

HOW AN EXPLOSIVE CAN SHAVE COSTS - AND YET
SEEM EXPENSIVE.

Per-Anders Persson

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Summary:

Recently, the ability of dry mix explosives to decrease mining costs has been discussed (1, 2). The present article is prompted by the apparent contradictions in the views put forward by different authors. It tries to explain why explosives specialists, with limited access to the multitude of factors that influence the economy of a mining operation, often tend to over-simplify the issue. The questions arising in the work of optimizing a mining operation are far more complicated (and for that matter, far more exciting) than the question whether the left hand side in a physical chemical equation is positive or negative. The view is put forward that mining men are better positioned than anybody else to decide whether a given explosive shaves their mining costs or not. Explosives manufacturers can serve mining men best by supplying reliable strength data for their products and by seeing to it that their products are reliable and well functioning systems. The existence of simple computer programs and slide rule methods for charge and hole pattern calculations that are manageable by the mining engineer is pointed out and exemplified.

Introduction

Explosives people obviously strive to create the impression of understanding the business of mining better than do the miners. Yet, they seem vague and undecided when asked what is the best explosive for mining. The former conduct is understandable, human, but unforgivable. The latter is forgivable, as will be shown, but difficult perhaps to understand. Difficult in particular, considering that dynamite was invented in 1867, ANFO also in 1867, and the first water gel blasting agent containing no explosive ingredients in 1901 (3, 4, 5).

The present article advocates the view that cost calculations, in order to be meaningful, have to be carried out by the people who have all the input data for the particular mine in question, i.e. the mining engineers and operators. Explosives people should supply them with good, dependable, field tested systems of equipment and explosive products together with the necessary data on the basis of which reliable and sound cost calculations can be carried out. Such calculations may well show, because of the different special conditions prevailing in different mines, that the explosive that saves cost in one mine can increase costs in another.

The typical mine

Let us consider the scene of a typical large mine. Is there, then, such a thing as a typical large mine? Certainly not. Each large or small mine around the world has its own special operating conditions. One has high wages, another has poor rock, a third has a very hot climate, a fourth has sub-zero temperatures and no daylight for long periods each year. The only thing they all have in common is a fairly large staff of able mining people, each one in his capacity carefully chosen to do his part of the very complicated business of modern mining. Their objective may or may not be to get the greatest amount of ore out of the ground at the lowest possible cost. Very likely they have a number of important side-objectives to take into consideration, such as the long range employment situation in the district. They will have to look far into the future and try to guess what raw material prices and types of ores will prevail 10 years ahead and they will have to keep themselves acquainted with new developments in metallurgy and fabrication.

There are certainly no simple universal answers and solutions to the question how to operate a mine in the best way. In fact, even the internal process of decision-making in today's big business is becoming a complicated and many-faceted activity by many people rather than the one man show we all tend to believe it to be.

It is then imperative that good, reliable input data and computational tools for cost calculations are put in the hands of the mining people, so that they themselves can do the cost versus production optimization. It is also imperative that products offered for sale to them are practical, well tested and functioning systems. Such systems must include a great deal of operational reliability and a guarantee for long time availability and delivery potential.

Toxic reaction products underground

In underground mining, particularly in large, deep mines, the problems of ventilation and air pollution underground become increasingly difficult. Where loading and transporting is done using diesel-powered equipment while blasting is done using ANFO, a rough estimate indicates that about 10% of the total fuel oil consumption is burned in the ANFO explosive. Each kilogram of diesel oil burned in a truck underground requires several thousand kilograms or cubic meters of ventilation air to be brought down into the mine through kilometer-long ducts. Obviously, the explosive reaction products, particularly as they are developed intermittently, are part of the air pollution problem underground where such a problem exists. Depending upon the explosives ingredients, different amounts and different kinds of toxic reaction products are generated. The toxic gas generation is also influenced by the cartridge material and its relative weight, as well as by the willingness to detonate, the degree of packing, and the stemming of the borehole. For each kind of explosive, toxic gas tests should be carried out, first by laboratory methods, then by measuring in production blasting. When offering a new explosive for sale to be used underground, a responsible explosive manufacturer should have carried out (and be willing to present for the mining man's inspection) full scale blasts where the reaction generated toxic gas concentration have been measured. For each new kind of explosive, the final composition and consistency are normally adjusted to give a minimum of toxic gases. Because of the complexity of the problem of optimizing in the mine, where the costs of different explosive ingredients and compositions must be weighed against the gains to be made by improving the atmosphere underground, it is highly advisable that it be solved by pooling the knowledge of the mining people and the explosives people. Explosives people again know too little about the rest of the problem to be able to put their great potential of chemical and physical know-how to bear on the problem on their own.

Figure of merit for blasting performance of explosives

In the special area of explosives and blasting, we need a figure of merit for the strength of explosives to use as an input figure in cost calculations.

The basic mechanism of rock blasting is now fairly well understood (6,7, 8, 9). We no longer have to worry about the details of the detonation mechanism or the shock impedance matching between the rock and the explosive. These are important factors influencing the stability of detonation and thus have to be kept in mind when using an explosive close to its critical diameter. That, however, is an area we want to stay well clear of in order to have a reliable, well functioning system. There is ample experimental evidence to show that explosion energy or heat of explosion is the one major factor in the figure of merit for practical production rock blasting, and that gas volume is the other. Tests carried out by Langefors and Kihlström (10) gave the following expression for the relative rock breaking capacity s

$$s = \frac{5}{6} \frac{Q}{Q_0} + \frac{1}{6} \frac{V_g}{V_0} \dots\dots\dots (1)$$

where $Q_0 = 5000$ kJoules/kg and $V_0 = 0.850$ m³/kg are the values of explosion energy and gas volume of the standard explosive used for comparison, a Swedish dynamite called LFB dynamite^{*)}. To transform the figure of merit into the universal system with 94.4/5.6 AN/FO as a standard, we divide the s -value with 0.84 or, if we want the 50/50 nitroglycerine/nitroglycol blasting gelatine as a standard, we multiply with 1.27. These factors were the experimental values for ANFO and the blasting gelatine, respectively (table 2). The physical meaning of equation (1)

^{*)} LFB-dynamite is a registered trade-name for a low freezing point, plastic dynamite, containing 35% nitroglycerine/nitroglycol and with AN as an oxidizer. It has now been replaced by the less sensitive Dynamex plastic dynamites of about the same strength.

is simply that a 10 percent change in explosion energy has between 4 and 5 times greater effect on the figure of merit than a 10% percent change in the gas volume. The figures for various oxygen balanced mining and construction explosives have been tested in Sweden and around the world in large scale operations involving very large quantities of explosives and in various kinds of rock (table 2).

For such oxygen balanced explosives, the calculation of explosion energy and gas volume is particularly simple, as the reaction products are simple and do not change with pressure and temperature. Table 1 gives some values for the calculation at oxygen balance for a selection of ingredients in present day explosives.

When using aluminium as a strength-increasing ingredient, or high TNT contents in order to increase sensitivity, it is sometimes advantageous to depart from the oxygen balance and use oxygen deficient compositions. The calculation then becomes somewhat more delicate, as one has to take into consideration to a certain extent the loading density and the slight changes in composition during expansion that may occur, particularly when excessive amounts of carbon are present. No explosive manufacturer should, however, have difficulties in supplying his customer with an accurate theoretical or experimental figure for energy and gas volume, even with these types of compositions. One exception may be aluminium explosives. For reasons, some of which have been dealt with at great length by Cook (11) and some of which we still do not understand, the high theoretical energy of the aluminium - oxygen reaction is not always obtainable in reality. This is particularly true in highly oxygen deficient explosive compositions.

Strength testing methods

In such cases strength testing methods of various kinds are used to check the calculated energy and gas volume values. As there is no one single testing method which accurately models large scale rock blasting, resort has to be taken to several methods. Earlier, the ballistic mortar and the Trauzl or lead block test were convenient methods to use for dynamite (7). Because of the small quantities 5 or 10 grams of explosive involved, the methods were well fitted into any explosives laboratory. Today, the small quantities present problems when we are dealing with explosives which may detonate readily and reliably in a 5" or 10" borehole but not in a 1 1/2" paper cartridge. Instead, we have been turning to underwater shooting tests using 5 or 10 kg charges. In these tests, the energy is derived from the measured values of the time for expansion and collapse of the reaction product gas bubble together with

the water shock wave amplitude and momentum. Another potentially useful technique is the tube expansion test, where a heavy gauge steel or aluminium tube of a sufficiently large internal and external diameter is filled with the explosive which is detonated. An analysis of a record of the radial acceleration and expansion velocity of the tube wall gives further information of the available explosion energy. Figure 1 shows a Kerr cell photograph of a model of such a tube, figure 2 shows a computer prediction^{*)} of the tube wall motion.

However much is derived from tests and theory, the final proof of the performance figures is still the performance in large scale practical blasting. It is usually impossible to judge from one single blast only, or from a small series of test blasts, because of the variation of the rock, and incidental differences in the accuracy of drilling, and in the accuracy of measuring the end result: overall site capacity and cost. One should not be content until one has the result of several months or a years operation, preferably including a comparison with previous results using other, well known explosives.

Drilling and blasting cost calculations

It has been stressed above that no outsider can do the work of cost analysis for the mining operator or the mining engineer. Nobody else has insight into the full complexity of the whole operation and the conditions laid down by previous decisions, existing equipment and planned changes in operation.

Fortunately we now have an arsenal of calculating methods to be put at the disposal of mining men, methods which were not available 10 years ago. The foundation for such calculations was laid in 1963, when Långfors and Kihlström published their epoch-making work "The modern technique of Rock Blasting" (8). In this book, they brought charge calculation and blast design, hitherto a magic performed by blast foremen, into the realm of engineering, predictable as any other engineering subject. The methods have now been tested during more than a decade and we can safely say they function very well. In Sweden, we are now increasingly programming these methods for rapid computer calculations. A small table-top computer is often all that is needed.

In the following we will compare the values of the cost of drilling and blasting with different explosives. These are described in table 3 a. The comparison is made for three widely differing kinds of blasting: open-pit, large-diameter bench blasting, underground long-hole, small-diameter blasting, and tunnelling. We have allowed the drilling cost to

^{*)} IBM 360/75

vary in a realistic way with the hole diameter, but for simplicity have kept the price of each explosive including the cost of loading constant.

a. Large-diameter, open-pit mining

Table 4 shows a comparison of the cost of drilling and blasting using different types and combinations of explosives in a 10" borehole, 15 meter bench height, open pit operation. Loading data, strength, and cost figures for the explosives are found in table 3b. The first and simplest test is to survey, using the values of explosive breaking capacity, price, and drilling cost, the approximate total cost assuming we could use one explosive only for the whole blast. This is done best by assuming a constant specific loading for the whole volume of rock, in our example (1 in table 4) 1.52 kg ANFO equivalent per solid cubic meter of rock. Next, we refine our calculation using the small table-top computer^{*)}, to take various factors such as bottom- and column charge explosives, bore hole deviations, etc. into account. The detailed result is shown in table 5, the cost results under 2 in table 4. The computer program was developed by N. Lundborg at the Swedish Detonic Research Foundation (13). Then, we look at the nearest known example of a full scale operation. We see that the computer program has under-estimated the need for a large column charge and a short unloaded borehole. The program is not written to take into account the considerable over-loading, particularly in the upper part of the column charge and close to the surface, that is apparently necessary to get the required fragmentation in large-diameter blasting where the average boulder size must be only a small fraction of the burden.

The open pit mining operator now has a beginning from where to start his costing work. Should he have an over-capacity of drilling, he may find advantage in using ANFO or Al-ANFO to keep his costs, albeit at a low rate of production, as low as possible. He may perhaps even mix the explosive himself, given suitable conditions and time. Should he, on the other hand, have a problem getting enough production out of a given drilling equipment, he might choose to use a high energy, high density slurry for a bottom charge in order to be able to drill half as many boreholes for the same size blasts.

b. Small-diameter, long-hole underground mining

An illustration of the futility of making sweeping statements about blasting costs is shown in table 6, where the drilling and blasting cost is compared for different explosives used in long-hole underground mining with 1.5 - 2" holes. Compared to the large-hole, open pit results

^{*)} Hewlett-Packard 9100 B

(table 4), we see that from a cost point of view the order of explosives is reversed. Because of the comparatively higher drilling costs, small diameter blasting is usually cheapest using high density, high energy explosives when drilling and blasting costs only are considered. However, the mining operator may find, as apparently did LKAB in Kiruna, Sweden, that the gains derived from the highly mechanized, bulk handling and loading system made possible by the ANFO alternative more than compensates the extra drilling costs.

c. Tunnelling

Perhaps tunnelling is the most difficult area of all in which to make reliable comparisons between different methods or explosives. This is partly because the charge and drilling pattern calculation is tedious, if carried out manually, partly because of the high bonus put on speed of loading when high labour costs are involved, partly because of the strong restrictions laid down by existing equipment and geometrical factors.

Figure 3 shows a comparison between a parallel hole pattern for a 25 m² gallery used in practice and a computer calculated and plotted pattern for the same gallery. The computer^{*)}, using a program (12) developed jointly by the Swedish Detonic Research Foundation, Atlas Copco AB, Nitro Nobel AB, and Nitro Consult AB, was fed with data for explosives from table 3, together with data describing the hole diameter and tunnel profile. The program calculates and lists for each hole the hole position and the bottom charge and pipe charge weights, together with pertinent data such as costs and explosives consumption. Table 7 gives a summary of the calculation results as given by the computer together with the input data as written out by the computer for the hole diameter and explosives used in practice.

Table 8 gives a comparison of the computed tunnelling cost per cubic meter solid rock between different explosives. In this table are also given, for comparison, the costs derived from the simple method of keeping the specific charge of the bottom part of the blast constant while varying the hole diameter to contain the necessary bottom charge in the same length of hole for all explosives.

Again, in order to be meaningful in the context of practical mining, these figures have to be put into place among the multitude of other figures in the total cost calculation for a particular mine.

^{*)} IBM 1130

d. General

The pillar-diagram in figure 4 shows the relative costs for drilling and blasting using different explosive systems. On the average, these costs in large-diameter open pit mining, long-hole underground mining, and tunnelling relate to each other, roughly, as the figures 1:3:9. These relations are due firstly to the difference in drilling cost between large and small diameter drilling, secondly to the less complete filling of the boreholes in tunnelling than in long hole underground mining. Figure 4 shows, again, how the minimum cost is achieved with different explosives in different applications, already when we take into account only drilling and blasting costs. In our examples, ANFO gives the lowest drilling and blasting cost (at the price of doubling the number of boreholes) in open pit mining, whereas the high density, high energy Dynamex dynamite comes out with the lowest drilling and blasting cost in both underground long-hole blasting and tunnelling. An even cheaper alternative would be Reolit A10 but this type of explosive, because of its, Al-content and not being oxygen balanced, is not as yet allowed for underground work in Sweden.

Fragmentation, fines, and total optimization

In mining, and increasingly in construction work, too, the blasting operation is followed by further mechanical treatment of the rock or ore material. The first of these is secondary blasting or boulder breaking, followed by loading, transport, mechanical crushing and the various grading and sorting operation.

Secondary blasting, a treatment given only to a few percent of the total volume of rock is without comparison the most expensive mode of size reduction. In many quarries, the man-power and shovel time spent on boulder treatment equals or even exceeds that spent on the rest of the blasting and loading operation. Often, complaints from neighbours about disturbances from quarries are caused by excessive and noisy boulder blasting rather than by the main blasts.

We know for sure that factors such as the size of the burden, the ratio between hole-distance and burden, the length of the stemming or unloaded part of the hole, and the specific charge (expressed in ANFO equivalents) all influence the amount of boulders as well as the fragmentation generally. It may not be possible to eliminate boulders entirely, but the total optimization certainly has to take the factor of boulders into account.

Fragmentation is important particularly from the point of view of the capacity of the shovel, transport, and primary crusher. In carefully

controlled full scale experiments with the wide-spacing method of blasting, we found that a 10 to 15 percent increase in shovel capacity was obtained as a result of an improvement in fragmentation at no change in the specific loading or the number and length and diameter of holes. Comparing different explosives systems, we must take into account and evaluate the changes in fragmentation caused by the differences in hole pattern due to different explosive strengths.

In mining, particularly in iron ore mining, it is often advantageous to keep as low a content of fines as possible in the ore produced. Experiments in LKAB, Kiruna (14) indicate that between 5 and 10 percent of the total, final amount of fines is produced in the process of blasting, the rest in the further handling. Even though this percentage is small, the economic loss incurred by transforming unnecessarily large amounts of ore into fines by using an unnecessarily violent blasting technique may be large. We must perhaps take this factor, too, into consideration in our final cost calculation.

When one tries to make the momentous decision of acquiring a certain size crusher, truck, shovel or drilling equipment or when deciding what explosive to use for a new mining operation the question of fragmentation is of major importance. To a certain extent, he can be guided by computations such as have been described above. However, no member of the mining profession would seriously consider making that decision without having seen for himself similar operations in action and without having taken the advice of colleagues engaged in these.

Thus, although we now have an arsenal of computational aids available for the mining engineer to use in his cost predictions, values derived from practice must still be regarded as an invaluable check-point.

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Table 1. Values for calculating explosion energy and gas volume of oxygen balanced explosives.

Substance	Oxygen balance kg O/kg substance	Contribution to explosion energy kJoule/kg	reaction product gas volume at NTP m^3/kg
<u>Oxidizers:</u>			
Ammonium nitrate	0.200	1584	0.8400
Sodium nitrate	0.4706	-1148	0
Potassium nitrate	0.3956	-1146	0
Calcium nitrate	0.4876	-1794	0.1364
Barium nitrate	0.3061	-1625	0.0858
Potassium chlorate	0.3917	402	0
Sodium chlorate	0.4509	652	0
Potassium perchlorate	0.4619	60	0
Ammonium perchlorate	0.3404	1456	0.5725
<u>Fuels and explosives</u>			
Aluminium	-0.8889	29762	0
Silicon	-1.140	30076	0
Carbon	-2.667	32956	1.867
Sulphur	-1.100	9276	0.700
Glycerol	-1.216	16158	1.703
Glycol	-1.291	17225	1.806
Paraffine wax	-3.45	43869	3.245
Diesel oil	-3.48	44246	3.288
Guar gum	-1.185	16233	1.521
Nitroglycerine (NG)	0.0352	6338	0.6907
Nitroglycol (EGDN)	0	6840	0.7368
50/50 NG/EGDN	0.0176	6589	0.7137
Trinitrotoluene (TNT)	-0.7398	14597	1.0850
Cyclotrimethylene tri- nitramine (RDX)	-0.2161	8987	0.9076
60/40 RDX/TNT			
Pentaerythritoltetra- nitrate (PETN)	-0.1012	7681	0.7795
Nitro cellulose (12% N)	-0.3850	9758	0.9327
"- (12.5%N)	-0.3517	9490	0.9090
"- (13% N)	-0.3184	9218	0.8850
Water (liq $\rightarrow$ gas)	0	-2323	1.244

Table 2. Experimental basis for using breaking capacity as a measure of explosive strength.

Explosive	Observed breaking capacity rel.to LFB	Explosion energy per kg KJ/kg	Gasvolume m ³ /kg	Calculated breaking capacity rel.to LFB	Calculated breaking capacity rel.to ANFO
LFB dynamite	1	5000	850	1.0	1.19
Gelatine dynamite	1.23	6750	714	1.27	1.51
LFIV dynamite	1.04 [±] 0.05	5300	834	1.05	1.25
Nitrolit	0.80 [±] 0.06	4000	834	0.83	0.98
Securit	0.93 [±] 0.03	4250	857	0.88	1.05
Nabit	0.88 [±] 0.04	4500	654	0.88	1.05
Amatol	0.90	4350	891	0.90	1.07
Ammonit	0.89 [±] 0.04	-	-	0.87	1.04
ANFO	0.86 [±] 0.04	3900	974	0.84	1.00
AN-TNT-slurry	(0.74 [±] 0.04)			0.74	0.88

Table 3a. Description of explosives.

ANFO	94.4/5.6 Ammonium nitrate/fuel oil mixture, either with prilled AN, giving relatively low loading density, or with chrystalline AN, giving higher loading density.
Dynamex	High density plastic dynamite in paper cartridges.
Nabit	Low density, powder dynamite in paper cartridges or plastic tubes. Mainly used to give a low loading density for column charges.
Reolit A10	High density TNT sensitized aluminized slurry containing about 15% aluminium. Not allowed in Sweden for underground work.
Reolit 10	High density TNT slurry.
Al-ANFO (5%) Al-ANFO (10%)	These experimental aluminized ANFO qualities, in which some fuel oil has been replaced by aluminium (5 or 10% of the whole weight) have not been field tested in Sweden. They are not allowed for underground work.

Table 3b. Values of breaking capacity, explosive density in borehole, and assumed price of some explosives.

	Breaking capacity rel. ANFO	Explosive density, ρ_0 g/cm ³	Loading density in borehole g/cm ³	Price US \$ ¹⁾ per kg explosive
<u>Open pit mining</u>				
Dynamex	1.19	1.45	1.27	0.500
Reolit A10	1.29	1.5	1.5	0.390
Reolit 10	0.86	1.5	1.5	0.232
ANFO	1.00	0.9	0.9	0.170
Al-ANFO (5% Al)	1.21	0.9	0.9	0.226
Al-ANFO (10% Al)	1.39	0.9	0.9	0.282
<u>Underground mining</u>				
ANFO	1.00	0.9	0.9	0.170
ANFO	1.00	1.1	1.1	0.170
Al-ANFO (5% Al)	1.14	1.1	1.1	0.226
Al-ANFO (10% Al)	1.29	1.1	1.1	0.282
Dynamex	1.19	1.45	1.27	0.500
Dynamex	1.19	1.45	1.45	0.500
<u>Tunnelling</u>				
Dynamex ²⁾	1.19	1.45	1.25	0.500
Nabit ²⁾	1.05	1.1	0.44	0.330
Reolit 10 ³⁾	0.86	1.5	1.32	0.232
ANFO ³⁾	1.00	1.1	1.1	0.168
Reolit A10 ³⁾	1.29	1.5	1.32	0.390
Al-ANFO (5% Al) ³⁾	1.14	1.1	1.1	0.226
Al-ANFO (10% Al) ³⁾	1.29	1.1	1.1	0.282

1) Price is assumed to include all costs up to the moment of the blast, i.e. it includes loading and transportation.

2) Add US \$ 0.20 for each hole (detonator)

3) Add US \$ 0.23 for each hole (detonator + booster)

Table 4. Comparison of costs for drilling and blasting using different explosives and different methods of cost prediction. Open pit large-diameter blasting.

EXPLOSIVE	SPECIFIC CHARGE (kg ANFO equivalent per solid m ³ rock)	COST US \$/SOLID m ³ ROCK			
		EXPLOSIVES (including loading & shooting)	DRILLING ³⁾	TOTAL	
1. <u>SIMPLE ESTIMATE</u>					
Dynamex only	1.190	0.50	0.09	0.59	
Reolit A10 "	1.190	0.36	0.07	0.43	
Reolit 10 "	1.190	0.32	0.11	0.43	
AI-ANFO (5% AI) "	1.190	0.24	0.13	0.37	
AI-ANFO (10% AI) "	1.190	0.26	0.12	0.38	
ANFO "	1.190	0.20	0.16	0.36	
2. <u>LOADING PATTERN ACCORDING TO COMPUTER CALCULATION</u>					
Bottom charge Dynamex Column charge "	0.752	0.32	0.08	0.40	
Bottom charge Reolit A10 Column charge Reolit 10	0.830	0.25	0.06	0.31	
Bottom charge ANFO Column charge ANFO	0.829	0.14	0.13	0.27	
3. <u>LOADING PATTERN IN PRACTICE</u>					
Bottom charge Reolit A10 Column charge Reolit 10	1.160	0.34	0.09	0.43	
4. <u>EXTRAPOLATED FROM 3</u>					
ANFO only ¹⁾	1.160	0.20	0.17	0.37	
ANFO only ²⁾	1.721	0.21	0.18	0.39	

1) Same specific charge in whole blast as in 3.

2) Same specific charge in bottom part of blast as in 3.

3) Cost of drilling is based on the figure of US \$ 5.88 per meter borehole or US \$ 0.12 per litre hole volume.

Table 5. Calculation of charge by computer, and comparison with practical case. Open pit large-diameter blasting.

			Computer calculation			Practical case
<u>Input data</u>			ANFO	A10+R10	Dx	A10+R10
Hole diameter	d_p	m	.250	.250	.250	.250
Rock constant	c	kg/tfm ³	.45	.45	.45	-
Degree of packing, bottom	p_b	kg/m ³	900	1500	1270	1500
Weight strength, bottom	S_b	-	.84	1.08	1	1.08
Weight strength, bottom	S_p	-	.84	.72	1	.72
Spacing of hole/burden	E/V	-	1.25	1.25	1.25	1.25
Hole inclination	n	m/m	0	0	0	0
$q_p \cdot S_p / q_b \cdot S_b$	m	-	1	.667	0.4	.667
Faults in collaring	a	m	0	0	0	-
Faults in alignment	x	m/m	0	0	0	-
Number of holes per row	M		1000	1000	1000	15
Number of rows	N		5	5	5	5
Bench height	K	m	15	15	15	15
<u>Calculated data</u>						
Hole deviation at bottom	S	m	0	0	0	-
Depth of hole	H	m	17.15	18.15	17.79	18
Drilled meter per volume	I	bm/tfm ³	.022	0.011	0.014	.025
Bottom charge	Q_b	kg	412.21	1005.69	753.69	750
Bottom charge, height	h_b	m	9.33	13.66	12.09	10
Bottom charge, konc.	q_b	kg/m	44.18	73.67	62.34	73.67
Column charge	Q_p	kg	227.24	92.24	66.02	500
Column charge, height	h_p	m	5.14	1.25	2.65	6.5
Column charge, konc.	q_p	kg/m	44.18	73.67	24.94	73.67
Total charge	Q	kg	639.45	1097.93	819.93	1250
Spec. charge	q	kg/tfm ³	.83	.664	.632	1.042
Rock volume/hole	$Vol.$	tfm ³	772.09	1654.49	1297.04	1200
Fragmentation of rock	L	m	.65	1.05	1.04	-
Hole distance	E	m	8.02	11.75	10.40	10
Burden	V	m	6.42	9.40	8.32	8

tfm³ = theoretical solid cubic meters of rock

Table 6. Result of comparative cost calculation for long-hole, small-diameter (1.5-2") underground mining using one explosive only.

Explosive	Degree of packing in hole kg/lit.	Cost US \$ per solid m ³ rock ¹⁾		
		Explosive (including loading)	Drilling ²⁾	Total
ANFO	0.9	0.20	1.58	1.78
AI-ANFO (5% AI)	0.9	0.24	1.39	1.63
AI-ANFO (10% AI)	0.9	0.26	1.23	1.49
ANFO	1.1	0.20	1.30	1.5
Dynamex	1.27	0.50	0.94	1.44
Dynamex	1.45	0.50	0.83	1.33

1) For specific charges 1.19 kg ANFO per solid m³ rock.

2) Cost of drilling is based on the figure of US \$ 5.88 per meter borehole or US \$ 1.20 per litre hole volume.

Table 7. Input data and some calculation results as written out by the computer.

CALCULATION OF DRILLING PATTERN, CHARGE WEIGHTS AND COSTS FOR TUNNELBLASTING-NCD 1971 58A PAGE 1

PROGRAM BY NITRO-CONSULT AB BOX 238 SKARHOLMEN S-127 25 SWEDEN

PARALLEL HOLE CUT VERSION 1

SWEDISH DETONIC RESEARCH FOUNDATION 3-A SUITABLE DIAMETER (DX+NA)

FOLLOWING CONDITIONS ARE VALID

PROFILE GIVEN BY COORDINATES	ROCK DRILL SERIE	38.00 MM	NO. OF WORKERS	0.
CONTOUR SMOOTH	SPEC. DRILL SERIE	86.00 MM	HOURLY-WAGE/WORKER	0.00 KR
	ROCK CONSTANT	0.40 KG/M3	EFFICIENCY	0.00 PROC
SECTION AREA	25.43 M2			
WIDTH	6.10 M	WEIGHT STRENGTH	PRICE	
ARCH-RADIUS	0.00 M	BOTTOM CHARGE	BOTTOM CHARGE	2.50 KR/KG
X-MIN CUT-CENTER	2.55 M	COL. CONTOUR	COL. CONTOUR	5.50 KR/KG
X-MAX CUT-CENTER	3.60 M	COL. BOTTOM	COL. BOTTOM	1.65 KR/KG
Y-MIN CUT-CENTER	1.00 M	COL. REMAIN.	COL. REMAIN.	1.65 KR/KG
Y-MAX CUT-CENTER	1.50 M		DETONATORS/HOLE	1.00 KR
HOLE DEPTH	3.20 M	DEGREE OF PACKING	1.25 KG/DM3	
ADVANCE/ROUND	3.03 M		DEPRECIATION/ROUND	0.00 KR
			DRILLING COSTS/M	7.58 KR

THE CALCULATIONS GIVE THE FOLLOWING RESULTS

HOLES DRILLED/ROUND	61 HOLES	COSTS	
DRILLED METERS/ROUND	193.76 M	DRILLING/ROUND	1468.74 KR
DRILLED METERS/CU.M. ROCK	2.50 M/M3	DRILLING/CU.M. ROCK	18.99 KR/M3
CONSUMPTION OF EXPLOSIVES			
BOTTOM CHARGES	25.00 KG	EXPLOSIVES/ROUND	230.83 KR
COLUMN CONTOUR	13.19 KG	DETONATORS/ROUND	61.00 KR
COLUMN BOTTOM	9.78 KG	TIME-DEPENDING/ROUND	0.00 KR
COLUMN REMAIN.	48.25 KG	REMAINING/ROUND	0.00 KR
/ROUND	96.23 KG	BLASTING/ROUND	291.83 KR
/CU.M. ROCK	1.24 KG/M3	BLASTING/CU.M. ROCK	3.77 KR/M3
MAX CO-OPERATING CHARGE IN DYNAMEX-EQUIVALENTS		TOTAL COST/ROUND	1760.57 KR
CUT	0.00 KG	TOTAL COST/CU.M. ROCK	22.76 KR/M3
BOTTOM-HOLE,	2.52 KG/HOLE		
STOPPING-HOLE	0.92 KG/HOLE		
CONTOUR-HOLE	0.60 KG/HOLE		

TABLE OVER HOLE-DATA

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HOLE NO.	X-COORD COLLAR-	Y-COORD COLLAR-	DIRECT. OF HOLE	DEPTH OF HOLE	BOTTOM CHARGE	COLUMN CHARGE	HEIGHT BOTTOM CHARGE	HEIGHT COLUMN CHARGE	DIAMETER OF HOLE	TYPE OF HOLE
	M	M	DEGREES	M	KG	KG	M	M	MM	
1	0.00	0.00	0.00	3.25	1.29	1.39	0.87	2.00	38.00	BOTTOM
2	0.91	0.00	0.00	3.25	1.29	1.39	0.87	2.00	38.00	BOTTOM
3	1.98	0.00	0.00	3.25	1.29	1.39	0.87	2.00	38.00	BOTTOM
4	3.04	0.00	0.00	3.25	1.29	1.39	0.87	2.00	38.00	BOTTOM
5	4.11	0.00	0.00	3.25	1.29	1.39	0.87	2.00	38.00	BOTTOM

Table 8. Comparison of cost for drilling and blasting in tunnelling.

Explosive	Specific charge (kg ANFO equivalent m^3 per solid m^3)	Cost US \$ per solid m^3 rock			
		Explosives	Detonators & Primers	Drilling	Total
1. LOADING PATTERN IN PRACTICE					
Dynamex + Nabit	1.70	0.61	0.17	4.08	4.86
2. EXTRAPOLATED FROM 1					
ANFO	2.29	0.50	0.19	5.19	5.88
Al-ANFO (5%)	2.28	0.70	0.19	4.47	5.36
Al-ANFO (10%)	2.28	0.74	0.19	4.14	5.07
Reolit 10	2.60	0.82	0.19	4.67	5.68
Reolit A10 + Nabit	1.70	0.44	0.19	3.85	4.48
3. COMPUTER CALCULATION					
A. Best diameter					
Reolit 10	2.85	0.93	0.19	4.54	5.66
ANFO	2.77	0.64	0.19	4.53	5.36
Al-ANFO (5%)	2.82	0.73	0.19	4.35	5.27
Al-ANFO (10%)	2.81	0.78	0.19	3.98	4.95
Dynamex + Nabit	1.34	0.62	0.16	3.92	4.70
Reolit A10 + Nabit	1.31	0.56	0.19	3.70	4.45
B. Constant diameter					
ANFO	2.26	0.55	0.20	4.18	4.93
Dynamex + Nabit	1.34	0.62	0.16	3.92	4.70
Reolit A10 + Nabit	1.42	0.59	0.18	3.79	4.56

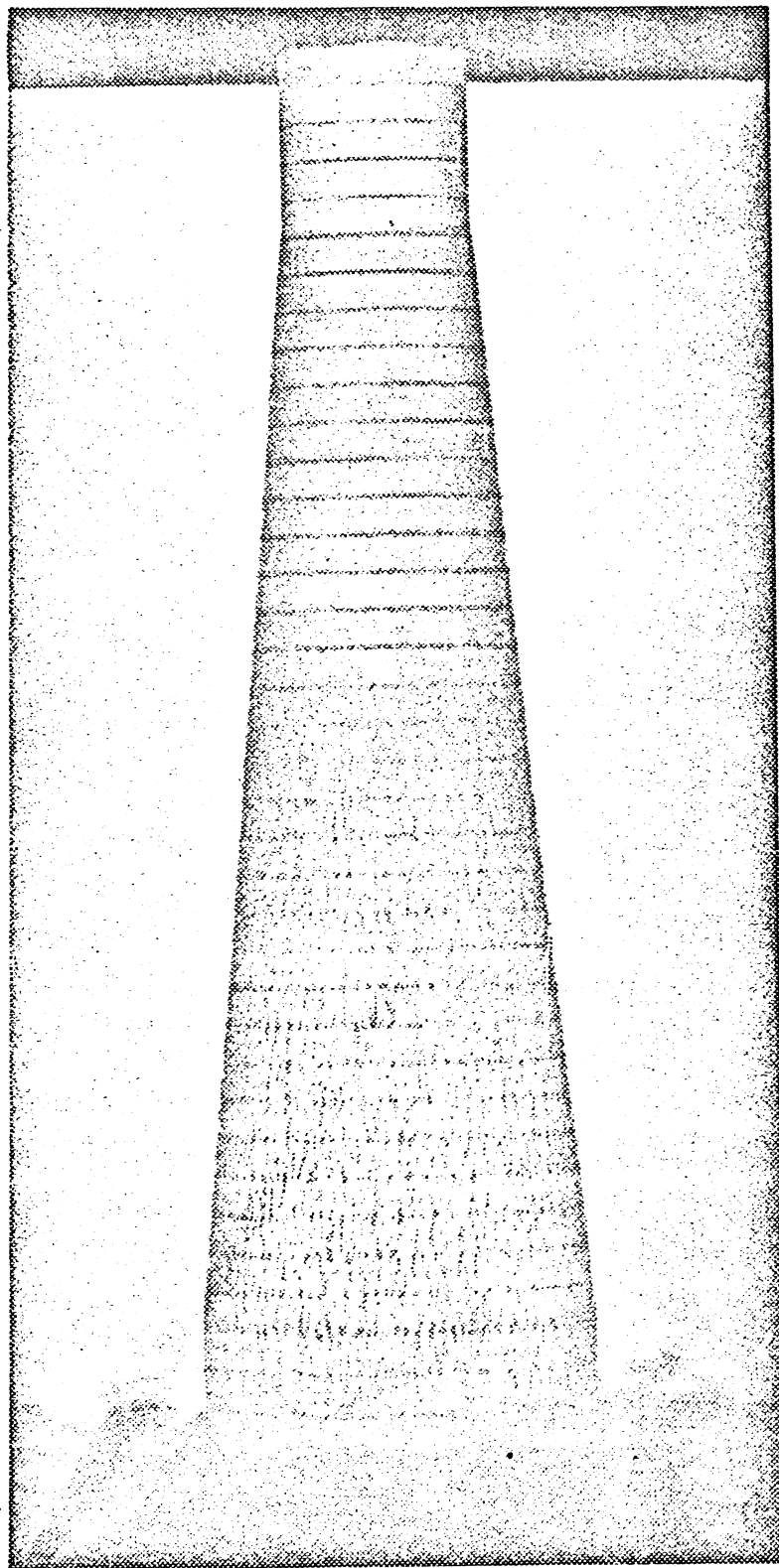


Fig. 1. Kerr-cell camera photograph of expanding aluminium tube containing a detonating charge. Exposure time: $0.060 \mu\text{sec}$. Tube dimensions: length 400 mm, $d_{\text{ext}} = 50 \text{ mm}$, $d_{\text{int}} = 20 \text{ mm}$. Explosive: 60/40 RDX/TNT, density 1.67 g/cm^3 .

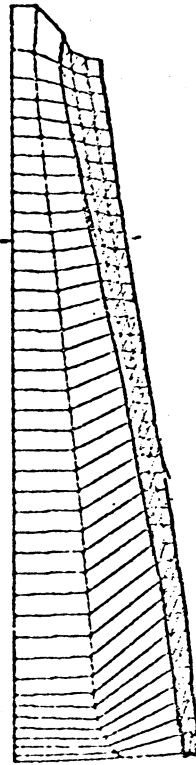
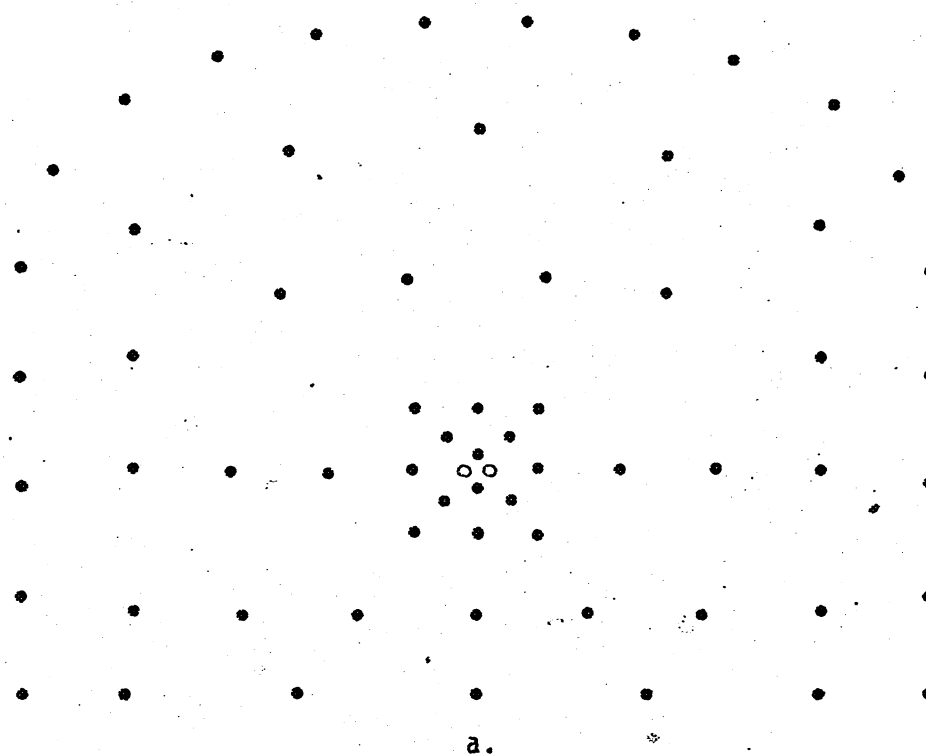


Fig. 2. Computer plot of calculated shape of expanding thick walled aluminium tube (compare figure 1).



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AREA 25.4 ADVANCE 3.0 DRILL -38.0 -35.0
 HOLES 61 KG/CU.M 1.2 DM/CU.M 2.5
 SCALE 1 TO 50 +-----+1 M

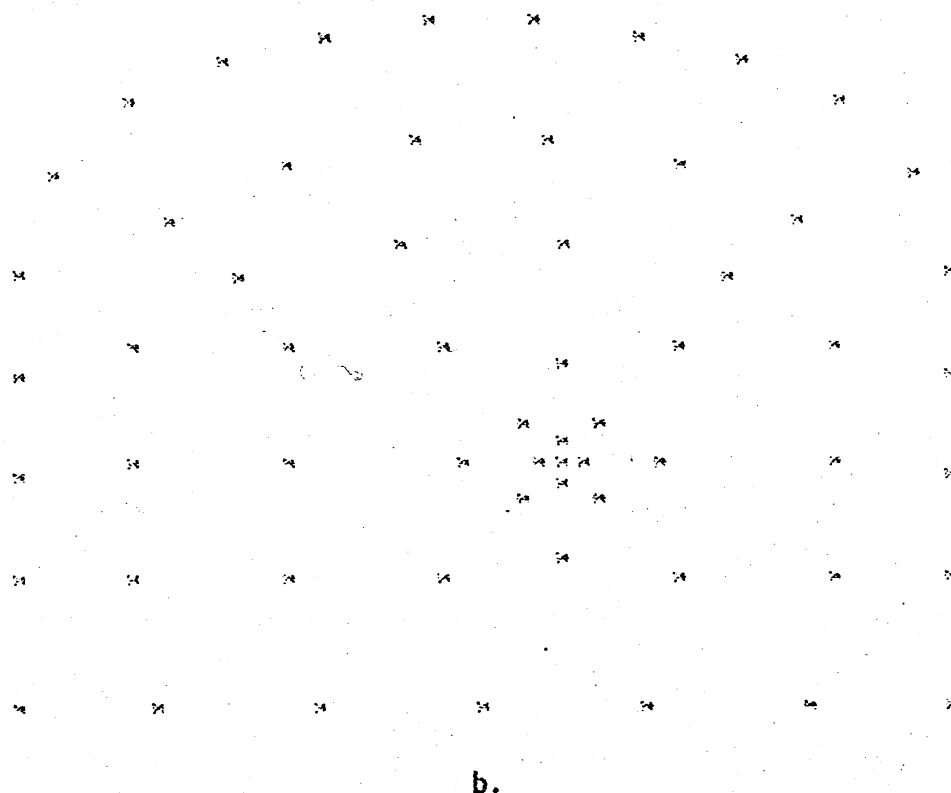


Fig. 3. a/ Parallel hole pattern for a 25 m² gallery used in practice.
 b/ Computer calculated pattern for the same gallery.

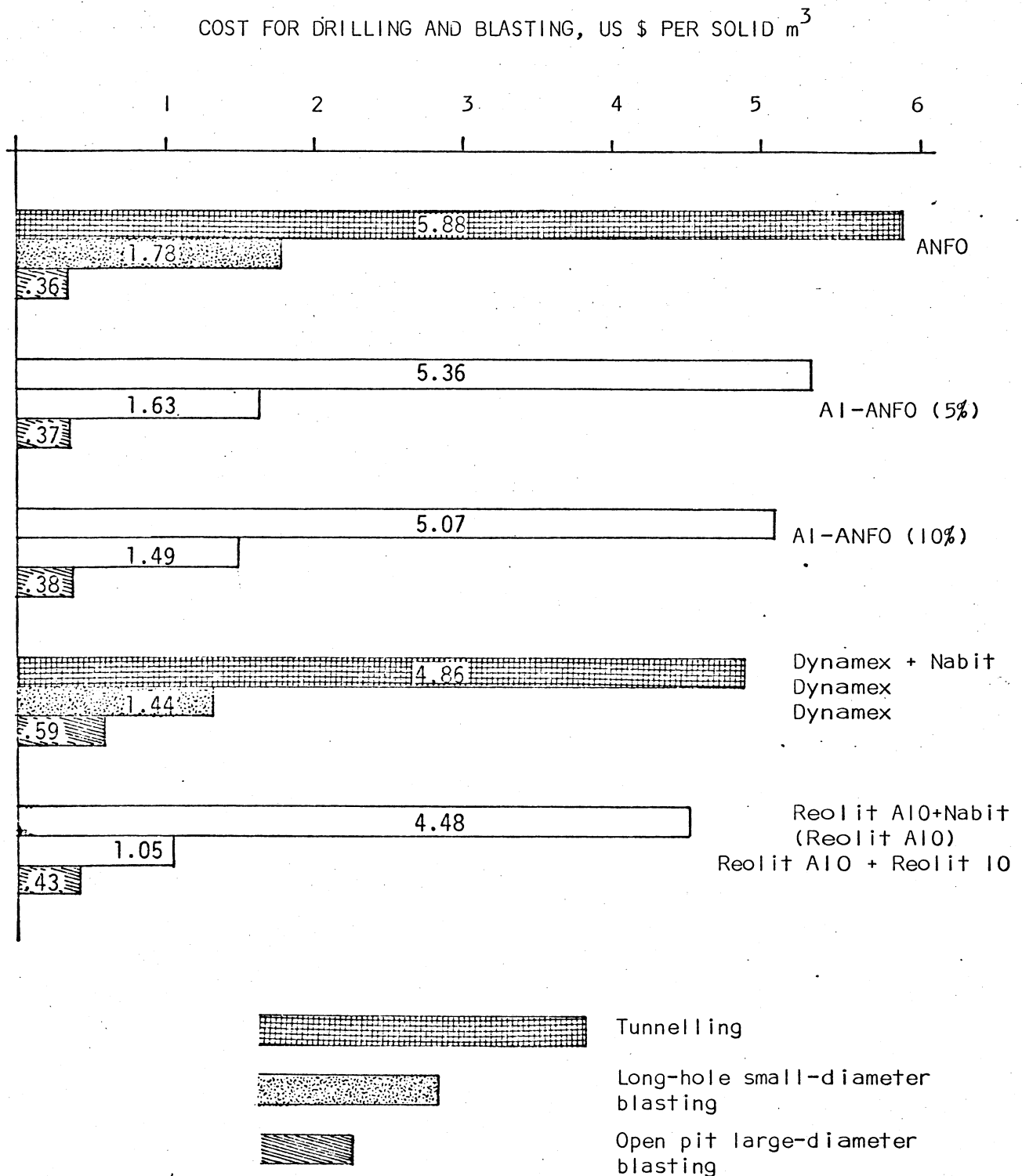


Figure 4. Comparison of drilling and blasting costs for different explosive systems and different kinds of blasting. Open pillars indicate experimental systems that are not operational in Sweden.