be considered. These problems are minimized by:

(1) limiting the dewatering use of a sump pump to the lowest bench of an open pit; and
(2) operating a few perimeter pumps simultaneously to reduce recharge rates in the event of power outages.

Assuming that the perimeter of an open pit is not large enough to accommodate enough wells to efficiently dewater to the depth required, a deep dewatering sequence is outlined below:

(1) drill as many perimeter wells as is feasible;
(2) depending on the amount of dewatering still required, either drill enough well holes inside the pit to complete the dewatering or install a sump pump to dewater the bottom bench.

ECONOMICS OF DEWATERING

Three basic methods of dewatering exist.

1. Start dewatering up to a year in advance of mining requirements. The mining level will gradually gain on the achieved dewatering and, theoretically, dewatering and mining levels should coincide for the last bench.

2. Drill new wells each year and keep the dewatering only one bench ahead of the mining schedule.

3. Install a relatively large number of pumps (the pumps should be able to dewater each bench in a two-month period). Turn all pumps on two months before the bench is required; when dewatered, turn most of the pumps off and maintain the water-table elevation.

All three methods have been used at Pine Point, although method (3) has only been partially attempted at the K-57 pit.

Method (1):

Method (1) offers limited flexibility, high operating costs and low capital costs. Dewatering is occurring so slowly for the bottom benches (because of the large amount of time involved) that, if mining rates are increased, limited flexibility is available to increase dewatering rates. To allow for variations in mining schedules and pump down time, moderate overdesign and the use of sump pumps is recommended. The method does have a number of very important advantages. Initial capital costs are low and water-level recovery rates are slow because of the well-developed cone of depression. Regional effects are maximized using this method and produce dewatering cost savings in surrounding pits.

Method (2:

This method was approximated at pit X-15 (where pumping has now ceased due to favourable regional dewatering effects from the W-17 pit). Well yields in this pit were extremely low and six pumps produced only approximately 2,000 USgpm. Each year, problems were encountered with well drilling in unconsolidated formations and dewatering was far behind schedule. The regional effect from the W-17 pit has lowered the water table at the X-15 pit an additional 27 feet and illustrates one advantage of Method (1). Method (2) is very costly because of the large number of different well drilling sessions involved. Flexibility is limited, especially if flow rates in new wells do not meet withdrawal requirements for the next bench.

Method (3):

This method has the most potential for cost savings in pit dewatering, but it has practical limitations. It utilizes the concept that the pumping rate for a particular pit should greatly exceed the dewatering rate required to dewater the benches in the time required (i.e. according to the mining schedule), plus the inflow or recharge into the pit area as the cone of depression develops. When the dewatering of a particular bench has been completed, pumping rates can be reduced to match the inflow and, therefore, hold the water-table elevation. As dewatering proceeds, higher continuous pumping rates will be needed to maintain dewatered ground. By keeping the cone of depression in the immediate area of the pit to a minimum size, the total volume of water pumped is less. The procedure applies to each bench, with more and more pumps being left operating to maintain the inflow from the larger cone of depression for the lower benches. Although capital costs are high, operating costs can be reduced, dependent on power demand.

H. G. Barker[11], of Cominco Ltd., completed an economic study of this method. An important finding of this study was the most economical pre-production length of time to operate all pumps in order to dewater a particular bench. He concluded that, for many applications, this pre-production period is approximately two months. Therefore, the required maximum pumping capacity is that which can dewater any bench in two months. In shallower dewatering applications (less than 100 feet), this method will require only 1¼ to 1½ times the pumping capacity of Method (1), but in deeper applications up to twice the number of pumps could be required.

There are a number of disadvantages associated with this method.

(a) Fast mining rates in pits are necessary to fully realize the economic advantages of this method. If the mining time is allowed to lag, the economics of Method (3) will approach those of Method (1), because the over-all extent of the cone of depression is dependent on time, transmissivity and the coefficient of storage, and not on withdrawal rates.
Mining at this projected rate in one particular pit is not often feasible because of ore-grade problems. One possible solution is to stockpile ore near the pit after mining in order to keep pumping time to a minimum.

(b) If regional drawdown is kept to a minimum by short dewatering times, the dewatering economics of surrounding pits could be affected. Method (1) more fully utilizes time to produce regional effects and, in areas where there is a large number of projected pits Method (3) may prove unfavourable.

(c) It is possible that this method will have disadvantages in areas where leaky artesian conditions exist. If the cone of depression is not fully developed in the leaky beds and the head reservoir supplying this leakage is not drained, problems could exist with unstable walls, and wet stripping and mining.
Power demand using Method (3) may exceed power supply. Given two or three open pits dewatering simultaneously in the artesian area of the property, not enough power could be supplied from present-day sources.

In trying to reach a realistic conclusion as to the most economical method of dewatering, the practical and physical limitations of the area must be considered. The influence of dewatering on the mining schedule has been minimal in the past, but costs are now major and power supply is limited. In extreme cases in the future, therefore, the mining schedule could be determined wholly by the amount of dewatering required in certain areas. Proximal orebodies may have to be mined and dewatered simultaneously to maximize regional drawdown effects.

It is concluded that, with slight overdesign and the optional use of sump pumps, Method (1) will be the most practical and economical dewatering procedure in future years.

DEWATERING IN THE SEMI-CONFINED AND CONFINED WATER TABLE AREAS

These areas present a unique problem for pit dewatering. Before any actual dewatering takes place in these aquifers (i.e. the rock becomes unsaturated), a water or artesian pressure must be released. Because the actual aquifer remains fully saturated while the pressure drop is occurring, the volume of water being released from the aquifer is attributed to a slight compression of the aquifer skeleton and a corresponding slight expansion of the water. The aquifer will remain fully saturated until the pressure head is entirely removed and actual dewatering of the aquifer (as versus pressure dewatering of the aquifer) will then take place. The pressure drop is directly related to the amount of water released from storage per unit drop in head per unit area, which, in turn, is indicated by the storage coefficient. For water-table (unconfined) aquifers, however, the coefficient of storage approximates the specific yield of the rock type of the aquifer. (Specific yield is the amount of water, expressed as a percentage of the fully saturated volume of water, that will drain from the rock under the influence of gravity.) It is easy to imagine, then, that the amount of water released from storage in an unconfined aquifer is much larger than that released from a confined aquifer. Therefore, when the transition from "pressure dewatering" to actual dewatering takes place, there will be a change in the coefficient of storage, within the range of 0.0001 (confined) to 0.3 (unconfined). This is very important, as the storage coefficient is a measure of the volume of water that has to be withdrawn to achieve specific drawdowns. In calculating drawdowns in originally confined water table areas, it is very important to use the water-table coefficient of storage, if drawdowns below the top of the confined aquifer are required (Fig. 9). The "pressure dewatering" will occur much faster than the actual desaturation of the aquifer and, using the confined coefficient of storage, will lead to extreme underdesign.

There are two methods which describe the transition from confined to unconfined conditions. They may have limited use during single pump aquifer tests, because they require that the water level drops below the bottom of the confining layer, but they could perhaps be applied when the first four or five pumps are operating.

The first method has been devised by Boulton[2, 3] and put into useful type curve form by Prickett[18]. It involves matching early and late field-drawdown data to different type curves. The time from the start of pumping after which true unconfined conditions exist can be calculated, and the data after this time are used to obtain an unconfined value for the coefficient of storage.

The second method, by Moench and Prickett[14], also involves matching field data to type curves. Time-drawdown data from observation wells, which have already undergone storage-coefficient conversion during the period of pumping, may also be analysed if the pumping well has undergone conversion.

The reader is referred to the above articles for complete step-by-step details.

Separate from the above problem, two other possible problems may be associated with dewatering in artesian areas:

(a) the water pressure beneath the pit floor during initial stripping and mining;
(b) wet stripping and mining conditions caused by slow ground-water drainage from confining beds.

Confined aquifers have a hydrostatic pressure. This pressure is equal to the height of the pressure surface (i.e. the height to which the water will rise) above the top of the aquifer. Unless pit dewatering commences well before stripping and mining, ground water at pressure may exist below the pit floor. Brealey[6] discusses the consequences of such conditions.

Slow drainage of confining beds and minor aquifer zones may result in wet stripping and mining conditions.

If the confining beds have a very slow drainage rate, the upper section of the main aquifer may be dry while the confining beds are still draining. The rate of drain of the confining bed is a function of its transmissivity. It will be important, therefore, to start dewatering considerably in advance of mining in order to ensure dry confining beds in the pit walls. Test holes may have to be drilled in the confining beds to follow the development of the cone of depression and also to determine its aquifer coefficients.

Conclusions

As dewatering requirements become deeper and general ground-water conditions become more complex, pit dewatering will continue to be a major expense in the mining budget of Pine Point Mines Limited. As such, the system warrants a continuing study of methods of improvement and subsequent lowering of over-all unit costs.

As mining of orebodies progresses more into the confined and semi-confined ground-water areas of the property, more extensive aquifer tests will be needed to accurately predict long-term drawdowns. This may necessitate withdrawals of up to 5000 USgpm over a 30-day period to obtain realistic values of the coefficient of storage.

Method (1) appears to represent the most economical and practical dewatering design, especially if proximal orebodies are mined and dewatered simultaneously to ensure efficient regional drawdown effects.

Because of increasing dewatering costs and limited power supply, it is entirely possible that future mining schedules and forecasts will be determined entirely by open-pit dewatering requirements.

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References

(1) Barker, H. G., 1971, Economics of dewatering at Pine Point, N.W.T., unpublished Cominco Ltd. report.

(2) Boulton, N. S., 1963, Analysis of data from non-equilibrium pumping tests allowing for delayed yield from storage, Proc., Institution of Civil Engineers, V. 26, No. 6693.

(3) Boulton, N. S., 1964, Discussions on the analysis of data from non-equilibrium pumping test allowing for delayed yield from storage, Proc., Institution of Civil Engineers, V. 28, Aug.

(4) Brashears, M. L., 1968, Ground water conditions and dewatering of ore pits at Pine Point, N.W.T., Canada, Leggette, Brashears & Graham report to Cominco Ltd.

(5) Brashears, M. L., and Slayback, R. G., 1971, Pumping test methods applied to dewatering investigations at Pine Point Mines, N.W.T., Canada, AIME Ann. Mtg., New York, March.

(6) Brealey, S. C., 1964, Ground water control in open-cast mining, Paper 18, Proc., Symposium, Institution of Mining and Metallurgy, Nov. 19.

(7) Calver, B., and Farnsworth, D. J. M., 1969, Open-pit dewatering at Pine Point Mines, CIM Bulletin, V. 62, No. 692; CIM Trans., V. 72.

(8) Cooper, H. H., and Jacob, C. E., 1946, A generalized graphical method for evaluating formation constants and summarizing well-field history, Am. Geophys. Union Trans., V. 27, No. 4.

(9) Davis, S. N., and DeWiest, 1966, Hydrogeology, John Wiley and Sons Inc.

(10) Ferris, J. G., *et al.*, 1962, Theory of aquifer tests, U.S. Geol. Survey Water Supply Paper 1536-E.

(11) Hantush, M. S., and Jacob, C. E., 1955, Non-steady radial flow in an infinite leaky aquifer, Trans. Am. Geophys. Union, V. 35, No. 6.

(12) Jacob, C. E., 1946, Radial flow in a leaky artesian aquifer, Trans. Am. Geophys. Union, V. 27, No. 2.

(13) Jacob, C. E., 1947, Drawdown test to determine the effective radius of an artesian well, American Soc. of Civil Engineers.

(14) Moench and Prickett, 1972, Radial flow in an infinite aquifer undergoing conversion from artesian to water table conditions, Water Resources Research, V. 8, No. 2, April.

(15) Prickett, T. A., 1965, Type-curve solution to aquifer tests under water table conditions, Ground Water, March-April.

(16) Skall, H., 1975, The Paleoenvironment of the Pine Point lead-zinc district, Econ. Geol., V. 70.

(17) Theim, G., 1906, Hydrologische methoden, (see Wenzel, L. K., 1936, The Theim method for determining permeability of water-bearing materials, U.S.G.S. Water Supply Paper 679-A).

(18) Theis, C. V., 1935, The relation between the lowering of the piezometric surface and the rate and duration of discharge of a well using ground water storage, Am. Geophys. Union Trans., V. 16, Pt. 2.

(19) Todd, D. K., 1959, Ground Water Hydrology, John Wiley and Sons, Inc.

(20) Walton, W. C., 1960a, Application and limitation of methods used to analyze pumping test data, Water Well Journal, Feb.-March.

(21) Walton, W. C., 1960b, Leaky artesian aquifer conditions in Illinois, Illinois State Water Survey Rep. of Invest. No. 39.

(22) Walton, W. C., 1962a, Selected analytical methods for well and aquifer evaluation, Illinois State Water Survey Bull. 49.