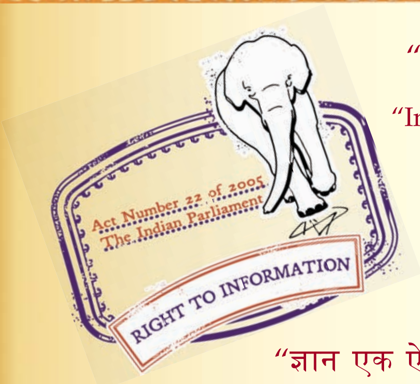
IS 14448 (1997): Code of practice for reinforcement of rock slopes with plane wedge failure [CED 48: Rock Mechanics]



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IS 14448: 1997
(Reaffirmed - 2012)

भारतीय मानक

समतल वेज पात सहित शैल ढलानों के प्रबलन — रीति संहिता

Indian Standard

CODE OF PRACTICE FOR REINFORCEMENT OF ROCK SLOPES WITH PLANE WEDGE FAILURE

ICS 93.020

BUREAU OF INDIAN STANDARDS
MANAK BHAVAN, 9 BAHADUR SHAH ZAFAR MARG
NEW DELHI 110002

October 1997 Price Group 5

AMENDMENT NO. 1 DECEMBER 2006
TO
IS 14448 : 1997 CODE OF PRACTICE FOR REINFORCEMENT OF ROCK SLOPES WITH PLANE WEDGE FAILURE

(*Page* 1, *clause* **3.5.2**, *heading and line* 1) — Substitute 'Tendon' *for* 'Tandon'.

(*Page* 2, *clause* **4.1**, *line* 5) — Substitute 'parameters affecting the slope stability' *for* 'slope parameters'.

(*Page* 2, *clause* **4.2.4**, *line* 8) — Insert 'that is' *after* 'parallel'.

(*Page* 3, *clause* **6.1**, *line* 3) — Substitute 'occurs' *for* 'occures'.

(*Page* 3, *clause* **6.3.6**, *line* 7) — Substitute 'hydrofracturing' *for* 'hydrofacturing'.

(*Page* 4, *clause* **6.4**, *line* 1) — Delete '(F. A. L.)'.

(*Page* 4, *clause* **6.4.2.3**, *line* 2) — Substitute '6' *for* '5'.

(*Page* 4, *clause* **7.1**, *lines* 2 *and* 3) — Delete 'that is' and insert 'It should be ensured that' after 'provided'.

(*Page* 4, *clause* **7.1**, *line* 4) — Substitute 'vegetation' *for* 'vegitation'.

(*Page* 5, *clause* **8.1**, *line* 1) — Substitute 'of' *for* 'or'.

(*Page* 5, *clause* **8.4**, *line* 3) — Substitute 'obtained' *for* 'obtianed'.

(*Page* 5, *clause* **9.2.1.1**, *line* 4) — Substitute 'θ' *for* 'ϕ'.

[*Page* 6, *Fig.* 2(a)] — Substitute 'ψ_p' *for* 'ψ_b'.

(CED 48)

Reprography Unit, BIS, New Delhi, India

Rock Mechanics Sectional Committee, CED 48

FOREWORD

This Indian Standard was adopted by the Bureau of Indian Standards, after the draft finalized by Rock Mechanics Sectional Committee had been approved by the Civil Engineering Division Council.

This Indian Standard is an attempt to bring more Indian Standards covering types of reinforcement and its design in various rock engineering applications.

Landslides, slips, rockfalls are some of the terms used to describe the movement of rocks under the influence of the gravity. These movements sometimes become seriously damaging or even disastrous. In highways, foundations and hydraulic engineering structures in rock, there is a need for assessing the degree of stability of man-made as well as natural slopes.

Stability can usually be improved by adding/improving the shear resistance of the rock-mass or by increasing the systems resisting force against sliding by tying or anchoring/bolting the sliding rock wedge or slab to the parent rock-mass.

Rock-reinforcement can be effectively used to stabilize the unstable natural or artificial slopes, which are isolated by joints and faults. The rock-reinforcement consisting of rock- bolts, rock anchors/cable anchors, can be used more frequently with or without steel straps, welded mesh or chain link, mesh/fabric-link, etc. The frequent use of rock-reinforcement to stabilize the rock-slopes is due to its following advantages:

a) Versatility,
b) Simple to apply in various rock conditions,
c) Relatively inexpensive,
d) Flexible performance, and
e) Easily combined with other control measures.

Technical Committee responsible for the formulation of this standard is given in Annex A.

In reporting the result of a test or analysis in accordance with this standard, if the final value, observed or calculated, is to be rounded off, it shall be done in accordance with IS 2 : 1960 'Rules for rounding off numerical values (*revised*)'. The number of significant places retained in the rounded off value should be the same as that of the specified value in this standard.

Indian Standard

CODE OF PRACTICE FOR REINFORCEMENT OF ROCK SLOPES WITH PLANE WEDGE FAILURE

1 SCOPE

This standard covers the design aspects of 'reinforcement of rock-slopes with plane wedge failure'. This standard does not cover the type of rockbolts and their installation. This will be covered by Indian Standard on 'Type of Rockbolts and their installation' (draft under preparation).

2 REFERENCES

The Indian Standards given below contain provisions which through reference in this text, constitute provision of this standard. At the time of publication, the editions indicated were valid. All standards are subject to revision, and parties to agreements based on this standard are encouraged to investigate the possibility of applying the most recent editions of the standards indicated below:

IS No.	*Title*
456 : 1978	Code of practice for plain and reinforced concrete (*third revision*)
4031 (Part 5) : 1988	Methods of physical test for hydraulic cement : Part 5 Determination of initial and final setting times (*first revision*)
11309 : 1985	Method for conducting pull-out test or anchor bars and rock bolts

3 TERMINOLOGY

3.1 For the purpose of this draft code, the following definitions shall apply.

3.2 Anchoring

By anchoring, a compressive force is introduced into the rock-mass directly towards the critical joint. The anchoring creates a compressive stress which increases the inherent strength of the rock-mass and help the rock-mass to support itself against sliding.

3.3 Structure

It denotes any construction work on or below the surface of slope whose stability is to be secured by anchoring/reinforcing. This term also includes subterranean cavities and artificial as well as natural slopes.

3.4 Rock-Medium

Rocks are understood in their natural position and infills, and it includes solid rock-masses.

3.5 Anchor

It is a device to transfer forces in a given direction from the structure to the rock-medium. An anchor is composed of three parts.

3.5.1 *The Anchor Head*

It is situated at the external (movable) end of the anchor, it is used for connecting the face/base plate and transferring of anchoring forces to the rock-mass and also used for prestressing of the anchors.

3.5.2 *The Tandon*

Tandon is the middle part of the anchor connecting the head and the root. The tandon usually allows, by its elastic deformation, establishing of permanent prestress of the anchor during the anchoring process.

3.5.3 *The Anchor Root*

It is situated at the internal (distant) end of the anchor and is used for fixing the anchor into the rock-mass.

3.5.4 *Anchor Rope/Cable*

It consists of strands of steel wires having high yield stress. Generally the steel wires have size 2-7 mm diameter.

3.6 Anchorage

This ensures the co-operation of the rock-medium with the structure, consisting of steel bars/bolts and bundles/strands of steel wires/ropes/cables. The effect of anchoring is increased by prestressing the anchorage in its free section, that is between the end fixed in the rock and the external anchor fixed in the structure.

3.7 Fixed Anchor Length (F. A. L.)

It is the length of that part of the rock-anchor farthest away from the structures over which tensile forces are transmitted to the surrounding rock-mass. F. A. L. shall not be less than 60 and 100 d_s for deformed and plain bars respectively,

where

d_s = dia of bar/bolt/cable.

The extensive field study has shown that F. A. L. shall be as per Table 1:

Table 1 Fixed Anchor Length for Different Rock Conditions

a) 2 m, for very good rock conditions	(RMR = 81 to 100)
b) 3 m, for good rock conditions	(RMR = 61 to 80)
c) 4 m, for fair/poor rock conditions	(RMR = 21 to 60)
d) 6 m, for very poor rock conditions	(RMR = 0 to 20)

4 REQUIREMENT OF STABILITY

4.1 The stability of natural or artificial slopes generally depend upon the geometry, frequency and orientation of joint sets, dip of slope and its plane of weakness and condition of the slopes. Other slope parameters are climate, hydrology, tectonic movements, presence of breccia and human activities in immediate and/or adjacent area, underground openings. Blasting bring about, years later, changes affecting the stability of slopes.

4.1.1 When any slope (natural/artificial) shows the sign of instability, then it becomes essential to stabilize the slope by adopting the effective control measures depending upon the risk involved in it, in short term as well as in long term.

4.2 Modes of Failures

4.2.1 There are mainly three type of modes of failures in the rock slopes namely planar slide, 3 dimensional wedge failure and toppling failure. In this code only one mode that is plane wedge failure is described (*see* Fig. 1).

4.2.2 The plane wedge failure occurs under gravity alone when a rock-mass rests on an inclined geological discontinuity, such as bedding/weakness plane that daylights into the free slope face that is $\Psi_f > \Psi_p$ [*see* Fig. 1(b)]. In case of 3 dimensional wedge failure, plane may be assumed at apparent dip of intersection of joints (Ψ_i).

4.2.3 Movement shown in Fig. 1(c), supposes that the restraint of sliding has been overcome not only along the surface of sliding/failure but along the lateral sides of the slide as well.

4.2.4 Sliding shall occur when the inclination of the plane of slip shall be greater than the friction angle of that plane that is $\Psi_p > \phi_j$ [*see* Fig. 1(b)]. However, in the case of unreinforced steep slopes ($\Psi_f > 90° - \Psi_p + \phi_j - \lambda$), failure may occur by over-toppling and the same is not considered here. In topping failure, the strikes of slope and joints are nearly parallel within $\pm$ 15° and the dip direction of bedding planes/continuous joints is opposite to

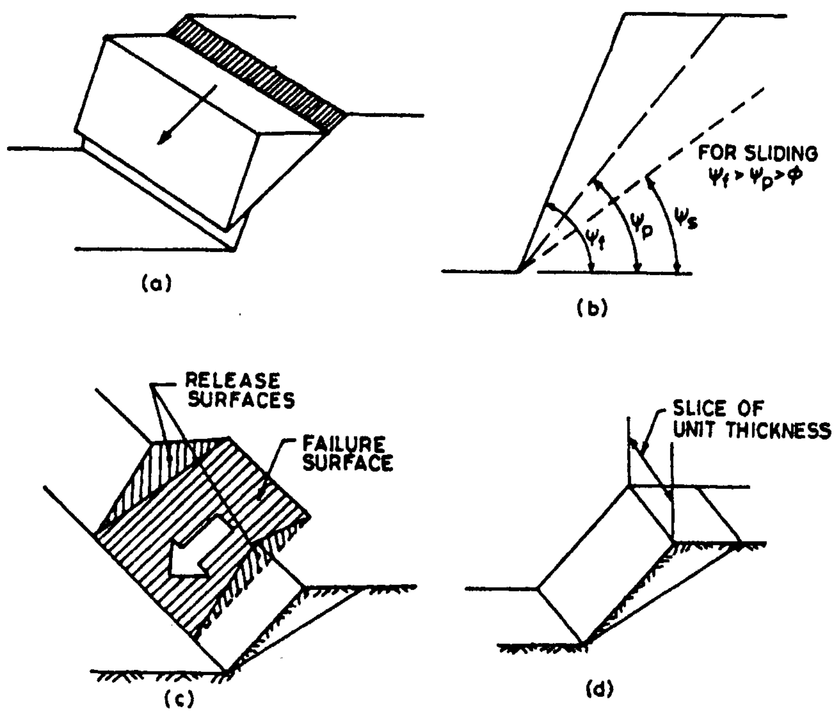

FIG. 1 PLANE WEDGE FAILURE GEOMETRY

that of the slope. The horizontal component of earthquake acceleration is accounted for by λ which is equal to $\tan^{-\alpha_n}$. Safe cut slope angle to check over-toppling is therefore equal to $(90^o - \Psi_p + \phi_j - \lambda)$. Safe cut slope angle to check 3 dimensional wedge failure is equal to the angle of intersection of critical joint planes.

4.2.5 The most likely plane of weakness/critical plane, along which sliding will occur in relation to the orientation of the slope face should be assesssed from the kinematic model analysis/stereographic plots.

5 PRINCIPLE OF ROCK REINFORCEMENT

5.1 The rock reinforcement reinforces and mobilzes the inherent strength of rock-medium which helps the slope to support itself.

5.2 The rock shall be reinforced by the steel bolts/cables to take over the tensile stresses and also a part of the shear stresses which increases the stabilizing force or decreases the disturbing/sliding force.

5.3 The rock-medium shall be locked by induced prestressing in the bolts which activates/increases the frictional forces along the surface of separation in the anchorage zone and transfer the load of unstable rock wedge to the deeper parent rock-mass.

5.4 The density of rock-reinforcement can easily be modified by changing its diameter, length and spacing which is frequently required by local rock-mass conditions. Another advantage is that it can be easily combined with additional control measures such as shotcreting and drainage system, etc.

5.5 Tests to be Conducted

5.5.1 *Kinematic Test*

This test shall be a graphical representation of rock faults/joints, etc, on the spherical stereographic plane from which the most likely unstable wedge/critical plane of weakness may be assessed.

5.5.2 *Pull-out Test*

The pull-out test should be conducted at site as per IS 11309 for assessing the anchor load bearing capacity and for the checking of the designed/assumed F. A. L. of a particular anchor.

6 DESIGN CONSIDERATIONS OF ROCK-REINFORCEMENT

6.1 The rock-bolt or cable reinforcement system should be designed to stabilize the slippage of the slope (natural/artificial), which occures due to the geological, climatic, hydrological, tectonic movements and human activities in the immediate and/or adjacent area of the structures.

6.2 For stabilizing the rock slopes, cement grouted rock-anchors/bolts/cables should be used.

6.3 Cement Grout

The delay between drilling the hole and its grouting should always be kept minimum and as a policy, one should always drill and grout the anchor on the same day. The possibility of significant shrinkage in the grout diminishing the intimate contact between the grout and rock might provide a weaker surface than that adjacent to the steel. Hence, it should be ensured that the grout either includes a small amount of a suitable expanding element or that the mix is such that the shrinkage would be insignificant.

The following specifications for a cement grout are recommended to provide flowability, expansion on hardening and higher shear strength to transfer load from the anchor through the grout to the rock.

6.3.1 There shall be no indications of the false set [*see* IS 4031 (Part 5)].

6.3.2 A flow time of 25 to 30 seconds through the standard flow cone (volume 1 725 ml, orifice diameter 12.5 mm).

6.3.3 Minimum compressive strength (on a 7.5 cm diameter cylinder and 15 cm long) should be 285 kg/cm^2, after 28 days. The water cement ratio of the grout should be 0.4 to 0.5.

6.3.4 The amount of aluminium powder necessary to obtain expansion (to avoid shrinkage) should not be more than 0.005 percent by weight of cement, for a grout of 0.4 to 0.5 water cement ratio. Other admixtures may also be used to reduce shrinkage.

Excessive quantity of aluminium powder should be avoided because it produces foaming in the grout, especially when the water cement ratio is also high. Hence, both should be strictly controlled.

6.3.5 For cement mortar grout, equal part of cement and sand are first mixed and sufficient water is then added to bring the mortar to a stiff consistency. The maximum particle size of sand should not exceed 2 mm.

6.3.6 The situation where pressure grout is required, the pressure should be limited to 50-70 percent of overburden pressure, although 15 percent overburden pressure should be applied whenever need arises. The use of excessive pressure loading gives rise to possibility of hydrofacturing and surface heaving.

6.3.7 The diameter of drill-hole should be 30 mm more than diameter of the anchor so that sufficient grout-cover prevents corrosion. Suitable preventive measures shall be adopted to prevent corrosion depending upon aggressive nature of rock mass.

6.4 The rock-anchor (F. A. L.) may fail in one or more of the following.

6.4.1 *Failure of Rock/Grout Bond*

The following expression for the length of anchorage based on an equivalent uniform distribution for bond stress as skin friction along the fixed anchor may be used:

$$L = P.F. / \pi.D.\tau \qquad ....(1)$$

where

L = Effective/fixed anchor length (F. A. L.) in mm;

P = Pull-out force per anchor, in N;

F = Factor of safety;

τ = Bond stress at failure, in N/mm^2; and

D = Diameter of the borehole (d_s + 30 mm), in mm.

The actual distribution of bond stress is exponential and depends upon the elastic modulus of the rock, grout material, diameter and length of the anchorage. However, the assumption of uniform distribution holds good for rocks of low elastic modulus. Hence, a factor of safety of 3 to 5 is recommended.

The safe bond strength ($\tau t/F$) to be used for preliminary design of rock anchors, is given in Table 2, if test data is not available.

Table 2 Safe Bond Strength for Different Rock Conditions

Very poor to poor rock condition	(RMR = 0 to 40)	0.35-0.70 N/mm^2
Fair to good rock condition	(RMR = 41 to 80)	0.70-1.05 N/mm^2
Very good rock condition	(RMR = 81 to 100)	1.05-1.40 N/mm^2

However, the above safe bond strength should not be more than one thirtieth of the minimum uniaxial compressive strength of rock material or the grout.

6.4.2 *Failure of the Grout/Anchor Bond*

If the uniaxial compressive strength of the grout (f_c) is known the permissible/allowable stress in shear (τ_a) may be obtained from the following expression:

$\tau_a = f_c/15 < 1.2$ N/mm^2 (for plain bars)

$\tau_a = f_c/10 < 2.5$ N/mm^2 (for deformed bars)

6.4.2.1 Sometimes anchor bars are pulled out because of insufficient fixed anchor length that is length of anchor bars beyond critical discontinuity.

6.4.2.2 The fixed anchor length shall not be less than 60 and 100 times the diameter of deformed anchor bars and plain anchor bars respectively.

6.4.2.3 The fixed anchor length shall not be less than 2, 3, 4, and 5 m for very good, good, fair/poor/ and very poor rock conditions respectively.

The anchorage length shall be obtained from the following expression:

$$L = \frac{P.F}{\pi D \tau_a} \qquad .....(2)$$

where

L = Effective/fixed anchor length, in mm;

F = Factor of safety;

D = (d_s + 30 mm); d_s = Diameter of anchor bar, in mm; and

τ_a = Permissible shear stress in N/mm^2.

6.4.3 *Failure of the Anchor or Top Anchorage*

The diameter of the anchor bar shall be calculated from the following expression using yield stress of steel (σ_y); which can be taken from IS 456

$$P = A_s \, \sigma_y / F \qquad .....(3)$$

where

A_s = Area of steel bar, in mm^2;

σ_y = Yield stress of steel used for anchor, in mm^2;

F = Factor of safety which will be taken as 1.30; and

P = Pull force (both static and dynamic), in N.

6.4.4 *Bearing Failure of Rock-Mass at the Face of Anchorage*

In order to minimise bearing failure (which depends upon the load carrying capacity of the rock-mass), a large bearing plate founded on a thin grout over a large contact area should be recommended.

7 PROCEDURE FOR INSTALLATION OF ROCK ANCHORS

7.1 The site shall be made clear and clean where the rock anchors/rockbolts are to be provided that is clay overburden or loose pieces of rock or debris or vegitation should be removed up to a clear rock formation.

7.2 The drill holes should be driven slightly of larger diameter (about 30 mm) than the diameter of steel bars which is to be provided as anchor bars.

7.3 Drill hole should be washed with water till the clear water flows out from it. Then the drill hole should be filled with cement grout. Plain or ribbed steel shall be driven into the drillhole filled with cement grout either with a sledge hammer or a power hammer.

7.4 A base plate of 150 × 150 × 10 mm shall be fitted on the protruding portion of the bolt and tightened with a check nut before insertion in the grout filled hole. Alternatively concrete/shotcrete base slab may be made at the bolt heads.

8 ASSESSMENT OF LOAD BEARING CAPACITY OF ROCK-REINFORCEMENT

8.1 Failure or rock-reinforcement system shall be dependent on the material properties of rock-bolts and the quality of the rock-mass in which it is provided. There are many causes which effect the load bearing capacity of rock-bolts.

8.2 If the rockbolt is subjected to some components of shear, the load bearing capacity of bolt shall be reduced, the reduction is due to type of rockbolt and its angle between bolt and the joint surface. The cables bear the shear displacement more effectively than the solid rockbolts, due to the fact that on the application of shear force, the cable wires can reorient themselves.

8.3 The pull-out test on rock-anchors should be done as specified in IS 11309.

8.4 The values of bond strength (τ_f) between grout and rock or bond strength (τ_a) between grout and steel anchor obtianed from pull-out test should be used for redesigning rock-anchors.

9 DESIGN OF ROCK-REINFORCEMENT

9.1 For the analysis of the stability of slope, partial factors of safety for each of the parameter should be used instead of using the single factor of safety.

9.1.1 Higher factor of safety should be applied to ill-defined parameters such as water pressure and cohesive strength. Low factor of safety should be applied to those quantities (weight of wedge), which is known to a greater degree of precision.

9.1.2 The following partial factors of safety should be used:

F_c = 1.5 (for cohesive strength),
F_ϕ = 1.2 (for frictional strength),
F_w = 1.0 (for weight of wedge),
F_s = 1.5 (for steel and grout anchor), and
F_v = F_u = 2 (for water pressure).

9.2 Design of Rock Anchors

9.2.1 *Static Case*

By considering the condition of limiting equilibrium of the plane wedge along the sliding plane (Fig. 2 without considering a_h.W force), it is found:

Disturbing Force = Resisting force

$$R\,[\mathrm{Cos}\theta\,(\mathrm{Tan}\phi j/F_\phi) + \mathrm{Sin}\theta]$$
$$= [w.\sin\Psi_p.F_w + V.\cos\Psi p F_v - (w.\cos\Psi_p.F_w - V.\sin\Psi_p.Fv - U.F_u) \times (\mathrm{Tan}\,\phi_j/F\phi) - (Cj/F_c)\,(H - Z_c)\,\mathrm{cosec}\,\Psi p] \quad ...(4)$$

where

Ψ_p = dip of discontinuity
Ψ_f = dip of slope face
Z_c = depth of tension crack
= H [1 − cot Ψ_f tan Ψ_f]
≤ H/2
$V = \frac{1}{2}\gamma w\, Z^2$
Z_w = depth of water in tension crack
W = weight of wedge
$= \frac{1}{2}\gamma H^2[\mathrm{Cot}\,\Psi_P - \mathrm{Cot}\,\Psi_f] - \frac{1}{2}\gamma Z_c^2 \mathrm{Cot}\Psi_P$
ϕ_j = Sliding angle of friction of discontinuity
C_j = cohesion along discontinuity
U = Uplift on discontinuity = $\frac{1}{2}\,(H - Z_c)\,\gamma w\, Z w\, \mathrm{cosec}\Psi_P$
γ = Unit weight of rock mass
γw = Unit weight of water

From this expression, the total anchor capacity [R per unit length of slope periphery] may be obtained as all other parameters are known.

In case of a tension crack developed on the slope face, the above equation should be modified accordingly.

9.2.1.1 *Optimum orientation of rockbolts*

To provide the economical rock reinforcement for obtaining the maximum anchor bolt/cable capacity, the anchors should be so provided that the angle ϕ (angle between the bolt and normal to the plane of discontinuity) is made equal to (90° − ϕ_j),

where

ϕ_j = Sliding angle of friction of critical joint plane. The angle between normal to the slope and cable anchor should be kept less than, ϕ_j/1.2 for preventing sliding of base plate along the plane of slope. Generally dip of bars is about 10° so that mortar does not flow out.

9.2.1.2 *Calculation of anchor spacing*

The spacing of the bolts is kept uniform along the slope. The spacing (S) is given by the following expression (*see* Fig. 2b):

$$S = [(H - Z_c)\,\mathrm{cosec}\,\Psi_p\,\cos\theta/\mathrm{Rb}\,\cos\,(\theta + \Psi_P - \Psi_f)]\,P \quad ....(5)$$

where

R = Total anchor force required to stabilize the slope, ≥ 0

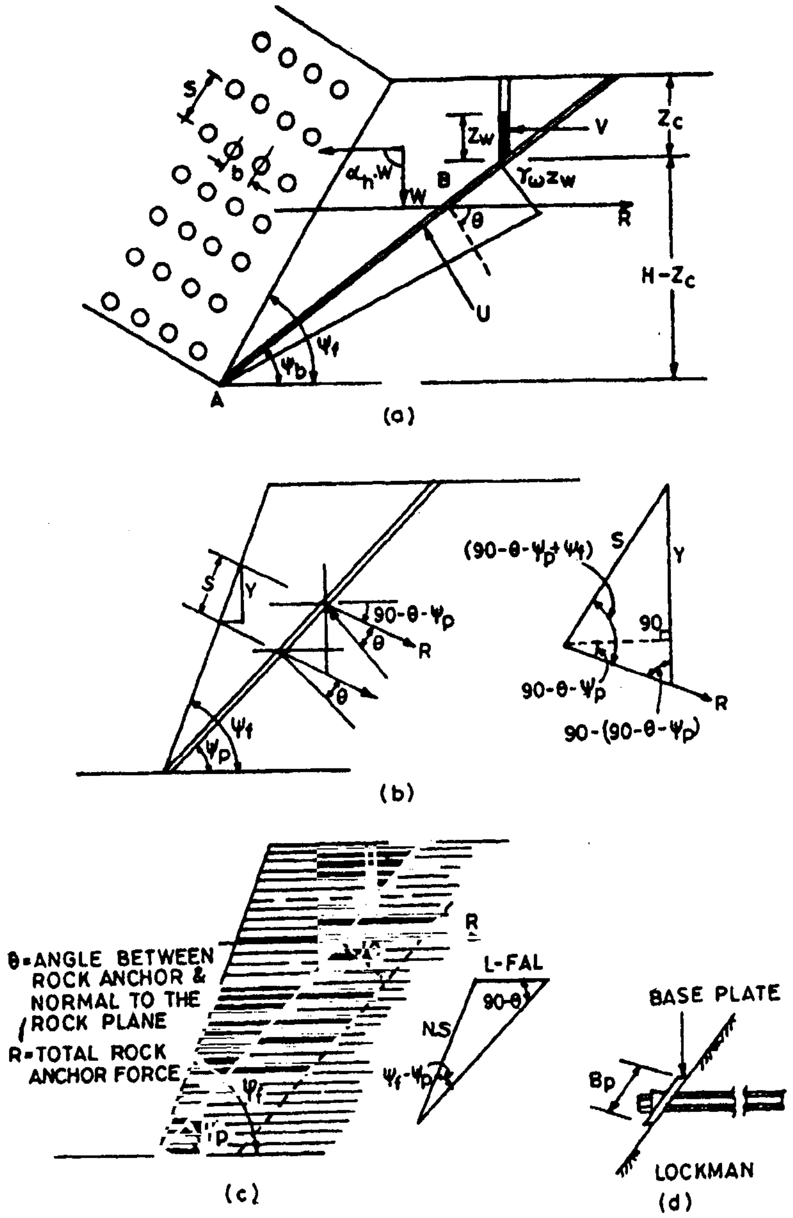

FIG. 2 ROCK REINFORCEMENT (PLANE WEDGE)

[R = Total anchor force and θ = angle between rock anchor and normal to the joint plane]

S = Spacing of bolt along the slope (generally kept 1.5 to 2.5 m depending upon total anchor force R),

b = Spacing of bolt across the slope,

P = Bolt force or safe anchor capacity,

$$P = (\frac{\pi}{4} d_s^2 (\sigma_y/F_s) \quad \text{....(6)}$$

(10 to 60 tonnes)

where

d_s = Diameter of the bolt (25 – 40 mm)

σ_y = Yield strength of the steel bolt as per IS 456

Other terms are already defined earlier. It would be better to stagger anchors.

9.2.1.3 *Number of anchors*

The number of anchors in a column along the dip direction of the slope face up to crack tip is given by the following expression (*see* Fig. 2B):

$$N = (H - Z_c) \sin (\theta + \Psi_p)/Sb \cos \theta + \Psi_p - \Psi_f \quad \text{...(7)}$$

$$= 0 \text{ if } R \leq 0$$

9.2.1.4 *Length of the rockbolts*

Uniform length of rock anchors/bolts is used for ease of construction. The length of the bolt on the slope [*see* Fig. 2(c)] is given by :

$$L = [N S \sin (\Psi_f - \Psi_p)/\cos\theta] + \text{F. A. L.} \quad .....(8)$$

where

F. A. L. = Fixed anchor length obtained from Table 1.

9.2.2 *Design of Base Plate or Base Slab for Anchors*

The size of the base plate or base slab is determined on the basis of bearing capacity of the rock-mass. The size of square base plate/slab is given by [*see* Fig. 2 (d)].

$$Bp = \sqrt{P/Q_a} \quad ...(9)$$

where

P = Anchor capacity or pretension,

Q_a = Allowable bearing pressure of rock-mass,

Bp = Side length of the steel base plate (25-75 cm) or base shotcrete concrete slab (75 - 100 cm)

9.3 Rechecking of the Plane of Weakness (After Design of Rock- Anchors)

The stability of the slope at the end of the rock-anchors (at the extended plane as shown dotted in Fig. 2(c) should also be verified. Alternatively minimum length of anchor (L_r) should be more than 0.2H.

10 DRAINAGE SYSTEM

There is no need of reinforcement if factor (static) of safety of unreinforced slope is more than 1.2 and dynamic factor of safety is also more than 1.0.

$$F_{dyn} = \frac{C_j A + a_j [W\cos\Psi_p - \alpha_h W\sin\Psi_p - V\sin\Psi_p - U]}{W\sin\Psi_p + \alpha_h W\cos\Psi_p + V\cos\Psi_p} \quad ...(10)$$

where

ah = 0 (static case) or 0.10 for blasting,

= coefficient of horizontal component of acceleration due to earthquakes, blasting or machine vibrations or road traffic, and

$A = (H - Z_c)\,\text{cosec}\ \Psi_p$

10.1.1 The term within bracket is normal force across discontinuity and must be more than zero. Otherwise failure will take place by over-toppling of wedge.

10.2 Drainage system should be provided if drained slope ($U = V = 0$) has static factor of safety more than 1.2 and dynamic factor of safety more than 1.0. In the cases of smaller factors of safety, both rock anchors and drainage system shall be provided.

10.3 A drainage system consists of 38 mm dia drain holes dipping 10° towards valley, at spacing of 3 m × 3 m fitted with rolled tube of wire net.

10.3.1 There should be a catch drain at the toe of the cut slope to drain off water.

NOTE — If seepage is occurring even in normal season, it is an indication that slope will give way soon. Drainage is then immediately required.

10.4 Rock reinforcement is needed if static factor safety and dynamic factor of safety are less than 1.2 and 1.0 respectively even after complete drainage.

10.4.1 Unreinforced rock slope or cut may be considered stable if static factor of safety is more than 1.2 and dynamic displacement is less than one percent of height of slope or 1 m whichever is less.

10.4.2 In case of toe cutting, expressions for W, S, L_r and N will change. Basic principles of design will be same.

11 COMPUTER AIDED DESIGN

Computer softwares such as SASP may also be used for the design. Computer programs like WEDGE may be used for the design of rock-reinforcement of slopes with 3 dimensional wedge failures.

ANNEX A

(Foreword)

COMMITEE COMPOSITION

Rock Mechanics Sectional Committee, CED 48

Chairman	*Representing*
PROF BHAWANI SINGH	University of Roorkee, Roorkee
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ASSISTANT RESEARCH OFFICER	Irrigation Department, UP
DR R. L. CHAUHAN	Himachal Pradesh State Electricity Board, Shimla
CHIEF ENGINEER (R & D)	Irrigation Department, Haryana
DIRECTOR (ENGINEER) (*Alternate*)	
SHRI DADESHWAR GANGADHAR DHAYAGUDE	Asia Foundations and Constructions Ltd, Mumbai
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